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No. 78 JUNE 1913

# Bulletin of the American Institute of Mining Engineers.

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TECHNICAL JOURNALS AND OTHERS DESIRING TO REPUBLISH ARTICLES CONTAINED IN THIS *BULLETIN* SHOULD APPLY FOR PERMISSION TO THE EDITOR, AT 29 WEST 39TH STREET, NEW YORK, N. Y.

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# BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.

No. 78 JUNE 1913

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Entered as second class matter, October 16, 1911, at the post office at Philadelphia, Pa., under the Act of March 3, 1879.

**Butte Meeting.**—It is expected that the technical program of the Butte meeting will be a sufficient attraction to draw to that point every member who can possibly get away long enough to attend. Attention is particularly called to the papers already sent in by the Committee on Precious and Base Metals, and the papers which they are undertaking to secure for the meeting, as published on pp. ix and x. In addition to these papers, and in addition to those published in this *Bulletin*, the office of the Institute already has in hand several papers, which will be published at an early date, and presented in abstract and discussed at the meeting.

Of the discussion of these papers a special feature is being made. Therefore, let every member come to Butte who can do so.

A meeting of the Board of Directors of the Institute will also be held at Butte.

In the May *Bulletin*, we stated for the information of members the railway excursion rates for two suggested tours in connection with the meeting, which included Glacier Park before the meeting and Yellowstone after it.

The itineraries there given and rates quoted were planned and offered by the railway agent bureaus in New York. Circulars have been sent to the members of the Institute by A. E. Vaughan & Co., a tourist trip agency of New York City, containing details of tours nearly identical with those outlined in the *Bulletin*, and quoting rates covering the expenses of the whole trip. Members can easily compare these rates with the rates offered by the railroads, and published in the last *Bulletin*.

In order that no misapprehension may exist, the statement is here made that the Institute and its officers have no connection whatever with Vaughan & Co.'s proposals and that any such tours will be strictly personal matters between said company and the members of the Institute who take them. Nevertheless, the Board of Directors calls to the attention of members the circumstance that tours such as these will afford an opportunity for members to travel together to the meeting.

**New York Meeting of the Institute in October.**—Attention is called to the notes of the Iron and Steel Committee on p. viii of this *Bulletin* for particulars of the general meeting of the Institute on Oct. 16 and 17, 1913, under the auspices of the Iron and Steel Committee.

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**Information Desired on Radium.**—In view of the great and increasing importance attached to radium and radium deposits, members are urged to communicate to the Secretary any information they may have as to deposits containing radio-active minerals.

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**Formation of Local Sections.**—In accordance with the By-Laws of the Institute, a local section may be authorized by the Board of Directors at the written request of 10 members residing within an appropriate distance of a central point. A section must consist of at least 25 members, who shall all be either members or associates of the Institute. The widest autonomy is given to local sections as to their government, number of meetings, membership, etc., in so far as is consistent with the Constitution and By-Laws of the Institute.

It is the policy of the Board of Directors to contribute from Institute funds, as far as possible, for the necessary running expenses of the local sections, and to encourage the formation of these sections in every reasonable and appropriate manner.

The Directors greatly desire the co-operation of members to secure the establishment of local sections in all important centers. It is believed that, for the first, the best interests of the Institute will be furthered when local sections have been organized as follows:

| | | |
|---|---|---|
| New York, | St. Louis, | San Francisco, |
| Boston, | Chicago, | Southern California, |
| Philadelphia, | Lake Superior, | Arizona, |
| Pennsylvania Anthracite District, | Spokane, | Salt Lake, |
| Pittsburg, | Puget Sound, | Denver. |

It is hoped that members who are influential in these districts will take the lead in forming new sections and encouraging those already organized. Correspondence with the headquarters of the Institute is invited.

---

**Junior Members.**—The attention of professors of universities and others is called to the provisions of the Constitution in regard to Junior Members: A student may apply for Junior Membership up to the day of his graduation, and if elected, he may be a Junior Member of the Institute for a period of three years upon the payment of $5 per year; if application is delayed until after graduation, the candidate must become a full Member or an Associate, with the payment of $10 initiation fee and $10 per year from the beginning. If these facts are called to the attention of graduating classes in mining schools, it is believed that many students will prefer to apply for membership previous to their graduation.

**Nominations for Officers.**—The Committee appointed by the Board of Directors to nominate officers for the Institute for the year 1914 desires that every member participate in the work by offering suggestions and proposing names for the various offices, thereby obtaining the best ticket that can be recommended.

The officers to be elected at the annual meeting in February, 1914, are:

One officer, known as Director and President.
Two officers, known as Director and Vice-President.
Five officers, known as Director.

The attention of members is called to Articles V and VII of the Constitution, and to By-Law XIII, which reads as follows:

"The Geographical districts to be considered by the Committee on Nominations shall be as follows, until otherwise ordered by the Board.

District No. 1. New England, New York, and New Jersey, excepting New York City and district, which is provided for in the Constitution. [N. B.—New York City and district is designated District 0.]
District No. 2. Pennsylvania.
District No. 3. Ohio, Indiana, Illinois, Iowa, and Missouri.
District No. 4. Minnesota, Wisconsin, and Michigan.
District No. 5. Montana, North and South Dakota, Wyoming, Nebraska, Kansas, Washington, Oregon, Idaho, and Alaska.
District No. 6. California and Nevada.
District No. 7. Utah, Colorado, Arizona, and New Mexico.
District No. 8. Louisiana and Texas.
District No. 9. Other Southern States and District of Columbia.
District No. 10. Mexico.
District No. 11. Canada."

An excerpt from Article VII of the Constitution reads as follows:

"In making such selections [namely, the 8 Directors to be elected], the Nominating Committee shall, so far as practicable, distribute the representation on the Board geographically, so that seven members shall be residents of the district including New York City and the territory within a radius of fifty miles of the headquarters of the Institute, and one member a resident of each of the geographical districts enumerated in the By-Laws."

The officers of the Institute whose terms expire are as follows:
President, Charles F. Rand (not eligible for re-election).
Past President, Charles Kirchhoff.
Vice-Presidents, Karl Eilers, District 0; Waldemar Lindgren, District 1.
Directors, James Douglas, District 0; James Gayley, District 0; Albert R. Ledoux, District 0; Charles W. Merrill, District 6; and C. Snelling Robinson, District 3.

It is hoped that the members of the Institute will help the Committee to the utmost by making suggestions and proposals as requested above, which should be sent to the Chairman of the Nominating Committee, John Hays Hammond, 71 Broadway, New York, N. Y.

It is the intention of some of the members of the Nominating Committee to attend the meeting at Butte. This will enable visiting members to express their views in person.

JOHN HAYS HAMMOND,
*Chairman, Nominating Committee.*

## LOCAL SECTION IN COLORADO.

*Committee of Arrangements.*

VICTOR C. ALDERSON, *Chairman*,
F. H. BOSTWICK, S. A. IONIDES,
E. LENEVE FOSTER, CHARLES J. MOORE.

Within two days of the last meeting of the Board of Directors, application was received from 27 members of the Institute residing in Colorado for authority to establish a Local Section with headquarters at Denver, and authority was accordingly given by the Directors.

---

## LOCAL SECTION AT SAN FRANCISCO.

*Committee of Arrangements.*

S. B. CHRISTY, *Chairman*,
H. FOSTER BAIN, H. C. HOOVER,
EDWARD H. BENJAMIN, WILLIAM C. RALSTON,
F. W. BRADLEY.

On May 28, 1913, formal application was received from 29 members of the Institute residing in and near San Francisco, Cal., for authority to form a Local Section of the Institute.

---

## BOARD OF DIRECTORS.

*Meeting, May* 23, 1913.—On the petition of 25 or more members residing in the different localities, the following Local Sections were established:

St. Louis Local Section.
Southern California Local Section.
Denver Local Section.

It was voted to publish a statement in regard to tours in connection with the Butte meeting, which is published on p. i of this *Bulletin*.

Mr. Edward Caldwell was appointed as expert advisor on printing.

The report of the Committee on Printing advising certain changes in methods of printing was accepted.

The report of the Iron and Steel Committee was accepted, and New York City was adopted as the place of the meeting of the Institute, under the auspices of that Committee, the latter part of October, 1913.

The changes in the By-Laws of the New York Local Section were approved.

*Monthly Meeting of Directors.*—Members of the Board of Directors and Chairmen of Committees are advised that, except in July and August, the regular monthly meeting of the Directors will be held in New York on the fourth Friday of each month. Those residing at a distance from New York are requested to so time their visits as to be able to attend.

## PERSONAL.

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

The following members registered at Institute headquarters during the month of May:

Emil Gathmann, Baltimore, Md.
Albert H. Fay, Washington, D. C.
Alfred E. Barlow, Montreal, Canada.
B. C. Yates, Lead, S. D.
Leonard S. Austin, Salt Lake City.
Henry A. Wentworth, Boston, Mass.
Frank M. Leland, Mackay, Idaho.
W. N. Crafts, Cleveland, Ohio.
Louis Hasbrouck, Kingston, N. Y.
F. V. Desloge, Desloge, Mo.
E. C. Templeton, Palo Alto, Cal.
L. D. Ricketts, Warren, Ariz.
J. E. Johnson, Jr., Ashland, Wis.
A. O. Granger, Cartersville, Ga.
Philip Argall, Denver, Colo.
A. C. Hodge, F.R.G.S., St. Austell, Eng.
S. M. Marshall, Johnstown, Pa.
Miltiades Armas, Paris, France.
Albert Sauveur, Cambridge, Mass.
G. A. Reinhardt, Cambridge, Mass.
Fred Crabtree, Pittsburg, Pa.
T. A. Rickard, London, England.
F. Lynwood Garrison, Philadelphia, Pa.
S. B. Christy, Berkeley, Cal.
H. A. Buehler, Rolla, Mo.
F. W. DeWolf, Urbana, Ill.
Howland Bancroft, Denver, Colo.

Something evidently occurred in the Department of Geology, Columbia University, on a recent date which is described as follows: "**Kwinnum Moon, Sinamokst Sun, Tenas Polaklie.**" The following notice of the event has been sent to us with a special request that it be printed for the benefit of Western members, who are well versed in foreign languages:

Ikt Ehkahnam, Mamook Hee-Hee — Professor Grabau
Mokst Ehkahnam, Oleman Sun kopa China, — Mr. T. T. Read, 02 S
Klone Ehkahnam, Hy-iu Cultus Hee-Hee, — Dr. A. R. Ledoux
Lakit Ehkahnam, Tipso Illahie Saghallie Illahie — Prof. D. W. Johnson
Kwinnum Ehkahnam, Mamook lip-lip chuok, — Dr. E. O. Hovey
Taghum Ehkahnam, Mamook weght kloshe Columbia University, — Professor Kemp.

**E. V. Daveler** has been appointed Superintendent of Mills for the Alaska Gastineau Mining Co., Juneau, Alaska.

**Louis D. Huntoon,** Chairman of the New York Section of the Institute, has become Eastern representative of the Stearns-Rogers Mfg. Co., at their New York office. Prof. Huntoon will continue also his professional consulting work.

**W. L. Saunders** has resigned as President of the Ingersoll-Rand Co., and become Chairman of its Board of Directors.

**Robert N. Norton** has become Chemist and Engineer of Tests of the Penn Steel Castings & Machine Co., Chester, Pa.

**Prof. William A. Carlyle,** until recently in charge of the metallurgical department of the Royal School of Mines, London, has resigned this position and become a consulting engineer, with offices in London.

**Arthur B. Foote** has become General Superintendent of the North Star Mines Co.

**George W. Evans** has gone to Alaska to study the commercial possibilities of the Matanuska coal field for the U. S. Bureau of Mines.

**Andrew W. Newberry** is now employed at Copper Cliff, Ont., by the Canadian Copper Co.

**Arthur W. Geiger** is now Manager of the California Ore Testing Co.

**Dr. John C. Branner,** who is at present in Brazil on professional geological work for the Brazilian government, has been made President of Stanford University.

**Robert E. Cranston** has been made Chief Engineer for Breitung & Co., Ltd., in charge of mine exploration.

**Knox Taylor**, President of the Taylor-Wharton Iron & Steel Co., has been elected President also of the Tioga Steel & Iron Co.

**Charles E. Schwarz** has been made Superintendent of the North American Smelting Co., Perth Road, Ont.

**W. L. Walker** has become Chief Geologist for the Associated Oil Co., in place of **W. A. Williams**, resigned, who has joined the engineering staff of the General Petroleum Co.

**James J. Martin** has been made General Manager of the Chicksan Mining Co., Chicksan, Korea.

## POSITIONS VACANT.

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons.)

The position of converter blower at a side-blow Bessemer steel casting plant is vacant.

The State Legislature of Utah has provided for the establishment of a metallurgical research department at the State School of Mines, Salt Lake City. Applications will be received for the position of head of this laboratory, and further details may be obtained by applying to Prof. Joseph F. Merrill, Director, University of Utah, Salt Lake City.

An engineering company owning patents for improvements in steel manufacture which have been adopted at many works in America, desires to secure the services of a technical iron and steel metallurgist as assistant to its chief metallurgical engineer.

## ENGINEERS AVAILABLE.

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons.)

An experienced mine superintendent, manager, or field engineer, is open for engagement. He has had 12 years' experience as superintendent and manager of large mines in the United States and Mexico, and experience in mine examination.

A member of the Institute with several years' experience in mining in the Lake Superior copper district, is open for engagement.

A member, at present in the Southeast, desires a position as assistant superintendent or assistant manager with a mining company operating in Washington or Alaska. Experienced assayer and engineer.

An expert mining and mechanical engineer of 40 years' experience is open for engagement.

A technical graduate, 33 years of age, with 11 years' mining experience as engineer, foreman, superintendent and manager of development work, would like to make a change.

A technical graduate of five years' experience underground is open for an engagement.

An engineer who has had 15 years' experience in exploration and development work for a large corporation will soon be available for a similar position, preferably to operate in the tropics. Speaks German, Spanish, and the Scandinavian languages.

## SUBSCRIPTIONS TO THE LAND FUND.

NEW YORK, N. Y., June 4, 1913.

TO THE PRESIDENT,
American Institute of Mining Engineers.

Dear Sir:

I have pleasure in making the following statement concerning subscriptions to the Land Fund of the American Institute of Mining Engineers up to June 1, 1913:

| | |
|---|---|
| Amount of Land Debt unsubscribed Aug. 2, 1912, | $68,000.00 |
| Amount subscribed in response to requests by the undersigned, | 68,000.00 |
| Amount so far collected, | 48,405.22 |
| Amount at present uncollected, | 19,594.78 |

Of this uncollected balance:

$17,639.75 is due by men who subscribed $1,000 or more.
$ 1,0:0.00 is due by men who subscribed $100 to $1,000.
$ 945.03 is due by men who subscribed less than $100.

In order that the full amount might be reported at the Annual Meeting of the Institute as having been subscribed, certain members who had already contributed liberally subscribed the balance not then subscribed for. Since that time a number of subscriptions have been received and others are expected, so that I hope that the guarantors will not have to be called upon for their additional donations.

Now that the dues will not have to be raised in order to relieve the Institute of debt, I hope that many of the members who hesitated to subscribe to the fund while the result was still doubtful will contribute their share.

Yours very truly,

JAMES DOUGLAS.

---

## THE INSTITUTE FORUM.

CAMBRIDGE, MASS., May 22, 1913.

TO THE PRESIDENT,
American Institute of Mining Engineers.

Dear Sir:

Referring to Mr. J. W. Finch's letter printed in the Institute Forum of the May *Bulletin*, I am very much in sympathy with his suggestion that the *Transactions* be printed in several volumes devoted separately to the various subjects covered by the Institute papers, such as "Iron and Steel," "Mining Geology," "Precious and Base Metals," etc. The advantages of this plan appear to me so great and so obvious as to far outweigh any possible objections. At a meeting of the Iron and Steel Division Committee held on June 28, 1912, I moved that the Committee recommend to the Council that papers on Iron and Steel be published in separate annual volumes. The motion was seconded and passed. Since then I have never failed to repeat my request whenever an occasion presented itself, and I welcome the opportunity offered by Mr. Finch's communication to again call the attention of those interested to an innovation which appears to me so highly desirable.

ALBERT SAUVEUR.

## IRON AND STEEL COMMITTEE.

CHARLES KIRCHHOFF, *Chairman.*
ALBERT SAUVEUR, *Vice-Chairman.*
HERBERT M. BOYLSTON, *Secretary*, Abbot Bldg., Cambridge, Mass.

| | | |
|---|---|---|
| John Birkinbine, | J. Esrey Johnson, Jr., | Felix A. Vogel, |
| William H. Blauvelt, | William Kelly, | Leonard Waldo, |
| James Gayley, | Richard Moldenke, | William R. Walker, |
| Henry D. Hibbard, | Joseph W. Richards, | William R. Webster. |
| Henry M. Howe, | E. Gybbon Spilsbury, | Frederick W. Wood. |
| Robert W. Hunt, | A. A. Stevenson, | |

The Board of Directors of the Institute has authorized the holding of a meeting under the auspices of the Iron and Steel Committee in New York City, October 16 and 17, 1913.

Gratifying responses have been received to the request of the Committee for contributions, the following papers being already assured:

1. H. M. Howe, Equilibrium Temperature for $A_1$ in Carbon Steel.
2. H. M. Howe, The Divorcing of the Eutectoid in Meteorites.
3. H. M. Howe and A. G. Levy, Determination of the Position of $Ae_3$ in Carbon-Iron Alloys.
4. G. K. Burgess, J. J. Crowe and H. S. Rawdon, Thermal and Microscopical Examination of Professor Howe's Standard Commercial Steels.
5. H. M. Howe, Discussion of the Existing Data as to the Position of $Ae_3$.
6. J. H. Hall, Shock Tests of Cast Steel.
7. H. F. Miller, Jr., The Design of Regenerators for Open-Hearth Furnaces.
8. E. Stütz, Paper on the "Scoria Process."
9. Albert Sauveur, Mayari Steel.
10. Felix A. Vogel, Briquetting.
11. J. E. Johnson, Jr., Paper on "Cast Iron."

As far as possible these papers will be published in the *Bulletin before* the October meeting and will also be mailed in pamphlet form to any member requesting it, as well as to all persons designated by the authors.

The Committee has good reasons to expect additional papers by eminent steel manufacturers. The assistance of all members of the Institute interested in iron and steel is urgently requested. Suggestions with a view to making the October meeting a success will be gratefully received and carefully considered. Members intending to present papers should notify the Secretary as early as possible, giving the title of the paper and the probable date at which the manuscript may be expected. It is essential that the manuscripts be received at the latest by Sept. 1, and preferably by Aug. 1, in order that the paper may be printed and distributed before the meeting. If an author will not be able to get ready a paper within the time specified, it is hoped, however, that this will not deter him from forwarding the manuscript at the earliest possible date.

ALBERT SAUVEUR, *Vice-Chairman.*

## COMMITTEE ON PRECIOUS AND BASE METALS.

CHARLES W. GOODALE, *Chairman.*

L. D. RICKETTS, *Vice-Chairman.*

DARSIE C. BARD, *Secretary.*

Montana State School of Mines, Butte, Mont.

| | | |
|---|---|---|
| Leonard S. Austin, | Samuel S. Fowler, | Willet G. Miller, |
| David W. Brunton, | Thomas J. Grier, | Albert F. Schneider, |
| Theodore B. Comstock, | Hennen Jennings, | George C. Stone, |
| Stanley A. Easton, | Edmund B. Kirby, | Benjamin B. Thayer, |
| | Charles W. Merrill. | |

The Committee desires to be notified of all papers coming under its supervision which may be in preparation or contemplated for publication in the Institute *Transactions*. If this is done promptly needless duplication of subjects can be avoided. This notification is not to supersede the duties of the Committee on Publications, but merely for the purpose of keeping the Precious and Base Metals Committee informed of papers in preparation so that new material may be sought intelligently.

The following is a provisional list of papers secured by the Committee for the August meeting in Butte. Titles are not final and authorship may be changed in some cases.

### METALLURGICAL SUBJECTS.

#### *Smelting.*

Great Falls Converter Practice. By A. E. Wheeler.
Development of the Basic-lined Converter for Copper Mattes. By E. P. Mathewson.
Great Falls Flues and Dust-Chamber System. By C. W. Goodale and J. H. Klepinger.
Washoe Reduction Works Flues and Dust-Chamber System. By E. M. Dunn.
Reverberatory Smelting with Low-grade Coal. By C. D. Demond.
Notes on Electrolytic Refining of Copper. By Willis T. Burns.
The Tooele Plant of the International Smelting & Refining Co. By H. N. Thompson.
Recent American Advances in the Gold and Silver Assay of Copper Materials. By Edward Keller.
Distribution of the Precious Metals in Converter Copper.

#### *Milling, Etc.*

New Methods of Concentrating Butte Ores. By Albert E. Wiggin.
The Morning Mill. By Rush J. White.
Mining and Concentrating Methods in the Cœur d'Alêne District.
Circular Concentrating Tables. By Arthur Crowfoot.
Precipitating Copper from Mine Water in Butte. By J. C. Febles.
The Leaching of Butte Chrysocolla Ores. By C. Lorimer Colburn.
Baffle Plates in Settling Tanks. By C. D. Demond.
The Leaching of Copper Tailings. By Frederick Laist.
The Dorr Improvements in Cyanide Practice. By John V. N. Dorr.
Zinc Milling at Butte. By J. C. Pyle.
The McDougal Furnace. By Frank R. Corwin and W. B. Rogers.
Mining and Beneficiation as Practiced at the Butte & Superior Copper Co.'s Plant. By J. L. Bruce.
Great Falls Blast-Furnace Practice. By John A. Church, Jr.
Experiments in Zinc Milling and Flotation.

#### *Mining Subjects.*

Hydro-Electric Development in Montana. By Max Hebgen.
Use of Electricity in Butte Mining. By John Gillie.
The Zinc Mines of Butte. By James L. Bruce.
Gold Dredging at Ruby, Montana.
Timbering Methods at Butte.

Mill and Mine Accounting. By H. T. Van Ells. Presented by C. W. Goodale.
Mine Ventilation at Butte.
The Drumlummon Mine, Marysville District, Montana. By C. W. Goodale.
The Empire Mine, Marysville, Montana. By F. L. Sizer.
The Belmont Mine, Marysville, Montana.
The Penobscot Mine, Marysville, Montana,
The Bald Butte Mines, Marysville, Montana.
Controlling Mine Fires in the Butte Mines.
Stope Record at Butte.
Shaft Sinking Methods at Butte.
Stoping Methods at Butte.
Electrically Generated Compressed Air for Mine Hoists. By Bruno Nordberg.

*Geological Subjects.*

Geology of Butte. By R. H. Sales.
Geology Applied to Mining.
Mineral Associations at Butte, Montana. By D. C. Bard and M. H. Gidel.
Geologic Mapping at Butte. By F. A. Linforth.
The Southern Cross Mine. By Paul Billingsley.
Some Phases of Mining Geology as Developed by Recent Litigation in the Cœur d'Alêne Mining District. By F. T. Greene.
The Gillmore District, Idaho.
The Kendall District, Montana.

*General Subjects.*

The Cement Industry of Montana.
The Sapphire Mines at Yogo, Montana.
The Future of the Rock Phosphate Industry in Western America.
The Clay-Working Industry in Montana.
Notes on Chemical Analysis of Butte Ores and Products.
The Cement Industry in Montana. By W. H. Andrews.
Metaline Falls Cement Plant. By Milo W. Krejci.

The Committee is particularly desirous of securing at this time a representative collection of papers on the mining industry of Montana, and desires to hear from any member prepared to write on some phase of this subject. All such papers may not be prepared in time for presentation at the August meeting of the Institute, but are nevertheless desired for this year, in the hope that the next volume of the *Transactions* may be devoted in large part or entire to Montana subjects.

CHARLES W. GOODALE, *Chairman.*
DARSIE C. BARD, *Secretary.*

## NEW YORK LOCAL SECTION.

*Executive Committee.*

LOUIS D. HUNTOON, *Chairman.*
ARTHUR S. DWIGHT, *Vice-Chairman.*
E. MALTBY SHIPP, *Treasurer.*
GEORGE F. KUNZ,
THOMAS T. READ, *Secretary.*
Woolworth Building, New York, N. Y.

*Meeting, May* 2, 1913.

The Annual Meeting of the New York Section was held at the Engineering Societies Building, Friday evening. May 2, 1913, at 8 p.m. Dr. Kunz presided. The Annual Report of the Secretary-Treasurer was read and accepted. Mr. Colvocoresses offered the following amendments to the By-Laws:

ARTICLE II: First paragraph amended to read as follows:

"The officers of this Section shall be a Chairman, Vice-Chairman, Secretary, and Treasurer."

In the second paragraph the word "four" to be changed to "five."

ARTICLE V: The following paragraph to be inserted:

"The funds of this Section shall be received by the Treasurer, and shall be paid out only by check signed by the Treasurer, and in case of his absence or inability, the Chairman, Vice-Chairman, or the Secretary may act for the Treasurer."

On motion duly seconded the amendments were adopted as read.

The Nominating Committee, appointed by the Chairman, recommended the following men for officers: *Chairman*, Louis D. Huntoon; *Vice-Chairman*, Arthur S. Dwight; *Secretary*, Thomas T. Read, and *Treasurer*, E. Maltby Shipp. There being no other nominations, Mr. Rand moved that the Secretary be authorized to cast the vote for the members in favor of the men nominated. Motion carried. In reply to the circulars mailed regarding the summer outing the Secretary reported that 89 members approved and 9 did not approve of such an outing.

The Chairman introduced **Dr. M. Belowsky**, of the University of Berlin, who addressed the meeting in German. Following Dr. Belowsky, the speaker of the evening, **A. H. Fay**, of the Bureau of Mines, discussed **Metal Mine Accidents.**

**Mr. Healy**, one of the engineers in charge of the Catskill Aqueduct work, presented a paper giving a comparison between the accidents in metal mines and those in the Catskill work.

**T. B. Stearns** then explained the operation of the McKesson-Rice screenless sizer.

*Report of the Secretary-Treasurer.*

*Annual Meeting of the Institute.*

Receipts.

| | |
|---|---|
| Contributions from Members of the New York Section, | $820.00 |
| Receipts from Banquet, | 705.00 |
| | $1,525.00 |
| Disbursements, | 1,477.99 |
| Balance, | $47.01 |

*Financial Statement of the New York Section.*

Receipts.

| | |
|---|---|
| Total, 132 Members @ $6.70, | $885.00 |
| To Balance from Annual Meeting, | 47.01 |
| To Advance from Institute, | 200.00 |
| Total, | $1,190.37 |
| Expenses, | 998.94 |
| Balance, | $191.43 |

From this balance of $191.43 must be deducted the expenses of this meeting, which will amount to approximately $135.

LOUIS D. HUNTOON, *Secretary-Treasurer*,
115 Broadway, New York, N. Y.

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## PUGET SOUND LOCAL SECTION.

*Executive Committee.*

CHESTER F. LEE, *Chairman.* C. R. CLAGHORN, *Vice-Chairman.*
W. C. BUTLER. M. K. RODGERS.
JOSEPH DANIELS, *Secretary-Treasurer*,
University of Washington, Seattle, Wash.

The Puget Sound Local Section closed its first year of activity with a dinner-meeting at the College Club on Saturday, May 24. The meeting expressed itself in favor of a committee of the Institute on mining methods, and in favor of having all members resident in territory covered by a local section *ipso facto* members of that section.

The Alaska Committee of the Seattle Chamber of Commerce and the local branch of the American Mining Congress propose to feature the mining industry in the coming Potlatch, an annual festival held in Seattle in July, and to hold a smoker for mining men and give space in the parades to floats representing mining. Our co-operation was asked in handling this matter, and a committee representing the Puget Sound Section was appointed to assist.

Harvey L. Glenn, of the U. S. Assay Office in Seattle, spoke on Mint Methods. He explained the systems employed in receiving, weighing, melting, assaying, and paying for the gold bullion brought to the various assay offices maintained by the government. The Seattle office handles on an average $14,000,000 in bullion each year, and since 1898, the date of establishment, more than $200,000,000 have passed through this office. This amount comes mainly from Alaska, but a certain share of the production is diverted to San Francisco and to the British Columbia banks and assay office, where a lower assay charge is made. At the present time, the discrimination against the Seattle office in favor of San Francisco is $1.25 on $1,000, or one-eighth of 1 per cent. The transportation charge to San Francisco from Seattle is 75 cents per $1,000, leaving a net charge of 50 cents per $1,000 against any bullion coming into Seattle. This charge of 50 cents per $1,000 is being met by the Seattle Chamber of Commerce, which reimburses the Alaskan depositors of bullion in order that the money may remain in Seattle and not be sent to San Francisco. Seattle is very much wrought up over the possibility of closing the assay office, as well as the charge referred to. The past growth and the present prosperity of the city have come with the development of Alaska, and Seattle feels

that anything which will tend to divert Alaskan gold from her port is a blow at her prosperity.

Another phase of Seattle's relations to Alaska was brought out in an illustrated paper by V. V. Clark on The Clark System of Placer Mining. Mr. Clark is the principal designer and inventor of a new gold-saving machine developed as a result of several years' experience in Alaska placer ground having limited water supply. This gold saver is the nucleus of five machines, ranging from a prospector's shoveling-in machine to a dredge. Briefly, the gold saver is a sluice box closed at one end, the receiving end, and open at the discharge end, which is higher than the receiving end, thus permitting the maintenance of a constant water supply and level within the box. The placer sand and gravel is moved forward by a reciprocating conveyor line of novel construction designed to prevent obstructions stopping the machine and to automatically, on the return stroke, raise the flight over the gravel in the box, thus permitting a thorough agitation of the gravel and settling of gold values. The principle on which the machine is based was thoroughly tested out by experimental work for three months, and the mechanical details have been simplified to a high degree.

At the present time a hydraulic machine, embodying the principle of the gold saver, is being built in Seattle for an Alaskan mining company. This machine is designed to handle 100 yards of material per hour, taking the gravel furnished by three 3-in. nozzles operating under 200 ft. head. It is made of steel, weighs 21 tons, is 86 ft. long, with a gold saver 34 in. wide, is supported on skids, and includes a boom for handling large boulders and for moving the machine. A 25-h-p. motor will be used for the reciprocating conveyor. The machine will be shipped to Alaska within a few weeks and its operation will be watched with considerable interest by local mining men.

A summary of the work of the year is here given. The first called meeting for organization was on April 20, 1912. Since then, seven meetings have been held—all but one in Seattle. One of the meetings was held at the University of Washington under the auspices of the College of Mines; another was held at the Tacoma Smelter in company with the American Chemical Society and the Engineers' Club of Seattle. Five papers by members, and three papers by non-members have been read. The average attendance has been 14, or about 50 per cent. of the membership near enough to Seattle to attend the meetings without excessive trouble. The expenses of the section have been met by contributions from the local members.

JOSEPH DANIELS, *Secretary.*

## COMMITTEE ON INCREASE OF MEMBERSHIP.

C. R. CORNING, *Chairman.*
ADOLPHE E. BORIE, *First Vice-Chairman.*
THOMAS T. READ, *Secretary*, Woolworth Bldg., New York, N. Y.

*Vice-Chairmen.*

JOHN H. ALLEN,
RICHARD M. ATWATER, JR.,
GEORGE D. BARRON,
A. CHESTER BEATTY,
J. PARKE CHANNING,
GEORGE M. COLVOCORESSES,
ROBERT PEELE,
CHARLES P. PERIN,
JOSEPH A. VAN MATER,
ARTHUR L. WALKER.

D. C. Bard,
W. de L. Benedict,
John C. Branner,
Palmer Carter,
Allen J. Clark,
F. Crabtree,
George G. Crawford,
O. C. Davidson,
E. V. D'Invilliers,
James S. Douglas,
Walter Douglas,
Howard N. Eavenson,
Howard Eckfeldt,
R. C. Gemmell,
F. Louis Grammer,
Ernest A. Hersam,
Edwin C. Holden,
William L. Honnold,
Reginald E. Hore,
C. Colcock Jones,
Eugene P. Kennedy,
Chester F. Lee,
Richard S. McCaffery,
James F. McClelland,
Milton H. McLean,
Philip N. Moore,
T. H. O'Brien,
James J. Ormsbee,
Edward W. Parker,
John B. Porter,
R. M. Raymond,
Robert H. Richards,
LeRoy Salsich,
Frederick H. Sexton,
Henry L. Smyth,
F. W. Traphagen,
Elton W. Walker.

Meetings of this committee were held at the Institute headquarters on April 18, and May 2 and 15.

It was resolved that meetings shall be held monthly on the third Friday of each month.

It was ordered that personal letters be sent by the Chairman of the Committee to the Deans of the principal mining schools in this country and abroad, requesting their co-operation in urging the desirability of membership in the Institute for recent graduates, and offering to furnish as many copies of the *Year Book* and recent *Bulletins* of the Institute as may be desired. The individual members of the Committee were urged to send out as many letters as possible to their friends and acquaintances. After discussion as to the proper field of activity of the Committee, it was decided not to urge the sending in of applications except by those who are clearly eligible to membership so far as can be known.

At the meeting of April 18 communications from absent members of the Committee and from professors in mining colleges were read for the information of those present. It was resolved that the roster of the Engineers' Club be canvassed and all members of that Club engaged in the mining profession, but not already members of the Institute, be urged to join. A. Chester Beatty volunteered to prepare a list of members of the Institution of Mining and Metallurgy who might be invited to join the Institute, and a similar list of men engaged in mining in South Africa who would be desirable members. It was ordered that membership lists of technical societies, at home and abroad, be procured, similarly checked over and submitted to the Committee before the letters are sent out. The Secretary of the Institute pointed out that the cost of sending such letters will be borne by the Institute, and also offered use of the Institute office force in writing and mailing all such letters as were incident to the work of the Committee or of members, and also to supply such separate papers or back numbers of the *Bulletin* as might be required. Robert Peele presented a list of mining schools to which communications should be addressed, and the Chairman requested all members of the Committee, or others, to send him the names of societies, schools and other organizations likely to be interested in the work of the Institute.

THOMAS T. READ, *Secretary.*

## EMMONS VOLUME ON ORE-DEPOSITS.

*Ore-Deposits—a continuation of the "Posepny" Volume.*

Comprising papers descriptive of ore-deposits and discussions of their origin, edited, with an introduction, by Dr. S. F. Emmons. The volume contains also a Biographical Notice of Dr. Emmons by his associate and friend, Dr. George F. Becker, and a comprehensive Bibliographical Index of the Science of Ore-Deposits, prepared by Prof. John D. Irving, of the Sheffield Scientific School of Yale University. Dr. Emmons had finished his editorial work and written his Introduction before his lamented death in 1910; and the Volume contains his last words upon the subject to which he had given the work of his life, and on which he was justly regarded as the foremost authority.

*Contents.*

Genesis of Certain Ore-Deposits. By S. F. EMMONS.
Structural Relations of Ore-Deposits. By S. F. EMMONS.
Geological Distribution of the Useful Metals in the United States. By S. F. EMMONS. Discussion, by JOHN A. CHURCH, ARTHUR WINSLOW, S. F. EMMONS, and WILLIAM HAMILTON MERRITT.
Torsional Theory of Joints By GEORGE F. BECKER. Discussion, by H. M. HOWE, R. W. RAYMOND, C. R. BOYD, and GEORGE F. BECKER.
Allotropism of Gold. By HENRY LOUIS.
Superficial Alteration of Ore-Deposits. By R. A. F. PENROSE, JR.
Some Mines of Rosita and Silver Cliff, Colorado. By S. F. EMMONS.
Genesis of Certain Auriferous Lodes. By JOHN R. DON. Discussion, by JOSEPH LE CONTE, S. F. EMMONS. G. F. BECKER, ARTHUR WINSLOW, W. P. BLAKE, and J. R. DON.
Influence of Country-Rock on Mineral Veins. By WALTER HARVEY WEED.
Igneous Rocks and Circulating Waters as Factors in Ore-Deposition. By J. F. KEMP.
Consideration of Igneous Rocks and Their Segregation or Differentiation as Related to the Occurrence of Ores. By J. E. SPURR. Discussion, by A. N. WINCHELL.
Chemistry of Ore-Deposition. By WALTER P. JENNEY. Discussion, by JOHN A. CHURCH.
Ore-Deposits near Igneous Contacts. By WALTER HARVEY WEED. Discussion, By W. L. AUSTIN.
Ore-Deposition and Vein-Enrichment by Ascending Hot Waters. By WALTER HARVEY WEED.
Basaltic Zones as Guides to Ore-Deposits in the Cripple Creek District, Colorado. By E. A. STEVENS.
Geological Features of the Gold-Production of North America. By W. LINDGREN. Discussion, by W. G. MILLER and W. L. AUSTIN.
Osmosis as a Factor in Ore-Formation. By HALBERT POWERS GILLETTE.
Ore-Deposits of Sudbury, Ontario. By CHARLES W. DICKSON.
Genesis of the Copper-Deposits of Clifton-Morenci, Arizona. By W. LINDGREN.
Copper-Deposits at San Jose, Tamaulipas, Mexico. By J. F. KEMP.
Magmatic Origin of Vein-Forming Waters in Southeastern Alaska. By A. C. SPENCER.
Genetic Relations of the Western Nevada Ores. By J. E. SPURR.
Are the Quartz Veins of Silver Peak, Nevada, the Result of Magmatic Segregation? By J. B. HASTINGS.
Occurrence of Stibnite at Steamboat Springs, Nevada. By W. LINDGREN.
Summary of Lake Superior Geology with Special Reference to Recent Studies of the Iron-Bearing Series. By C. K. LEITH.
Geological Relations of the Scandinavian Iron-Ores. By H. SJÖGREN.
Formation and Enrichment of Ore-Bearing Veins. (With Supplementary Paper.) By GEORGE J. BANCROFT.
Distribution of the Elements in Igneous Rocks. By H. S. WASHINGTON.
Agency of Manganese in the Superficial Alteration and Secondary Enrichment of Gold-Deposits of the United States. By W. H. EMMONS.
Cognate Papers.
Bibliography of the Science of Ore-Deposits. By J. D. IRVING, H. D. SMITH, and H. G. FERGUSON.

The volume will contain about 1,000 pages, and it is expected to be off the press about June 25. Price, bound in cloth, $5; in half morocco, $6. Both the Emmons and the Posepny Volumes on Ore-Deposits, bound in cloth, $8; in half morocco, $10.

## INTERNATIONAL GEOLOGICAL CONGRESS.

(For further particulars apply to the Secretary, 12th International Geological Congress, Victoria Memorial Museum, Ottawa, Canada; cable address, Geocong, Ottawa.)

*Applications to Join Excursions.*

Applications to join excursions are being received at a much more rapid rate than was anticipated; therefore, those desiring to join excursions should make no further delay in sending their applications.

*Papers and Proposals.*

Every endeavor will be made to print in advance of the meeting such contributions as are received up to June 1. Though every effort will be made to print contributions received after this date, it is not probable that this will be possible.

*Headquarters During Session at Toronto.*

The headquarters will be at the University of Toronto, where there will be a branch post-office open at all hours and at which registered mail may be received or despatched, money orders sent or cashed, etc. A bank will be established at which money may be exchanged. A typewriting service will be maintained, also a telephone service and messenger service. Railway and steamship agents will also be in attendance.

A restaurant service providing luncheon in the middle of the day will be arranged.

*Changes and Additions to Excursions.*

Several changes and additions have been made in the excursions, the particulars of which may be learned by application to the Secretary.

*Special Dates.*

Sunday, July 13.....Excursion A. 1.....Quebec and Maritime Provinces, leaves Montreal.
Wednesday, July 23....Excursion A. 3.....Sudbury-Cobalt-Porcupine (Ontario), leaves Montreal and Toronto.
Thursday, July 24.....Excursion A. 2.....Haliburton-Bancroft (Ontario), leaves Montreal.
Friday, August 1.....Special day at Ottawa.
Saturday, August 2.....Special day at Montreal.
Thursday, August 7.....Opening day of Session at Toronto.
Thursday, August 14...Last day of Session at Toronto. Excursions C. 1 and C. 2 start.
Tuesday, August 26.....Special day at Victoria, British Columbia.

*Railway Privileges.*

To professional members of the Congress and to dependent members of their families a special form of certificate will be given. The possession of this certificate will enable the person to whom it is issued to purchase and use during a limited period of time railway tickets at a much reduced cost, to and from any point in Canada.

Special railway rates are also provided for members attending the Congress from certain points on the Pacific coast of the United States.

## INTERNATIONAL ENGINEERING CONGRESS.

The International Engineering Congress of 1915 will be held under the auspices of the American Society of Civil Engineers, American Society of Mechanical Engineers, American Institute of Electrical Engineers, Society of Naval Architects and Marine Engineers, and the American Institute of Mining Engineers, in connection with the Panama-Pacific Exposition. Arrangements for this Congress are now proceeding under the General Committee, the Institute representatives on which are given on p. xliii. An interesting technical program is promised. The time of the Congress is Sept. 20 to 25, 1915.

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## LORD KELVIN MEMORIAL WINDOW.

The memorial window in Westminster Abbey erected to the memory of the late lord Kelvin, as a tribute to his contributions to engineering and science, will be dedicated on July 15, 1913.

As noted on page v of the April *Bulletin*, the Institute's Committee received from 94 members subscriptions amounting to $507.37, which sum was paid to the Honorary Treasurer in London.

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## BACK VOLUMES OF THE TRANSACTIONS.

The Board of Directors has authorized the following offers of sets of back volumes of the *Transactions*, at considerably reduced prices, to Members, Libraries, and Scientific Societies:

| | Per Set. |
|---|---|
| I. Five volumes, bound in half-morocco, from No. 36 (1906) to No. 40 (1910), | $20 |
| II. Ten volumes, bound in half-morocco, from No. 31 (1902) to No. 40 (1910), including Mexican Volume, | 35 |
| III. Twenty volumes, bound in half-morocco, from No. 21 (1893) to No. 40 (1910), | 50 |
| IV. Thirty volumes, bound in half-morocco, from No. 11 (1883) to No. 40 (1910), | 60 |
| V. Thirty-nine volumes, bound in half-morocco, from No. 1 (1873) to No. 40 (1910), with the exception of No. 10 (1882), but including index for Volumes Nos. 1 to 35, and Nos. 36 to 40, | 75 |
| VI. Nine volumes, bound in half-morocco, from No. 1 (1873) to No. 9 (1881), | 25 |

## AFFILIATED STUDENT SOCIETIES.

Any society of undergraduates at a technical school, comprising students in any branch of engineering, metallurgy, chemistry, geology, etc., may be recognized by the Board of Directors in its discretion as an Affiliated Student Society. A circular giving details of the plan of affiliation may be obtained on application to the office of the Secretary of the Institute.

The following societies have been placed by authority of the Board of Directors on the above list:

### AFFILIATED STUDENT SOCIETIES.

The Mining Society of the Sheffield Scientific School, Yale University, New Haven, Conn. *President*, Francis S. Winslow; *Secretary*, Roger W. Newberry.

The University of Illinois Student Branch of the American Institute of Mining Engineers, Champaign, Ill. *President*, M. L. Nebel; *Secretary*, Willis Leriche.

The Engineering Society of the University of Nevada, Reno, Nev. *President*, P. E. Raymond; *Secretary*, R. M. Seaton.

The University of Wisconsin Mining Club, Madison, Wis. *President*, Rudolph J. Stengl; *Secretary*, Mack C. Lake.

The Mining and Geological Society of Lehigh University, South Bethlehem, Pa. *President*, Robert C. Mickel; *Secretary*, Edward B. Snyder.

The School of Mines Society of the University of Minnesota, Minneapolis, Minn. *President*, C. A. Walker; *Secretary*, A. C. Bierman.

The Mining Engineering Society of the Massachusetts Institute of Technology. *President*, Ralph E. Wells, Jr.; *Secretary*, Louis W. Currier.

The Student Auxiliary Society of the American Institute of Mining Engineers of the University of Kansas, Lawrence, Kan. *President*, Walter E. Rohrer; *Secretary*, H. R. Brown.

The Associated Miners of the University of Idaho, Moscow, Idaho. *President*, Hallard W. Foester; *Secretary*, Walter P. Scott.

The State College of Washington Mining and Geological Society, Pullman, Wash. *President*, J. V. Quigley; *Secretary*, S. A. Swanson.

The Tejas Technical Society, School of Mines, University of Texas, Austin, Tex. *President*, G. C. Cartwright; *Secretary*, David S. Alley.

The Ohio State University Student Branch of the American Institute of Mining Engineers, Columbus, Ohio. *President*, Hugh B. Lee; *Secretary*, E. P. Elliott.

The Stanford Geology and Mining Society, Stanford University, Cal. *President*, B. E. Parsons; *Secretary*, E. D. Nolan.

The Senior Mining Society of Columbia University, New York, N. Y. *President*, Roger L. Strobel; *Secretary*, Clark G. Mitchell.

Mining Association of the University of California, Berkeley, Cal. *President*, D. M. Drumheller, Jr.; *Secretary*, J. B. Orynski.

Tufts College Chemical Society, Tufts College, Mass. *President*, P. G. Savage; *Secretary*, W. S. Frost.

University of Washington Mining Society, Seattle, Wash. *President*, Oliver P. Searing; *Secretary*, Albert R. Sherman.

Student Branch of the American Institute of Mining Engineers, Iowa State College, Ames, Iowa. *President*, Lyle A. Butler; *Secretary*, A. J. Crawford.

Missouri Mining Association of the Missouri School of Mines, Rolla, Mo. *President*, L. S. Copelin; *Secretary*, L. J. Boucher.

The Pick and Shovel Club of the Case School of Applied Science, Cleveland, Ohio. *President*, M. T. Whelan; *Secretary*, Lloyd A. Collier.

Colorado School of Mines Scientific Society, Golden, Colo. *President*, Alan Kissock; *Secretary*, George Wilfley.

Mining Engineering Society of the University of Arizona, Tucson, Ariz. *President*, J. Gary Lindley; *Secretary*, H. O. Coles.

Kentucky Mining Society, College of Mines and Metallurgy, University of Kentucky, Lexington, Ky. *President*, Oliver W. Smith; *Secretary*, Charles E. Straub.

Mining Society of Pennsylvania State College, State College, Pa. *President*, Ralph E. Kirk; *Secretary*, Willard M. Lindsey.

## LIBRARY.

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS.

AMERICAN SOCIETY OF MECHANICAL ENGINEERS.

AMERICAN INSTITUTE OF MINING ENGINEERS.

UNITED ENGINEERING SOCIETY.

WILLIAM P. CUTTER, Librarian.

The Library of the above-named Societies is open from 9 A.M. to 9 P.M. on all week-days, except holidays, from September 1 to June 30, and from 9 A.M. to 6 P.M. during July and August. The Library contains about 55,000 volumes, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library can render valuable service through correspondence; letters requesting information will receive special attention. The Library is prepared to furnish references and copies of articles on mining and metallurgical subjects; to determine the existence of mining-maps, and to furnish general information as to the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

The members of the Institute can be of service to the Library by forwarding copies of mining-reports, maps privately issued, and similar material, which will be classified, indexed, and made available to other members. Suggestions for additions to the Library, either by purchase or personal solicitation as gifts, will be welcomed. It is hoped that members while in the city will use the Library freely, and assurance is given that most careful service will be rendered to them.

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## LIBRARY RESEARCH-WORK.

During the year 1912 there were 160 searches made for members and non-members of the Founder Societies, and copies of the references have been preserved for the use of others. This work has been largely based on requests sent in by mail, from Japan, South Africa, South America, Mexico, Canada, and England, as well as from different parts of the United States. The library receives 670 technical periodicals which are available through the indexes for this special purpose.

## Library Accessions.

May 1 to May 31, 1913.

[Copies of the list of additions to the Libraries of the American Society of Mechanical Engineers and the American Institute of Electrical Engineers can be obtained on application to the Secretary of the American Institute of Mining Engineers.]

ALBANY SOCIETY OF CIVIL ENGINEERS. Year Book, 1912. Albany, 1913. (Gift of Society.)

AMERICAN ELECTROCHEMICAL SOCIETY. General Index to Transactions, vols. I–XX. South Bethlehem, 1913. (Exchange.)

AMERICAN INSTITUTE OF CONSULTING ENGINEERS. Expert Evidence. A discussion held at a meeting of the Institute Oct. 25, 1912. New York, 1912. (Gift of American Institute of Consulting Engineers.)

AMERICAN INSTITUTE OF MINING ENGINEERS. Transactions. Vol. XLIII. New York, 1913. (Gift.)

ART CLUB OF PHILADELPHIA. Charter, Constitution and by-laws and List of Members, 1913. Philadelphia, 1912. (Gift of the Art Club.)

AUSTRIA. BERGWERKS INSPEKTION IN OESTERREICH, 1910. Wien, 1913. (Exchange.)

BOSTON PUBLIC LIBRARY. Annual Report, 61st. 1912–13. Boston, 1913. (Exchange.)

CALIFORNIA STATE MINING BUREAU. Bulletin No. 63. Sacramento, 1913. (Exchange.)

CANADA. MINES DEPARTMENT. Memoir No. 17–E. Ottawa, 1912. (Exchange.)

——— Memoir No. 35. Reconnaissance along the national Transcontinental Railway in Southern Quebec. Ottawa, 1912. (Exchange.)

CAPE OF GOOD HOPE. Geological Commission. Annual Report, 16th, 1911. Cape Town, 1912. (Exchange.)

COMMERCE AND NAVIGATION OF THE UNITED STATES, 1912. Washington, 1913. (Exchange.)

THE COMMERCIAL TREND OF THE PRODUCER-GAS POWER PLANT IN THE UNITED STATES. (Bulletin No. 55, U. S. Bureau of Mines.) Washington, 1913. (Exchange.)

DEVONIAN AND MISSISSIPPIAN FORMATIONS OF NORTHEASTERN OHIO. (Bulletin No. 15, Ohio Geological Survey.) Columbus, 1912. (Exchange.)

ENGINEERING INDEX, 1908–1910. New York, 1909–11. (Purchase.)

ENGINEERS' SOCIETY OF PENNSYLVANIA. Year Book, 1913. N. p., 1912. (Gift of Society.)

ENRICHMENT OF SULPHIDE ORES. (Bulletin No. 529, U. S. Geological Survey.) Washington, 1913. (Exchange.)

EXPERIMENTELLE UNTERSUCHUNG DES KUPPELOFEN SCHMELZPROZESSES. Dissertation zur Erlangung der Würde eines Doktor-Ingenieurs. By Friedrich Hüser. Düsseldorf, 1912. (Gift of Author.)

GEOLOGICAL SOCIETY OF LONDON. List of Members, 1913. Piccadilly, 1913. (Exchange.)

GEOLOGY AND ECONOMIC RESOURCES OF THE LARDER LAKE DISTRICT, ONT., and adjoining portions of Pontiac County, Que. (Memoir No. 17–E, Canada Department of mines.) Ottawa, 1912. (Exchange.)

GEOLOGY AND UNDERGROUND WATERS OF THE WICHITA REGION, NORTH CENTRAL TEXAS. (Water Supply Paper No. 317, U. S. Geological Survey.) Washington, 1913. (Exchange.)

ILLINOIS. STATE MINING BOARD. Annual Coal Report, 31st. Springfield, 1913. (Gift of Illinois Bureau of Labor Statistics.)

ILLUSTRIERTE TECHNISCHE WÖRTERBUCHER IN SECHS SPRACHEN. Band XI.—Eisenhüttenwesen. München, 1911. (Purchase.)

INSTITUTION OF MINING AND METALLURGY. Constitution and by-laws and list of members, Jan., 1913. London, 1913. (Exchange.)

——— Presidential address, by Bedford McNeill. March 13, 1913. London, 1913. (Exchange.)

INTERNATIONAL CABLE REGISTER OF THE WORLD, 1913. New York–London, 1913. (Gift of International Cable Directory Co.)

[NOTE.—This amazing directory is intended for use together with the Western Union telegraphic code, as the consummation of "a system by which the cost of domestic and inter-continental communication is reduced to a minimum." Without committing either

anybody else, or even myself, to a final position concerning the Government control of telegraphs, I feel it a duty, as well as an innocent pleasure, to bear witness to my own experience, through more than half a century, of the superiority of the service rendered to me by the Western Union Co., as compared to that which I have received from any Government in the world—and I have, at one time or another, employed a good many of them! Let me hasten to add that I am not puffing this company as against its great American rival. If that rival should get up a world code and a world directory as good as these of the Western Union, I would praise it with equal heartiness. My main feeling is, that, under the present system of private enterprise and competition, we Americans have better telegraph service than the citizens of any other nation. Perhaps recent experiences in the way of the loss or intolerable delay of U. S. mail-matter, and the smashing of packages by the new U. S. Parcel Post service, have unduly prejudiced me against the Progressive proposition that the Government should be made more efficient by having more functions given to it!—R. W. R.]

IRON AND STEEL WORKS DIRECTORY. Second Supplement, 1912. Philadelphia, 1912. (Purchase.)

IRON MINING IN MINNESOTA. (Bulletin No. 1, Minnesota School of Mines.) Minneapolis, 1912. (Gift of Minnesota School of Mines.)

JAHRBUCH DER ELEKTROCHEMIE UND ANGEWANDTEN PHYSIKALISCHEN CHEMIE. XIII. Jahrgang. Halle, 1913. (Purchase.)

JOHN CRERAR LIBRARY. Annual Report, 18th, 1912. Chicago, 1913. (Exchange.)

KONIGL. TECHNISCHEN HOCHSCHULE ZU MÜNCHEN. (Exchange.)

EISMANN, A. Der Synchronmotor als Phasenzahl-Umformer. München, 1912.

EISNER, G. Dampfturbinen mit veranderlicher tourenzahl. N. d.

GERSTACKER, L. Ueber Eisencyanide. Augsburg, 1912.

HARTMANN, A. Z. Ueber die Elektrolyse geschmolzener Borate. München, 1913.

HELLER, E. Ueber die Formgebung von Steuernocken. Berlin, 1912.

HILLEBRAND, F. Die Theorie des Drehstrom-Kollektor Nebebschlussmotors mit getrennter Erreger-und Kompensationswicklung. Berlin, 1912.

HUMBURG, K. Das Pendeln bei Gleichstrommotoren mit Wendepolen. Berlin, 1912.

MUENCH, K. Ueber das anodische Verhalten des Eisens. Rothenburg, 1913.

NÜRNBERG, A. N. Graphodynamische Untersuchung einer vierzylindrigen Fahrzeugmaschine mit veränderlichem Hub. Berlin, 1912.

STOKVIS, L. G. Der Spannungsabfall des Synchronen Drehstrom-Generators bei unsymmetrischer Belastung. München, 1912.

STROHMEIER, F. Ueber die Vestner'sche Fällung von Eisen, Kobalt, Nickel und Mangan. München, 1912.

UNTERSUCHUNGEN ÜBER FERROSILIZIUM. N. d.

WIESELSBERGER, C. Ueber die statische Längsstabilität der Drachenflugzeuge. Berlin, 1913.

ZAGELMEIER, F. Ueber die quantitative Bestimmung des mangans. München, 1913.

LEHRE VON DEN ERZLAGERSTÄTTEN. By Richard Beck. Berlin, 1901. (Gift of H. H. Knox.)

LORD, NATHANIEL WRIGHT. A Memorial. Columbus, 1912. (Gift of Ohio State University.)

MAP OF WEST VIRGINIA SHOWING COAL, OIL, GAS, IRON ORE AND LIMESTONE AREAS. 1913. (Exchange.)

METAL WORKER. Vols. 1–78. New York, 1874–1912. (Purchase.)

MINING INDUSTRY in that part of Northern Ontario served by the Temiskaming and Northern Ontario Railway, 1912. Toronto, 1912. (Gift of Ontario Legislature.)

MINING MANUAL AND MINING YEAR BOOK FOR 1912. By W. R. Skinner. London, 1913. (Purchase.)

NATIONAL MINE RESCUE AND FIRST-AID CONFERENCE, Pittsburg, Pa., Sept. 23–26, 1912. (Bulletin No. 62, U. S. Bureau of Mines.) Washington, 1913. (Exchange.)

NEW INTERNATIONAL YEAR BOOK, 1912. New York, Dodd, Mead & Co., 1913. (Gift of Publishers.)

[NOTE.—I need not repeat the statement which I made, in the *Bulletin* for June, 1912, concerning the immediate and supreme value of such annual cyclopedias as this, which is the sixth of the series begun in 1908, and covers the events of 1912. The stirring events

and burning questions of that year are treated with fullness and candor. The text is, as usual, all new, and in no part merely carried forward in revised form from previous volumes. The presidential compaign, the Balkan war, the Panama Canal, and many other subjects of foreign and domestic interest, are treated with satisfactory thoroughness and intelligence; and statistics of all kinds are given in remarkably up-to-date and convenient form.—R. W. R.]

NEW JERSEY. GEOLOGICAL SURVEY. Bulletin No. 9. Trenton, 1913. (Exchange).

NEW ZEALAND. GOVERNMENT STATISTICIAN. Report on the Results of a Census, 1911. Wellington, 1912. (Exchange.)

OHIO GEOLOGICAL SURVEY. Bulletin No. 15, Fourth Series. Columbus, 1912. (Exchange.)

OIL AND PETROLEUM MANUAL FOR 1913. By W. R. Skinner. London, 1913. (Purchase.)

ORE DEPOSITS OF THE HELENA MINING REGION, Montana. (Bulletin No. 527, U. S. Geological Survey. Washington, 1913. (Exchange.)

OVER EEN NIEUWE VERWERKINGSMETHODE VAN PYRIETHOUDENDE TINERTSEN, DOOR CH. TH. GROOTHOFF. N. p., n. d. (Gift of Billiton Maatschappij.)

PETROLEUM IN SOUTHERN CALIFORNIA, 1913. (Bulletin No. 63, California State Mining Bureau. Sacramento, 1913. (Exchange.)

PHILIPPINE ISLANDS. BUREAU OF SCIENCE. Annual Report, 11th. Manila, 1913. (Exchange.)

RELATORIO APRESENTADO AO PRESIDENTE DA REPUBLICA DOS ESTADOS UNIDOS DO BRAZIL. By J. B. Gonçalves. Rio de Janeiro, 1912. (Exchange.)

REPUBLICA ARGENTINA, MAP. N. d. (Gift.)

SAMMLUNG BERG UND HÜTTENMÄNNISCHER ABHANDLUNGEN. Pt. 124, 125. Kattowitz, 1913. (Purchase.)

SCHOOL OF MINES AND INDUSTRIES, Bendigo, Victoria. Prospectus, 1913. Bendigo, 1913. (Gift of School of Mines.)

STEEL RAILS. By W. H. Sellew. New York. D. Van Nostrand Co., 1913. Price, $12.50 net. (Gift of Publishers.)

[NOTE.—This book, compiled by William H. Sellew, Principal Assistant Engineer of the Michigan Central railroad, is an endeavor to systematize existing knowledge upon the subject, and to present the important elements of the great problems which now occupy so much attention in the railroad world, namely: the modern conditions of service imposed upon railroad rails; the qualities required by those conditions; and the way in which, by the control of chemical composition, mechanical manufacture, and heat-treatment, such qualities may be secured in the highest degree and most effective combination. The volume is beautifully printed and abundantly illustrated.—R. W. R.]

U. S. BUREAU OF MINES. Bulletin Nos. 55, 62. Washington, 1913. (Exchange.)

——— Technical Paper No. 38. Washington, 1913. (Exchange.)

——— Monthly Statement of Coal-Mine Accidents in the United States, Jan. and Feb., 1913. Washington, 1913. (Exchange.)

U. S. CENSUS BUREAU. Special Reports—Financial Statistics of Cities having a population of over 30,000, 1910. Washington, 1913. (Exchange.)

U. S. GEOLOGICAL SURVEY. Bulletin Nos. 527, 529. Washington, 1913. (Exchange.)

——— Water Supply Paper No. 317. Washington, 1913. (Exchange.)

——— 59 topographic maps. (Exchange.)

U. S. INTERSTATE COMMERCE COMMISSION. Annual Report, 26th, 1912. Washington, 1913. (Gift of Interstate Commerce Commission.)

WALLAROO & MOONTA MINING & SMELTING CO., LTD. Reports and Statements of Accounts. 23d Annual Report. Adelaide, 1912. (Gift of Wallaroo & Moonta Mining & Smelting Co.)

WASTES IN THE PRODUCTION AND UTILIZATION OF NATURAL GAS AND MEANS FOR THEIR PREVENTION. (Technical Paper No. 38, U. S. Bureau of Mines. Washington, 1913. (Exchange.)

GIFT OF ENGINEERING AND MINING JOURNAL.

AKTIEN-GESELLSCHAFT FÜR BERGBAU, Blei -und Zinkfabrikation zu Stolberg und in Westfalen zu Aachen. Bericht über das Betriebsjahr, 1912.

AMALGAMATED COPPER CO. Report for the eight months ending Dec. 31, 1912. New York, 1912.

ANACONDA COPPER MINING CO. Report for the year ending Dec. 31, 1912. New York, 1912.

BROWN, FREDERIC. Latin America. Prepared for the Pan American Society of the United States. Sept., 1912.

CONGRÈS INTERNATIONAL DU PÉTROLE. Troisième session, 1907. Compte Rendu. Vol. I. Bucarest, 1912.

KÜHN, EMIL. Die chemischen vorgänge bei der Cyanlaugung von Silbererzen. Halle a/S., 1912.

NEW JERSEY. Bureau of Statistics of Labor and Industries. Annual Report, 35th. Camden, 1913.

RUSSIAN MINING JOURNAL (IN RUSSIAN). No. 12, 1912.

STANDARD CONSOLIDATED MINING CO. Annual Report, 34th. 1913.

SVERIGES OFFICIELLA STATISTIK. Handel Berättelse för ar 1911, af Kommerskollegium. Stockholm, 1913.

UTAH. State Bureau of Immigration, Labor and Statistics. First Report, 1911–12. Salt Lake, 1913.

STANDARDS AND RULES FOR ELECTROTECHNICAL CONSTRUCTION OF LARGE CURRENTS. low and high tension, approved of by the 6th all Russian Electrotechnical Congress, St. Petersburg, 1912.

GIFT OF MINERAL INDUSTRY.

CANADA. Customs Department. Trade and Navigation unrevised monthly statements of imports entered for consumption, January, February, 1913. Ottawa, 1913.

MEXICO. Secretaria de Fomento, Colonizasion e Industria. Anuario Estadistico de la Republica Mexicana, 1907. Mexico, 1912.

ROUMANIE. Ministère de l'Industrie et du Commerce. Annuaire Statistique de la Roumanie. Bucuresti, 1912.

TRADE CATALOGUES.

CARNEGIE STEEL CO., Pittsburg, Pa. Steel derricks and drilling rigs. Ed. 4. 1913.

CHICAGO PNEUMATIC TOOL CO., Chicago, Ill. Bull. 34 L. General pneumatic engineering information. 47 pages.

—— Catalogue No. 42. Boyer Railway Speed Recorder. 20 pages.

IMPROVED EQUIPMENT CO., New York, N. Y. Bull. No. 6, Silica retorts and settings for gas benches. 1913.

JEFFREY MFG. CO., Columbus, O. Bull. 31 L. Swing hammer pulverizer. 4 pages.

LESCHEN & SONS ROPE CO., St. Louis, Mo. Leschen's Hercules. May, 1913.

WIEDEKE, GUSTAV & CO. Dayton, Ohio. Bull. No. 26. Ideal tube expander. 24 pages.

**United Engineering Society Library.**

BRIEF HISTORY OF TELEPHONE ACCOUNTING. By Charles G. DuBois. N. p., 1913. (Gift of American Telephone & Telegraph Co.)

LIBRARY OF CONGRESS. Classification. Class Q. Science. Washington, 1912. (Gift of Library of Congress.)

NEW YORK PUBLIC LIBRARY. List of Works relating to Electric Welding. New York, 1913. (Gift of W. B. Gamble.)

REPORT OF THE COMMITTEE appointed pursuant to House resolutions 429 and 504 to Investigate the Concentration of Control of Money and Credit. Feb. 28, 1913. (Gift of House of Representatives.)

UNIVERSITY OF ILLINOIS. List of Serials in the University of Illinois Library. Compiled by F. K. W. Drury. Urbana, 1911. (Gift of University of Illinois).

GIFT OF WILLIAM MCCLELLAN.

CONSERVATION. Vols. 14, 15. Washington, 1908–09.

ELECTRICAL WORLD. Vols. 17, 21, 23–32. New York, 1891, 1893–1898.

ENGINEERING NEWS. Vols. 51–53. New York, 1904–05.

ENGINEERING RECORD. Vols. 58–61. New York, 1908–10.

RAILROAD AGE GAZETTE. Vols. 45–46. New York, 1908–09.

RAILROAD GAZETTE. Vols. 42–44. New York, 1907–08.

RAILWAY AGE GAZETTE. Vol. 47. New York, 1909.

## MEMBERSHIP.

### NEW MEMBERS.

The following list comprises the names of those persons who became members during the month of May, 1913:

*Members.*

BASTIN, EDSON S., Geol.....U. S. Geological Survey, Washington, D. C.
BUCHERER, LOUIS, Cons. Engr.....Apartado 1471, Mexico City, Mexico.
BURDICK, CHARLES A., Min. Engr.....74 Broadway, New York, N. Y.
CARR, HOMER L., Min. Engr.....109 W. 88th St., New York, N. Y.
CARTER, RUSSELL S., Mech. Engr. & Asst. Genl. Mgr., Ingersoll-Rand Co., 11 Broadway, New York, N. Y.
CLAMER, GWILLIAM, 2d Vice Prest., Ajax Metal Co., Frankford Ave. and Richmond St., Philadelphia, Pa.
DAVIES, JOHN E., Engr....Morro Velho, Villa Nova de Lima Mines, Brazil, So. America.
DE SMITH, GEORGE, JR., Mine Surveyor.....P. O. Box 250, Virginia City, Nev.
DWYER, C. EUSTACE, Min. Engr.....Federal Mining & Smelting Co., Wallace, Idaho.
GATHMANN, EMIL, Mech. Engr.....Calvert Bldg., Baltimore, Md.
HALL, ROBERT D., Asso. Editor.....*Coal Age*, 505 Pearl St., New York, N. Y.
HANNA, HOWARD M., Jr.....M. A. Hanna & Co., Cleveland, Ohio.
HERIOT, ERIC M., Min. Engr., 19 Campden Hill Gardens, Kensington, London, England.
JANSEN, P., Min. Engr.....Lebong Tandai, Sumatra, Dutch East Indies.
KELLOGG, LEE OLDS, Min. Engr., *Engineering and Mining Journal*, 505 Pearl St., New York, N. Y.
KOEHLER, CARL F., Met.....P. O. Box 305, Elyria, Ohio.
LOVE, JAMES W., Min. Engr.....Tennessee Copper Co., Ducktown, Tenn.
PHIBBS, WILLIAM R., Asst. Mgr.....Northwestern Iron Co., Mayville, Wis.
ROHLFFS, WALTER V., Min. Engr.....1120 Lombard St., San Francisco, Cal.
SAYRE, ROBERT H., Min. Engr.....Central City, Colo.
SEAVER, KENNETH, Chief Engr. and Asst. Genl. Sales Mgr. Harbison-Walker Refractories Co., Pittsburg, Pa.
STRAND, CARLYLE H., Met.....Testing Dept., Penna. R. R. Co., Altoona, Pa.
TEEL, WILLIAM H., Commercial Traveler.....205 19th Ave. N., Seattle, Wash.
ZIMMERSCHIED, KARL W., Met.....84 Marston Court, Detroit, Mich.

*Change of Status—Associate to Member.*

EUSTIS, AUGUSTUS H.....Boston, Mass.
JONES, ZECHARIAH.....Republic, Wash.

*Associates.*

YOUNG, FRANCIS E.....55 Congress St., Boston, Mass.
STEBBINS, HOWARD S., Iron Ores, Oglebay, Norton & Co., 65 Wade Bldg., Cleveland, Ohio.

*Junior.*

NORRIS, ROBERT VAN A., JR., Student.....40 S. Franklin St., Wilkes-Barre, Pa.

### CANDIDATES FOR MEMBERSHIP.

The following persons have been proposed during the month of May, 1913, for election as members of the Institute. Their names are published for the information of members and associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Council, which has the power of final election.

*Members.*

**Joseph Warner Allen**, New York, N. Y.
Proposed by W. D. Thornton, John D. Ryan, W. S. Harper.
Born, 1874, Camden, N. J. 1880–91, Primary schools, Bordentown Military Institute, Pingry School (Elizabeth, N. J.). Prepared for Princeton University but did not enter.

1892, D. Somers & Sons. 1898, Barnes-Tucker Coal Co. 1901, U. S. Steel Corp'n. 1907, Butte Coalition Mining Co. and allied companies. *Greene Consolidated Copper Co. and allied companies. *International Smelting & Refining Co. and allied companies. *Inspiration Consolidated Copper Co.

Present position: Secy. and Treas. of companies above marked *.

**Robert Ammon**, Great Falls, Mont.

Proposed by C. W. Goodale, A. E. Wheeler, Milo W. Krejci.

Born, 1888, Elyria, Ohio. May, 1910, B. S. Mining and Metallurgical Engineering, Case School of Applied Science, Cleveland, Ohio. July, 1910, Millman at Arthur Hill, Utah Copper Co.

Present position: Oct., 1910, to date, Investigator, Anaconda Copper Mining Co., Great Falls, Mont.

**Andrew P. Anderson**, Los Angeles, Cal.

Proposed by Sidney J. Jennings, S. W. Mudd. C. Colcock Jones.

Born, 1862, Sweden. Common School, Greenview, Ill. 1882, General Mining, Leasing, Prospecting. 1889–1901, Foreman of Cleveland Mine and Bullycharp Mine, Shasta and Trinity Counties, Cal. 1901–2, Gen'l Foreman, Trinity Copper Co. 1902–3, Supt. Bullycharp Mine. 1903–4, Supt. Sheep Ranch Mine. 1905, Supt. Mammoth Mine, U. S. S., R. & M. Co.

Present position: 1905 to date, held various positions with U. S. S., R. & M. Co.; at present Pacific Coast Manager.

**Robert Wilson Bissell**, New York, N. Y.

Proposed by Arthur L. Walker, Henry S. Munroe, Robert Peele.

Born, 1866, Allegheny, Pa. Sept. 1888–June, 1891, Mass. Inst. Tech. Member Class 1912, College of Mines, Univ. Washington; B. S. in Mining Engineering. July, 1912, to date, Grad. Student Columbia University. 1889–8 and summers of 1889 and 1890, Construction Dept. (Chief Engineer's Dept.) Penna. R. R. Co. 1891–1904, Duquesne Forge Co., Pittsburg, Pa. (Supt. last four years). 1904–5, Asst. to Gen'l Mgr., Lustre Mining Co., Sta. Ma. del Oro, Dgo., Mex. Supt. 1905–8 and 1909–10.

Present position: Graduate Student Columbia University.

**Julius S. Bradford**, Chiksan, Korea.

Proposed by Louis Cohen, John Gross, James J. Martin.

Born, April 30, 1886, Peoria, Ill. 1910, Grad. Mining Engineering Colorado School of Mines. June–Dec., 1910, Assayer, Doyle Cons. Mines Co., Mancos, Colo. Jan.–Nov., 1911, Supt Tambourine Mine, Wallstreet, Colo.

Present position: Nov., 1911, to date, Mine Surveyor and Supt. Kuksukohl Mine, Chiksan Mining Co., James J. Martin, General Manager.

**Russell B. Caples, Jr.**, Anaconda, Mont.

Proposed by Frederick Laist, E. P. Mathewson, C. D. Demond.

Born, 1888, Glasgow, Mo. 1901, Public Schools of Glasgow, Mo. 1901–07, Pritchett College, Glasgow, Mo. 1907–10, Missouri School of Mines; B. S. in Mine Engineering. Aug.–Sept. 1910, Underground Work, Bunker Hill & Sullivan Mining & Milling Co., Kellogg, Idaho.

Present position: 1910 to date, Metallurgical Chemist, Testing Department and Laboratory of Washoe Smeltery of Anaconda Copper Mining Co., Anaconda, Mont.

**Thomas S. Carnahan**, Bingham Canyon, Utah.

Proposed by D. C. Jackling, R. C. Gemmell, C. W. Whitley.

Born, 1882, Tazewell, Va. 1904, B. S., Missouri School of Mines. 1905, A. M., Columbia University. 1906, Engineer, Tonopah Mining Co. 1907–08, Mining Engineer, Tonopah, Nev. 1909–10, Manager, Liberty Min. Co., Tonopah, Nev. 1911 to date, Utah Copper Co., Bingham Canyon, Utah.

Present position: Asst. Mine Engineer, in charge Underground Mines.

**William Carnahan Chancellor**, Pittsburg, Pa.

Proposed by L. L. Beeken, Fred Crabtree, C. T. Griswold.

Born, 1887, Parkersburg, W. Va. 1903–6, Parkersburg High School. 1906–9, Carnegie Institute Technology; graduated in Metallurgical Engineering. 1909–11, Chemist and Metallurgist, Lewis Foundry & Machine Co., Correopolis, Pa. 1911–12, Metallurgical Dept. Homestead Steel Works, Foreman Open Hearth Dept. same plant.

Present position: Chemist and Metallurgist, McConway & Torley Co., Pittsburg, Pa.

**Simeon Stansifer Clarke**, Leadwood, Mo.
Proposed by P. N. Moore, O. M. Bilharz, C. J. Adami.
Born, 1882, Columbus, Ind. 1902–4, Western University of Penna. (now University of Pittsburg), special course. 1899–1902, High and manual training schools, Chicago, Ill., and Columbus, Ind.; also course in I. C. S. Misc. dates, vacations, etc., practical work in assay offices and on surveying corps, prior to attending college. Odd times between 1898–1904, sample boy, asst. assayer helper on engineering corps, miner, etc., in different parts of the country. 1904–5, assayer, Virginia Cons. Copper Co.; T. A. Dunshee, Mgr. 1905–6, Chemist and foreman of cupola and refinery; A. W. Pollock, Supt. 1906–9, Engineer and assayer, Madison Lead & Land Co.; S. W. Hall, Mgr. 1909–10, Foreman, Cobalt refinery, North American Lead Co.; J. F. Eager, Supt.
Present position: 1910 to date, Engineer, St. Joseph Lead Co.; R. R. S. Parsons, Genl. Mgr.

**Durward Copeland**, Rolla, Mo.
Proposed by G. H. Cox, H. A. Buehler, P. N. Moore.
Born, 1880, Pubnico, Nova Scotia. 1899–1903, Mass. Inst. Tech.; S. B. 1903–4, Instructor, Mass. Inst. Tech. 1904–7, Instructor in Metallurgy, Michigan College of Mines.
Present position: 1907 to date, Professor of Metallurgy and Ore Dressing, Missouri School of Mines and Metallurgy, Rolla, Mo.

**Frank R. Corwin**, Great Falls, Mont.
Proposed by C. W. Goodale, A. E. Wheeler, J. H. Klepinger.
Born, 1886, Union City, Mich. 1908, Grad. in chemistry from the University of Michigan; Ph. C. 1908, Chemist, Peerless Portland Cement Co., Union City, Mich. 1909 to date, Anaconda Copper Mining Co., Boston and Montana Reduction Department, Great Falls, Mont.
Present position: Blast Furnace Foreman.

**Waldemar F. Dietrich**, San Francisco, Cal.
Proposed by G. H. Clevenger, D. M. Folsom, H. W. Young.
Born, 1892, Portland, Ore. May, 1913, A. B., Stanford University, Geology and Mining. 1907–8, Placer Mining, Placer Co., Cal., principally at Dutch Flat and Alta.
Present position: Jan.–Aug., 1913, Underground work, Hecla Min. Co., Burke, Ida. Sept., 1913–June, 1914, Instructor in Mining, Stanford University, Cal.

**Carroll Ralph Forbes**, Rolla, Mo.
Proposed by H. A. Buehler, G. H. Cox, P. N. Moore.
Born, 1882, Beatrice, Neb. 1902, B. S., 1903, E. M., Michigan College of Mines. 1903–07, Engineer, Victoria Copper Mining Co., Victoria, Mich. 1907–08, Engineer, Nevada Douglass Copper Co., Yerington, Nev.
Present position: 1909 to date, Professor of Mining, Missouri School of Mines and Metallurgy, Rolla, Mo.

**William Alexander Forbes**, New York, N. Y.
Proposed by W. R. Walker, W. J. Olcott, T. W. Robinson.
Born, 1876, Stockton-on-Tees, England. 1891, Public schools, Stockton-on-Tees, England. 1891–95, High School, Middlesbrough, England. May–Oct., 1895, Asst. Chemist, Black Diamond Steel Works, Pittsburg, Pa. 1895–1900, Asst. Chemist, Homestead Steel Works, Homestead, Pa. 1900–07, Asst. Chemist, Chief Chemist, Asst. Blast Furnace Supt., National Tube Co., McKeesport, Pa.
Present position: 1907 to date, engaged on metallurgical work for Steel Corporation, particularly in coal, coke and blast furnace work. Secretary of Corporation Coke and Blast Furnace Committees, etc.

**Alan Douglas Richard Galloway**, La Paz, Bolivia, South America.
Proposed by P. A. Seibert, Franz Germann, Mark R. Lamb.
Born, 1885, Sydney, Australia. 1897–9, Sydney Grammar School. 1899–1901, St. Joseph's College, Sydney, Australia. 1901–3, Sydney Technical College. 1904, Practical work in Conrad Consolidated Mines, New South Wales. 1905–7, Ballarat School of Mines, Ballarat, Victoria. 1908, Assayer at New Hillgrove Prop. Mines, Ltd., Hillgrove, N. S. W., and ore buyer for Lohmann & Co., Sydney. 1909, Practical work at Canadian Consolidated Mining & Smelting Co., Trail, B. C., and Federal Lead Co. (Standard Mammoth Mine), Mace, Idaho. 1910, Consolidated Arizona Smelting Co., as engineer. 1911, Copper Queen Copper Mining Co., as draftsman. 1911–12, Cerro de Pasco Mining Co., Peru, as draftsman, also operated San Jose silver-lead mine, Peru. 1912, Engineer on examination and prospecting work in Bolivia with Bolivian Exploration Co.
Present position: Privately practicing in profession at La Paz, Bolivia, S. A.

**Emil Grebe**, Bisbee, Ariz.
Proposed by Walter Douglas, Gerald Sherman, Roger T. Pelton.
Born, 1872, Germany. 1890, general education at schools in Germany. 1893–6, University of Zurich and eidgenossiger Polytechnicum. 1897, summer, University of Geneva; winter, University of Zurich. Examinations, submitting of thesis, and promotion to Doctor of Philosophy. 1903–5, Postgraduate course at Berg Akademie at Freiberg, Saxony. 1890-92, Asst. to Mgr. A. Zacharias of Bleialfer Bergwerksgesellschaft, Bleialf, Germany. Traveling in U. S., Canada, and France. 1898–1903, geological field work in Rhenish Prussia, as private assistant to my father, A. Grebe, geologist for Prussian Government. 1893–7, during vacation, geological field work in Germany and Switzerland. 1903–5, during vacation, inspection of mines and metallurgical plants in Germany, Austria, and Belgium. 1905–7, Asst. Genl. Mgr. and Genl. Supt. of St. Lawrence Pyrites Co., Ladenburg, Thalmann & Co., New York. 1907–8, Consulting Engineer for Edward Griner, Boston, Mass.
Present position: 1909 to date, Exam. Engineer for Copper Queen Cons. Mining Co., Bisbee, Ariz., and Phelps, Dodge & Co.

**William Hugh S. Grigsby**, San Mateo, Costa Rica, C. A.
Proposed by P. G. Spilsbury, A. G. Keiller, Robert S. Hanckel.
Born, 1874, Tokyo, Japan. 1884–1891, Merchant Tailors School, London, E. C., England. 1895, Accountant, Javali Mine, Chontales, Nicaragua, the Javali Co., Ltd., London. 1897, Mine Foreman; 1898, Ass't Mgr.; 1900–7, Genl. Mgr., Javali Mine. 1900–6, Lessee, Santo Domingo Mine, Chontales, Nicaragua. 1902-3, Lessee, Los Encinas Mines, El Jicaro, Segovia, Nicarague. 1904–9, Owner, La Tranea Mine, Chontales, Nicaragua. 1910, Night Superintendent, Tres Hermanos Mine, Abangarez Gold Fields of Costa Rica. 1911, Mine Captain, La Union Mine, Miramar, Costa Rica.
Present position: Mine Superintendent, Aguacate Mines, San Mateo, Costa Rica.

**Harry Thomas Hamilton**, Bisbee, Ariz.
Proposed by Walter Douglas, Gerald Sherman, Roger T. Pelton.
Born, 1880, Groton, Conn. 1904, Yale College; A. B. Dec., 1905–Aug., 1907, Saddle Mtn. Mining Co., Christmas, Ariz. Sampling and Miscellaneous work. Sept.–Nov., 1907, Cananea Cons. Copper Mining Co., Cananea, Sonora, Mexico; Bank. Dec., 1907–Jan., 1908, Copper Queen Cons. Mining Co., Bisbee, Ariz.; Rigging Gang. Feb.–Mar., 1908, Shannon Copper Co., Metcalf, Ariz.; assaying and sampling. Mar., 1908–May, 1911, Calumet & Arizona Mining Co., Geological Dept. one year, Accounting Dept. one year.
Present position; June, 1911, to date, Cost work, Mining department of Copper Queen Cons. Mining Co.

**Robert Howard Hawley**, Garfield, Utah.
Proposed by D. C. Jackling, R. C. Gemmell, C. W. Whitley.
Born, 1871, Missouri. 1889–93, Colorado School of Mines; Degree of Mining Engineering. 1895–99, M. Guggenheim's Sons, Philadelphia Plant, Pueblo, Colo. Asst. Chemist, Chemist, and Second Asst. Supt. Messrs. A. Raht, R. D. Rhodes, E. P. Mathewson. F. D. Weeks, and L. G. Eakins. 1899–1900, M. Guggenheim's Sons, and American Smelting & Refining Co., Monterey Plant, Mexico. Head Assayer. Messrs. H. M. Dieffenbach, W. V. Morse, and H. S. Mulliken. 1900–5, United States Reduction & Refining Co., Colorado, Philadelphia and Standard Plants, Colorado City, Colo. Special Chemist. Messrs. C. M. Macneill, J. D. Hawkins, and H. W. Fox. 1903–5, United States Reduction & Refining Co., United States Smelting Co., Canon City, Colo. Superintendent. Messrs. C. M. Macneill, D. C. Jackling, J. D. Hawkins. 1905–08, Nipissing Mining Co., Cobalt, Ont. Metallurgical Engineer. Messrs. Samuel Newhouse, E. P. Earle, T. R. Drummond, and R. B. Watson. 1910 to date, Utah Copper Co., Magna Plant, Garfield, Utah. Metallurgical Engineer and Asst. Supt. Messrs. D. C. Jackling, F. G. Janney, Sr., and D. D. Moffat.
Present position: Asst. Supt. Magna Plant, Utah Copper Co., Garfield, Utah.

**Max Hebgen**, Butte, Mont.
Proposed by C. W. Goodale, E. P. Mathewson, John Gillie.
Born, 1870, Beaver Dam, Wis. 1887, Grad. High School. Started Law Course, University of Wisconsin. Left before end of first term to begin practical engineering work in the Westinghouse Electrical Co. Have been following electrical engineering and its application for twenty years. 1889, Westinghouse Electric Co., Blaxter & Spicer Construction Co. 1890, Denver Cons. Elec. Co., Butte Elec. & Power Co., Great Falls Power Co.
Present position: Vice-President, Genl. Mgr. and Chief Engineer, Montana Power Co.

**Rush Miner Hess**, Guayaquil, Ecuador, S. A.
Proposed by T. W. Mather, Arthur Winslow, J. W. Mercer.
Born, 1882, Chicago, Ill. 1897–1901, Grammar School, Evanston, Ill. Evanston Township High School, Evanston, Ill. 1903, University of Illinois, Urbana, Ill. 1909, Special in Mining, Mackay School of Mines, University of Nevada, Reno, Nev. 1906–8, South American Development Co., Guayaquil, Ecuador, S. A.; Shift boss and engineer. 1910, Shift boss and engineer, Barron and Cortez Mines, Cia. Real del Monte y Pachuca, Pachuca, Hgo., Mexico.
Present position: 1911 to date, Engineer, South American Development Co., Guayaquil, Ecuador.

**Howard Judd Hilton**, Ray, Ariz.
Proposed by D. C. Jackling, R. C. Gemmell, Louis S. Cates.
Born, 1886, Leadville, Colo. 1902–6, High School. 1906–10, Colorado School of Mines; E. M. Summer of 1905, Mill of the Silver Lake Mine, Silverton, Colo. (Concentrating tables). Summer of 1906, Silver Lake Mine (Underground and assistant to the Engineer). Summer of 1907, Camp Bird Mills, Ouray, Colo. (Sampling and assistant solution man at cyanide plant). Summer of 1909, State Geological Survey of Colo. (Geology and topography). 1910, Engineering in Uintah Basin, Utah (Location and topography of irrigation and oil lands).
Present position: Mining department of Ray Consolidated Copper Co.

**Arthur E. Hodgkins**, Port Henry, N. Y.
Proposed by F. S. Witherbee, L. W. Francis, E. C. Witherby.
Born, 1873, Negaunee, Mich. High School Education. 1896–1909, Bookkeeper, Cleveland Cliffs Iron Co., Ishpeming, Mich.; M. M. Duncan, Agent. 1900–2, Accountant, Donora Mining Co., Palmer, Mich.; Oscar Rohn, Genl. Mgr. 1903–9, Witherbee-Sherman Co., Mineville, N. Y.; S. Norton, Genl. Mgr.
Present position: Genl. Mgr., Cheever Iron Ore Co., Port Henry, N. Y.

**Charles Wiswell Johnston**, Butte, Mont.
Proposed by George A. Packard, B. H. Dunshee, C. W. Goodale.
Born, 1881, Philadelphia, Pa. 1895–1901, Roxbury Latin School, Roxbury, Mass. 1901–5, Mass. Inst. Tech.; degree S. B., in Mining Engineering. 1905–6, Asst. Supt. Smelter, Luster Mining & Smelting Co. 1906–7, Engineer, American Smelters Securities Co., Veta Grande Unit. 1907–8, Engineer, Veta Colorado Mining & Smelting Co. April–Dec., 1908, Supt., Veta Grande Unit, A. S. S. Co. Dec., 1908–July, 1909, Supt., Reforma Unit, A. S. & R. Co. (under same management as Veta Grande). 1909–10, Mgr., Winchester Rock & Brick Co. April–July, 1911, Mgr. for Davis Sulphur Ore Co. of one of their properties.
Present position: Oct., 1911, to date, Raven Copper Co.

**Charles Loughridge**, Denver, Colo.
Proposed by F. H. Bostwick, D. W. Brunton, R. J. Grant.
Born, 1858, Oskaloosa, Iowa. Academic course at Yale, graduated 1883. President Pennsylvania Mining Co., a copper smelter at Campo Seco, Calaveras County, Cal. President Kokomo Metals Co., Magnetic Separation of iron and zinc at Kokomo, Colo.

**David Douglas Moffat**, Garfield, Utah.
Proposed by D. C. Jackling, R. C. Gemmell, C. W. Whitley.
Born, 1880, Salt Lake City, Utah. Graded Schools and three years High School. 1898–1903, Granite Bi-Metallic Mining Co., Phillipsburg, Mont. 2½ years as Machinist, 11 months as Master Mechanic and balance of time as Millman in various departments. 1903–5, Ontario Silver Mining Co., Park City, Utah. Master Mechanic and Night Mill Foreman. March–July, 1905, Mill Supt., Penn-Wyoming Copper Co., Grand Encampment, Wyoming. 1905–8, Foreman at Newhouse Mines & Smelter Co., Newhouse, Utah. June–Nov., 1908, Mill Foreman at Boston Consolidated Mills at Garfield, Utah. June–Nov., 1909, Supt., Taylor-Brunton Ore Sampling Works, Silver City, Utah. Oct., 1909–April, 1910, Genl. Foreman of Boston Consolidated Mills, Garfield, Utah. April, 1910–May, 1911, Asst. Supt., Arthur Plant, Utah Copper Co., Garfield, Utah.
Present position: Supt. of the Magna Plant of the Utah Copper Co.

**James Kane Murphy**, Anaconda, Mont.
Proposed by E. P. Mathewson, Frederick Laist, C. D. Demond.
Born, 1889, Butte, Mont. 1903–7, Butte High School. 1909–11, Montana State School of Mines; degree of Engineer of Mines.
Present position: June, 1911, to date, Chemist, Anaconda Copper Mining Co., Washoe Smelter.

**Roland Burrows Oliver**, Brussels, Belgium.
Proposed by S. H. Ball, Allen H. Rogers, Lucius W. Mayer.
Born, 1879, Gilroy, Cal. 1886-98, Common and High School education, with one year at University of Southern California, Los Angeles, Cal. 1899–1905, worked as field assistant and traverseman, U. S. Geol. Survey. 1905, passed Civil Service examination and was appointed Asst. Topographer, U. S. Geol. Survey. 1907–9, Topographer for Forminière Party, Congo Free State, Africa. 1910–12, in charge party prospecting and examining placer ground, Belgian Congo.
Present position: In charge party prospecting and exploiting diamond fields, Belgian Congo and Portuguese Angola.

**Harold Robert Perry**, Boston, Mass.
Proposed by Robert H. Richards, H. O. Hofman, Charles E. Locke.
Born, 1887, Napanee, Ontario, Canada. 1910, S. B. degree from Mass. Inst. of Technology. 1909, Kerr Lake Mining Co., Cobalt, Ont.
Present position: 1910 to date, Mining Department at Mass. Inst. of Technology.

**Horatio Cadwallader Ray**, Pittsburg, Pa.
Proposed by M. E. Wadsworth, S. L. Goodale, H. B. Meller.
Born, ——, Tyrone, Pa. 1899, Tyrone Public Schools. 1899–1900, Penna. State College, Preparatory Dept. 1900–4, Penna. State College; B. S. in Mining Engineering. 1904–5, Asst. in Mineralogical and Metallurgical Laboratories, Pa. State College. Candidate for Engineer of Mines Degree, which I shall receive this spring. 1905–6, Assayer, White Silver Co., Cobalt, Ont.; C. A. O'Connell, Supt. Apr.–July, 1906, Engineer, Copper Queen Cons. Co., Bisbee, Ariz.; Chief Engineer, W. McBride. July–Aug., 1906, Sampler, Shannon Copper Co., Metcalf, Ariz. Aug.–Nov., 1906, Chicago, Milwaukee & St. Paul Ry., Western Mont. Dec., 1906–May, 1907, Lanyon Zinc Co., Leadville, Colo.; J. M. McClave, Mgr. Aug.–Oct., 1907, Shift boss, Colo. Zinc Co., Denver, Colo. Oct., 1907–May, 1908, Asst. Mgr., Griffin Mg. & Smg. Co., Sultepec, Mexico. June–Dec., 1908, Engineering at Portland, Ore. Jan.–June, 1909, Engineer, Colo. Fuel & Iron Co., Trinidad, Colo. June, 1909–Apr., 1910, Asst. Mgr., Griffin Mg. & Smg. Co., Sultepec, Mexico. Apr.–Sept., 1910, El Oro Mining Co., Fresnillo, Zac., Mex.
Present position: Sept., 1910 to date, Asst. Prof. of Met., School of Mines, Univ. Pittsburg.

**James Beverly Risque**, Copperhill, Tenn.
Proposed by W. D. Thornton, W. S. Harper, P. L. Foster.
Born, 1857, Georgetown, D. C. 1873–5, Preparatory Course, Georgetown College, D. C. Specialized later at Washington University, St. Louis, in Chemistry and Engineering. 1876–78, Assayer, Minebres Mining Co., N. M. 1878–83, Asst. Supt., Minebres Mining Co., N. M. 1883–4, Custom Assayer, Silver City, N. M. 1884–6, Leasing and Mining in N. M. 1886–93, Mgr., Bi-Metallic Mine, Granite, Mont. 1893–4, Supt., Old Abe Mine, N. M. 1894–5, Supt., Osborn Hill, Grass Valley, Cal. 1895–6, Supt. Oneida Mine, Jackson, Cal. 1896–1900, Supt., Virtue Mine, Baker City, Ore. 1900–3, Supt. Pinos Altos Mines, N. M. 1903–6, personal field work. 1906–7, Mining Supt., Calumet & Hecla, Lake Superior. 1907–10, Mgr., Utah Cons. Copper Co., Utah. 1910–13, Private practice, Salt Lake City, Utah.
Present position: Manager, Tennessee Copper Co., Copper Hill, Tenn.
Joined A. I. M. E. originally in 1883. Resigned about 1909.

**Selden Scott Rogers**, Great Falls, Mont.
Proposed by C. W. Goodale, A. E. Wheeler, Milo W. Krejci.
Born, 1886, Toledo, Ohio. 1896–8, Worthington Military Academy, Lincoln, Neb. 1898–1900, Private Tutor. 1900–1, Hill School, Pottstown, Pa. 1901–3, Volkmann School, Boston, Mass. 1906–9, Harvard College; A. B. degree. 1909–10, Graduate School of Applied Science, Harvard University, degree of Mining Engineer. 1909, employed as assistant to Mr. Walter E. Segworth, of Toronto, Canada, in examination of the Nova Scotia Mine, Cobalt, Ont., Canada. 1909, Assayer for the Trethewey Silver & Cobalt Mining Co., Cobalt, Ont. 1910–12, employed in testing department of Great Falls Smelter of Anaconda Copper Mining Co. Summer 1912, Assistant to manager of Arctic Placer Mining & Milling Co., Nome, Alaska.
Present position: Sept., 1912, to date, Investigator and later McDougall Foreman at Great Falls Smelter of the Anaconda Copper Mining Co.

**Jesse Herman Steinmesch**, Desloge, Mo.
Proposed by F. V. Desloge, Philip N. Moore, Bradley Stoughton.
Born, 1882, St. Louis, Mo. 1897–1900, St. Louis Manual Training School. 1902, Missouri School of Mines; B. S., E. M. 1903, Granite Bimetallic, Mining. 1906, Mine La Motte L. & S. Co., General.
Present position: 1906 to date, Ass't Supt., S. Desloge Consolidated Lead Co.

**Edward A. Suverkrop**, New York, N. Y.
Proposed by Arthur L. Walker, Robert Peele, Wm. Campbell.
Born, 1889, Glasgow, Scotland. Expect to get E. M. degree from Columbia in Sept., 1913. 1905–1906, Telegraph Hill Mine, Sutter Creek, Cal.; Miner. 1906–7, Orlean Bar Gold Mining Co., Orleans, Cal.: Miner. 1907–8, Cal. Mining & Dredging Syn., Orleans, Cal.; Miner. 1911–12, Original Mine, Butte, Mont.; Miner and timberman.
Present position: Student.

**J. Frank Thompson**, St. Francois, Mo.
Proposed by A. E. Ring, H. A. Guess, O. M. Bilharz.
Born, 1880, Fredericktown, Mo. 1895–9, Marvin College, Fredericktown, Mo. 1900–2, Missouri State University, Columbia, Mo.; C. E. Spending vacations while in college at Mine La Motte and Catherine Lead Co. at Fredericktown, Mo. 1901–11, as Mining Engr., engaged in Mine and General Survey work for the St. Louis Smelting & Refining Co. Underground Supt. 1911–13, Directing all mine operation and performing general work as Ass't. Supt. 1910–11, H. M. McChesney, Genl. Mgr.; G. C. Cole, Sec. 1911–13, A. J. Meier, Genl. Mgr.; J. A. Caselton, Sec.
Present position: Asst. Supt. St. Louis S. & R. Co., St. Francois, Mo.

**Roy Eugene Tremoureux**, Nevada City, Cal.
Proposed by A. D. Foote, Arthur B. Foote, William Hague.
Born, 1884, San Jose, Cal. 1902–7, 3 years University of California, 1 year underground at North Star and Keystone Mines, Cal., 1 year cyanide work. 1907–8, Assistant to surveyor, Tonopah Mining Co.; Jack Burgess, Surveyor. 1908–9, Assistant Accountant, North Star Mines, Grass Valley, A. B. Foote, Supt. 1909–10, Assayer, North Star Mines Co., Grass Valley, A. B. Foote, Supt. 1910–11. Metallurgist, Kenora Mines, Ltd., late R. B. Nickerson, Mgr., Kenora, Ont. 1911–12, Secretary, Tightner Mines Co., Middle Yuba Hydro Electric Power Co., Champion Mines Co. 1912–13, Foreman, Champion Mines, North Star Mines Co., and Asst. Supt.
Present position: Asst. Supt., North Star Mines Co., Grass Valley, Cal.

**Francis Edward Vaughan**, Bisbee, Ariz.
Proposed by Walter Douglas, Gerald Sherman, Roger T. Pelton.
Born, 1889, Ludden, N. D. 1904–8, High School, Bakersfield, Cal. 1908–12, University of California, Berkeley; B. S. Degree. 1919–11, Worked underground during summer with Empire Mining Co., Grass Valley, Cal., and with Yellow Aster Mining Co., Randsburg, Cal. 1912, summer, reported on Castar Mine, Ventura Co., Cal.
Present position: Oct., 1912, to date, Engineering Dept., Copper Queen, Cons. Mining Co., Bisbee, Ariz.

**Ernest Wander**, Waukon, Iowa.
Proposed by E. Gybbon Spilsbury, William B. Phillips, Horace H. Clark.
Born, 1886, Berlin, Germany. 1896–8, Real Gymnasium, Berlin, Germany. 1899–1901, Grammar School, Chicago, Ill. 1901–3, Manual Training High School, Chicago, Ill. 1904–5, College Preparatory Studies at Lewis Institute, Chicago, Ill. 1905–8, 1909–10, Missouri School of Mines and Metallurgy, Rolla, Mo.; Bachelor of Science in Mining Engineering. Dec., 1908 to Sept., 1909, Chemist, Peoples Gas Light & Coke Co., Chicago, Ill.; Frank Thomas, Supt. June–Dec., 1910, Assayer and Surveyor, New Pennsylvania Mining Co., Montezuma, Colo. March–Oct., 1911, Assayer and Surveyor, American Placer Co., Lee's Ferry, Ariz. Oct., 1911–March, 1912, Chemist and Assayer, Mariner & Hoskins, Chicago.
Present position: Chief Chemist, Missouri Iron Co.

**Edmund Worthington Westervelt**, Ray, Ariz.
Proposed by D. C. Jackling, Louis S. Cates, R. C. Gemmell.
Born, 1883, Emporia, Kan. 1901–2, Yale University. 1902–5. University of Rochester; A. B. 1911–13, Colorado School of Mines. 1905–8, Chief Clerk and Cashier, Western Electric Co., Chicago, Ill., and Dallas, Tex. 1909–10, representative in Joplin district of Edmund Lyon, Rochester, N. Y.
Present position: Engineering Department, Ray Consolidation Copper Co.

**Israel C. White**, Morgantown, W. Va.
Proposed by J. F. Kemp, Wm. B. Clark, Charles P. Berkey.
Born, 1848, Monongalia Co., Va. 1872, Grad. W. Va. University; A. B. 1875, A. M., W. Va. Univ. 1882, Ph. D., Univ. Arkansas. 1875–6, Studied Geology under Newberry at Columbia University, taking special and private instruction therein in addition to general class lectures. Also at same time took Chemistry under Chandler and blow-pipe analy-

sis. 1875–83, Asst. Second Geol. Survey, Pa. 1877–92, Prof. Geology, W. Va. University. 1884–91, Ass't U. S. Geological Survey during preparation and publication of *Bulletin* 65, U. S. G. S. 1888–97, Demonstrating the Anticlinal Theory of Oil and Gas rediscovered in 1883. 1892–1907, Treasurer, Geological Society of America.

Present position: 1897 to date, State Geologist, W. Va.

**George H. Wyman, Jr.**, Wallace, Idaho,

Proposed by William J. Hall, Rush J. White, Byron Wilson.

Born, 1884, Ventura, Cal. 1903, Boise, Idaho, High School. 1907, B. S. (M. E.), School of Mines, University of Idaho. 1907, Hercules Mining Co.; Harry L. Day, Mgr., Wallace, Idaho. 1908, Supt., Charles Dickens Mining & Milling Co.; A. D. Gritman, Mgr., Spokane, Wash. 1911, Supt., East Lincoln Mine, Pearl, Idaho; R. R. Rollins, owner, Des Moines, Iowa.

Present position: Experimental Engineer, Federal M. and S. Co.

**William Arthur Young**, Lewistown, Mont.

Proposed by D. C. Bard, Frank A. Linforth, C. H. Bowman.

Born, 1887, Miles City. 1900, finished common schools. 1903, finished High School. 1904–05, Attended Montana State College of Agriculture and Mechanic Arts for one and one-half years. 1906–9, Montana State School of Mines; degree of Engineer of Mines. 1905–6, Employed at practical mining for Barnes King Mining Co., at Kendall, Mont. June, 1909, entered employ of Gold Reef Lease, Giltedge, Mont., as mining engineer. 1911, Mine Foreman of Magannis Mine, Maiden, Mont., for same company.

Present position: July, 1911, to date, Member of Gold Reef Lease.

## *Associates.*

**Albert H. Anderson**, Worcester, Mass.

Proposed by F. H. Daniels, George N. Jeppson, Carl F. Dietz.

Born, 1889, Worcester, Mass. 1895–1903, Grammar School, Worcester. 1903–4, High School, Worcester. 1904–8, Evening Mechanical Drawing School, Worcester. 1908–10, American School of Correspondence. 1904. Chemical Laboratory, Norton Co., Worcester. 1905–6, Machine shop, same compauy. 1906, Drafting, Otis Elevator Co., Worcester. 1907, Drafting, Hobbs Mfg. Co., Worcester. 1907–9, Drafting, Norton Co. 1910, Chemical Laboratory, same company.

Present position: 1910 to date, Abrasive Mill Foreman, Norton Co.

**Alexander Vincent Dye**, Bisbee, Ariz.

Proposed by Walter Douglas, Gerald Sherman, Roger T. Pelton.

Born, 1876, Flora, Ill. 1901, A. B. degree, William Jewell College. 1902, A. M., same college. 1904, Ph. D., University of Leipsic, Germany. 1905, Special work in the University of Chicago. 1904–9, Head of department of Modern Languages, William Jewell College, Liberty, Mo. 1909–12, American Consul, Nogales, Sonora, Mexico.

Present position: Secretary to the General Manager, Phelps, Dodge & Co.

**Henry J. Hinterleitner**, Clearfield, Pa.

Proposed by Thomas A. Furniss, John McLeavy, Thomas Fisher.

Born, March 28, 1868, Pottsville, Pa. 1886, Pottsville High School. Feb.–Nov., 1887, Chainman, Coxe Bros. & Co., Drifton, Pa. 1887–91, Transitman, Philadelphia & Reading C. & I. Co., Shamokin, Pa. 1891–92, Mining Engineer, Silver Brook Coal Co., Silver Brook, Pa. 1892–1900, Mining Engineer, Clearfield, Pa. 1900–11, Mining Engineer, Spangler, Pa.

Present position: 1911 to date, Asst. to Genl. Mgr., Clearfield Bituminous Coal Corpn.

**Paul Munoz**, Los Angeles, Cal.

Proposed by Adolphe E. Borie, W. B. Devereux, Jr., H. A. J. Wilkens.

Born, 1883, Troy, N. Y. 1904, Graduated B. S. in M. E., University of Pennsylvania. 1904–5, Bethlehem Steel Co., Pa.; employed by A. E. Borie, V.-P. 1906–7, Las Cañas, Cuba, sugar plantation. 1908, Chaparra Sugar Co., Cuba; worked in mill. 1909–11, Braden Copper Co., Rancagua, Chile; worked under Mr. Pope Yeatman and R. T. White. 1912–13, Munoz & Munoz, contracting engineers, Los Angeles, with S. C. Munoz.

Present position: Member of firm of Civil and Mining Contracting Engineers.

**Sankar Govind Naravane**, New York, N. Y.

Proposed by Arthur L. Walker, Henry S. Munroe, William Campbell.

Born, 1884, Poona, India. 1898–1902, High School N. M. Vidylaya, Poona, India. 1902–5, Engineering, College of Science, Poona, India. 1909–13, Columbia University, School of Mines. 1905–6, Public works Department, Government of India, Surveyor on

Ghar Canal. 1906–7, charge of Steel Bridge construction on Ghar Canal. 1907–9, Sub-Divisional Officer of Roads and Buildings, Khandesh Sub-division. 1909, G. M. Gest & Co., Electric Engineers, New York, N. Y.
Present position: Senior Student, School of Mines, Columbia University.

**Bruno V. Nordberg**, Milwaukee, Wis.
Proposed by C. W. Goodale, B. H. Dunshee, John Gillie.
Born, 1857, Helsingfors, Finland. 1878, Graduated from Mechanical Engineering Department of the Polytechnic Institute of Helsingfors, Finland. 1875–80, worked in shops and engineering offices of different ship-building establishments in Northern Europe and Germany. 1880–90, E. P. Allis Co., Milwaukee, as designer in charge of their special work.
Present position: 1890 to date, Chief Engineer and President of Nordberg Mfg. Co.

*Junior Members.*

**Alexander I. Abrahams**, New York, N. Y.
Proposed by Arthur L. Walker, Robert Peele, William Campbell.
Born, 1890, London, England. 1908–11, Columbia College; A. B. 1911–13, Columbia School of Mines; E. M.
Present position: Student.

**Frederick Conrad Black**, Santa Cruz, Cal.
Proposed by S. B. Christy, E. A. Hersam, E. B. Durham.
Born, 1890, Santa Cruz, Cal. 1908–13, University of California.
Present position: Student.

**Anthony Wayne Caruthers**, New York, N. Y.
Proposed by Arthur L. Walker, Robert Peele, William Campbell.
Born, 1892, Irwin, Pa. 1898–1905, Irwin Public Schools. 1905–8, Irwin High School. 1908–9, Washington and Jefferson College.
Present position: 1909 to date, Student Columbia School of Mines.

**Davis Goldstien**, Passaic, N. J.
Proposed by Arthur L. Walker, Henry S. Munroe, William Campbell.
Born, ———, ———. 1905–9, Passaic High School.
Present position: 1909–13, Student Columbia School of Mines

**K. F. Hess**, Brooklyn, N. Y.
Proposed by Arthur L. Walker, Robert Peele, William Campbell.
Born, 1890, Lancaster, Pa. 1907, Boys' High School, Brooklyn, N. Y. 1909, Heffley Institute, Brooklyn, N. Y.

**Joseph Stanley Hook**, Ithaca, N. Y.
Proposed by H. Ries, R. V. Morris, L. C. Graton.
Born, 1888, San Francisco, Cal. 1910, A. B., Stanford University, in Geology. 1913, A. M., Stanford University, "Geology of the Pleasanton Quadrangle," Cal. 1909, Daly-West Mining Co., Park City, Utah. 1911–12, Geologic and Hydrographic Survey for Spring Valley Water Co., San Francisco. 1912, Instructor in Historical Geology, Cornell University.
Present position: Instructor in Economic Geology, Cornell University.

**Roy Leach**, Hayfork, Trinity Co., Cal.
Proposed by S. B. Christy, W. S. Morley, E. A. Hersam.
Born, 1884, Rohnerville, Cal.
Present position: Student (with four years' residence), College of Mining, University of California.

**Harry Rosmond Leonard**, Harrisburg, Pa.
Proposed by H. D. Pallister.
Born, 1889, Harrisburg, Pa. Will graduate from Penna. State College in June, 1913, from a four-year course in Mining Engineering. Summer of 1910–11, Pennsylvania Steel Co., Steelton, Pa.; G. S. Vichery, Supt. Summer of 1912, the U. S. Experimental Mine at Princeton, Pa.; L. F. Jones, Supt.
Present position: Student, Penna. State College.

**Robert Lunn, Jr.**, Jersey City, N. J.
Proposed by Arthur L. Walker, Henry S. Munroe, Robert Peele.
Born, 1889, New York City. 1904–8, Jersey City High School. 1908–10, Columbia College.
Present position: 1910–13, Student, School of Mines, Columbia University.

**Frederick J. Patchell**, Washington, D. C.
Proposed by Robert Peele, Henry S. Munroe, E. L. Kurtz.
Born, 1890, Washington, D. C. 1907–10, George Washington University. 1910–13, School of Mines, Columbia University. 1907–10, G. O. Totten, Jr., Washington, D. C., as draftsman and superintendent of construction.
Present position: Student, Columbia University.

**John Morgan Price**, Cleveland, Ohio.
Proposed by Charles H. Fulton, J. Burns Read, Zay Jeffries.
Born, 1889, Ironton, Ohio. 1913, Adelbert College, Western Reserve University; A. B. 1913, Case School of Applied Science; B. S. in Mining Engineering.
Present position: Student, Case School of Applied Science.

**George Anthony Prochazka, Jr.**, New York, N. Y.
Proposed by Arthur L. Walker, J. F. Kemp, Robert Peele.
Born, 1889, New York, N. Y. 1907, DeWitt Clinton High School.
Present position: Student, Columbia University.

**Walter Franklin Pyne**, New York, N. Y.
Proposed by Arthur L. Walker, Henry S. Munroe, Robert Peele.
Born, 1889, New York City, N. Y.
Present position: Student 1908–13, Columbia University; E. M.,

**William C. Schmidt, Jr.**, New York, N. Y.
Proposed by Arthur L. Walker, J. F. Kemp, Robert Peele.
Born, 1890, New York City, N. Y. 1904–8, De Witt Clinton High School, N. Y.
Present position: 1908–13, School of Mines, Columbia University.

**Ransom Evarts Somers**, Ithaca, N. Y.
Proposed by H. Ries, R. V. Norris, L. C. Graton.
Born, 1885, Waltham, Mass. 1904–8, Harvard College; Degree A. B. 1908–10, Harvard Graduate School; Degree A. M. (Geology). 1910–12, Graduate work at Harvard, and in Montana during summer of 1911. 1913, Graduate work at Cornell. Summer, 1910, Experimental man, Steptoe Valley Copper Smelter, McGill, Nev. 1909–10, Austin Teaching Fellow in Mining and Metallurgy at Harvard. 1910–12, Austin Teaching Fellow in Mineralogy and Petrography at Harvard.
Present position: 1912 to date, Instructor in Economic Geology at Cornell.

**Samuel Clyde Stillwagon**, Cleveland, Ohio.
Proposed by J. Burns Read, Zay Jeffries, Frank R. Van Horn.
Born, 1890, Niles, O. 1904–8, Niles High School. 1808–10, Hiram College, Hiram, O. 1910–13, Case School of Applied Science, Cleveland, O.
The two years at Hiram College were toward a B. S. degree not obtained there. The three years at Case School of Applied Science were spent in special work along Metallurgical and Mining work towards a degree to be granted May 29, 1913.

**Carl Trischka**, Richmond Hill, L. I.
Proposed by Arthur L. Walker, Robert Peele, William Campbell.
Born, 1884, New York, N. Y. 1908–09, Heffley Institute, Brooklyn, N. Y.
Present position: Student, Columbia University.

**Edmund Wendel**, Cleveland, O.
Proposed by Charles H. Fulton, J. Burns Read, Zay Jeffries.
Born, 1886, Cleveland, O.
Present position: Student, Case School of Applied Science.

**Alfred Leo Wise**, New York, N. Y.
Proposed by Arthur L. Walker, Robert Peele, William Campbell.
Born, 1892, New York City, N. Y.
Present position: Student, Columbia School of Mines.

**Carl L. Wood**, Cleveland, O.
Proposed by Charles H. Fulton, J. Burns Read, Zay Jeffries.
Born, 1891, Lyons, Ohio.
Present position: Senior in Metallurgical Department, Case School of Applied Science.

## CHANGES OF ADDRESS OF MEMBERS.

The following changes of address of members have been received at the Secretary's office during the month of April, 1913. This list, together with the list published in *Bulletin* Nos. 76, and 77, April and May, 1913, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Mar. 1, 1913, and brings it up to the date of June 1, 1913.

ALLEN, JOHN A., Care G. Finnemore, 1191 Rivadavia, San Juan, Argentine Republic, So. America.
AUSTIN, L. S.....Care A. Austin, Tooele, Utah.
BAIRD, GEORGE A.....1416 Westminster Bldg., Chicago, Ill.
BARBOUR, THOMAS J.....Metropolitan Club, 5th Ave. and 60th St., New York, N. Y.
BORDEAUX, ALBERT F. J.....Thonon-les-Bains, Savoy, France.
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## ADDRESSES OF MEMBERS AND ASSOCIATES WANTED.

| Name. | Last Address of Record, from which Mail has been Returned. |
|---|---|
| Armstrong, Richard E., | 416 E. So. 2d St., Salt Lake City, Utah. |
| Body, J. Francis, | Rochambeau Apartments, Baltimore, Md. |
| Casey, John P., | 1509 Montana St., El Paso, Tex. |
| David, W. M., | Care C. G. Hussey & Co., Second Ave., Pittsburg, Pa. |
| Ederheimer, Leopold, | Maine Mining & Mfg. Co., 52 Broadway, New York, N. Y. |
| Eggers, John H., Jr., | Hughes Creek Mine, Kings River P. O., Cal. |
| Ferguson, Donald, | P. O. Box 644, Goldfield, Nevada. |
| Fergusson, Hugh B., | 216 Loo Bldg., Vancouver, B. C., Canada. |
| Finletter, John R., | 625 St. Francis Hotel, San Francisco, Cal. |
| Geisendorfer, Henry A., | Ariz-Nev. Copper Co., Hillside, Ariz. |
| Heywood, William A., | 4 Broad St. Pl., London, E. C., England. |
| Hollis, R. W., | Silverton, Colo. |
| Horschitz, Richard J., | P. O. Box 453, Haileybury, Ont., Canada. |
| Jackson, Walter H., | Mapimi, Dur., Mexico. |
| Johnson, Dion L., | 325 Water St., Pittsburg, Pa. |
| Lampshire, John O., | Vulture Mine, Wickenburg, Ariz. |
| Malins, Francis A., | Metates Mine, San Marcos, Sin., Mexico. |
| Moore, Roy W., | P. O. Box 48, Velasco, Tex. |
| Nelson, D. W. C., | Baker City, Ore. |
| Peterson, Frank, | H. W. Hellman Bldg., Los Angeles, Cal. |
| Rathborne, Merwyn R. W., | Amargosa, via Las Vegas, Nev. |
| Reynolds, Llewellyn, | Socorro Mines, Mogollon, N. M. |
| Rickard, Harold, | Foley-O'Brien, Ltd., So. Porcupine, Ont., Canada. |
| Russell, Branch E., | Apartado 22, Nacozari, Son., Mexico. |
| Sanders, Wilbur E., | 825 E. Colorado St., Pasadena, Cal. |
| Stoddart, A. W., | 638 Salisbury House, London, E. C., England. |
| Van Ness, William W., | 622 Salisbury House, London, E. C., England. |
| Vinicombe, Robert E. B., | Post Restante, Vladikafkas, So. Russia. |
| Watson, Ralph W., | Calloo, Utah, Clifton Mail box. |
| Webster, Erastus H., | Hotel Cosmopolita, Guadalajara, Jal., Mexico. |
| Woods, Clarence, | Shawmut, Cal. |

## NECROLOGY.

The deaths of the following members were reported to the Secretary's office during the month of May, 1913:

| Date of Election. | Name. | Date of Decease. |
|---|---|---|
| 1907. | *Bayles, James C., | May 7, 1913. |
| 1897. | *McCarthy, M. E., | ——, 1912. |
| 1896. | *Williamson, William D., | December —, 1912. |

## BIOGRAPHICAL NOTICES.

*James Copper Bayles* was born in New York City, July 3, 1845, and received in the public schools of that city his early education. In 1862, at the age of 17, he enlisted for three months in the 22d Regiment of New York volunteers; and, at the end of that brief term, received from Governor Fenton a commission as Second Lieutenant in the artillery. He served for 18 months, during which he reached the rank of First Lieutenant. But a year's illness precluded further military activity. After recovering his strength, he secured employment with the Delamater Iron Works, of New York City, and, a little later, joined the engineering staff of the Raritan & Delaware Bay Railroad Co.

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* Member.

In 1865, he became associated with Gen. Charles G. Halpin, proprietor of the New York *Citizen*, of which, during an illness of General Halpin, he was the editor, in an exciting political campaign. Under this test, he exhibited in high degree the qualities indispensable to eminence in journalism—the ability to appreciate instantly, and to state off-hand with convincing force, the essential points of a pending controversy, combined with the physical and mental capacity of prolonged and intense labor. It is related that, on one occasion during this period, he wrote, in 30 consecutive hours, the "copy" for 26 printed columns. The reserve of power needed for such a "spurt," when coupled with the steady industry which fulfills, "on time," all regular duties, forms an invaluable qualification for a journalist. Brilliant fellows who are not reliable, and reliable fellows who are not brilliant, can always be found—and, of the two classes, the latter undoubtedly is most highly valued in the long run. But he who belongs to both classes may rise fast and far, if he has ambition too.

It is not surprising that Mr. Bayles followed the profession for which he was so well qualified, though the attractions of mechanical engineering may well have tempted him to another course. In 1868 and 1869 he was editor of the *Commercial Bulletin;* and in 1870 he resigned this place, to become editor of the *Iron Age*, a position which he retained for nineteen years. Having been, for a part of that period, the editor of a technical journal in New York City, I feel myself qualified to testify that, in my judgment, Mr. Bayles was an editor of the first rank, and that his versatility as a student of science, engineering, and even art (for he was an amateur painter of considerable ability), increased his editorial power, though no one of his avocations could have taken the place of his real vocation, as a basis for his career. Under his direction, the *Iron Age* acquired a dominant place in trade journalism; and I know that when he resigned its editorship, in 1889, his associates and his employer (David Williams) regretted his decision.

Meanwhile, however, he had engaged in other activities, which did not seriously interfere with his work on the *Iron Age.* In 1884, he was elected, and in 1885, unanimously re-elected, President of the American Institute of Mining Engineers, of which he had become a member in 1878.

The Institute has never had a more active, efficient and inspiring President. Especially in the preparation and conduct of the meetings, he was helpful and admirable. His addresses as official representative of the Institute were always dignified, graceful, and tactful; and as an after-dinner speaker, he ranked with Holley, Pechin and others of that brilliant company whose sparkling wit and ready eloquence made our early banquets memorable.

The following is a list of his contributions to the *Transactions:*

The Engineer and the Wage Earner, vol. xiv, p. 327 (1885).
Explosions from Unknown Causes, vol. xix, p. 18 (1890).
Microscopic Analysis of the Structures of Iron and Steel, vol. xi, p. 261 (1883).
Presidential Address, Chattanooga Meeting, vol. xiv, p. 3 (1885).
Presidential Address, New York Meeting, vol. xiii, p. 587 (1885)
Presidential Address, Philadelphia Meeting, vol. xiii, p. 288 (1884).
Presidential Reply to Address of Welcome, vol. xiv, p. 316 (1885).
Professional Ethics, vol. xiv, p. 609 (1886).
The Study of Iron and Steel, vol. xiii, p. 15 (1884).
Resolutions on the Death of David Thomas, vol. xi, p. 15 (1882).
Spirally-Welded Steel Tubes, vol. xix, p. 1112 (1891).
Spirally-Welded Tubing, vol. xvi, p. 547 (1878).
Remarks in Discussion of Mr. Yardley's Paper on Specifications for Cast-Iron Coated Water-Pipes, vol. xviii, p. 664 (1890).

In March, 1887, Mayor Abram S. Hewitt offered him the position of President of the Department of Health of the City of New York. It is reported that he declined this honor at first, but accepted it afterwards, on the assurance that it had no political party significance. Of that circumstance, I have no personal knowledge; but I can recall two incidents, creditable to both parties, which deserve, I think, to be made public here, for the first time.

At the time when Mayor Hewitt nominated Mr. Bayles as President of the Board of Health, he was personally much incensed by certain comments in the *Iron Age* on his course in Congress in connection with the tariff question, in which he thought that Mr. Bayles had done him great injustice, especially by not giving him a chance to explain or answer such criticisms in the same number of that paper. He told me afterwards that he had nominated Mr. Bayles for one of the most important places under his administration because he thought him the best man for the place, although, as an *editor*, the man was to be *condemned!* Still later, Mr. Hewitt expressed to me, as he had done officially in a public message, his satisfaction with the work of Mr. Bayles in his important department, and congratulated himself upon the appointment which he had made on public grounds, in spite of personal antagonism.

The other incident which I recall is the story which Mr. Bayles himself told me, in answer to my question, How he had got along with "Tammany." He said that, upon his assumption of official duty, he received the visit of a committee from Tammany Hall, appointed to ascertain what he was going to do with the patronage of the Health Department, and what weight he would give to Tammany recommendations of candidates for employment under him. He replied with a full and clear statement (such as he was eminently qualified to make) of the work of the Department of Health, and its bearing upon the life of babies and young children, to say nothing of people generally. He appealed to their knowledge of the existing demoralization of the Department, the head of which had been recently removed by the Governor. And finally, after having made them feel "pretty solemn," he told them that he couldn't deal with his great task unless he could have the very best men to carry out his orders; that they ought to help him find such men; that any candidates endorsed by them would receive his respectful consideration, and that their recommendation of a candidate as worthy of employment would have distinct favorable weight; but that, of course, the final test would have to be ascertained fitness, since, as he had explained, he was in charge of a sacred trust—the health of the people of New York, and of their children. At the close of his remarks, one of the Committee remarked, with the assent of the rest, that *that* was good square plain talk, and that he guessed it was right, too! The Committee departed in friendly spirit; and Mr. Bayles was never afterwards troubled by factional interference or partisan demands on the part of Tammany. I never knew a more striking instance of the power of "the art of putting things"—an art of which Mr. Bayles was master.

In 1884, he was elected President of the New Jersey State Sanitary Association, and in 1886 a lecturer in the Sibley School of Engineering of Cornell University. He was also a member of the Iron and Steel Institute of Great Britain, a charter member of the American Society of Mechanical Engineers, and one of the founders, and the first President, of the Engineers' Club of New York. He received the honorary degree of Ph.D. from Rutgers College.

Mr. Bayles resigned in 1889 his editorship of the *Iron Age*, in order to devote himself to a promising business enterprise, which resulted, after some years of intense effort and fluctuating prospects, in complete failure. In 1905 he was parted by death from his wife—a lady whose bright and intelligent cordiality will be remembered by those surviving veterans of the Institute who participated in its meetings during the '80's. Still later, he had to struggle with persistently recurring ill health. But, under all discouragements and afflictions, he continued his activity. For several years he was an editorial writer on the New York *Times*, and afterwards, until 1906, he was a special contributor to that journal on scientific and technical topics. During this period also, I had occasion, as Secretary of the Institute, to avail myself of his editorial experience and skill, in aid of the over-worked force of my department.

With failing health, he dropped at last even these occasional labors, and went to live with one of his sons at Easton, Pa. He died of pneumonia at the New York Medical College Hospital, on May 7, 1913, leaving two sons, both engineers: Lewis C. Bayles, of Easton, Pa., and Howard C. Bayles, at present engaged in Chile as a mining engineer.

*Alfred Wartenweiler* was born at Neukirch, Canton Thurgau, Switzerland, in 1848, and received his technical education at the Polytechnicum in Zurich. In 1869, he went to California, and soon became engaged in mining and milling on the Comstock lode, in Nevada, where he nearly lost his life in the great fire at the Yellow Jacket mine. Later, he worked for Deetken at the chlorination works in Grass Valley and Hornitos, Cal. In 1870, he was assistant superintendent at the Giant Powder Co.'s works near San Francisco, where he was seriously wounded by an explosion which destroyed the entire plant. In 1871, he became my assistant superintendent at the smelting works of Robbins Bros., in Salt Lake valley, Utah. During a prospecting tour down the Colorado river in 1872, he narrowly escaped death by drowning, and afterwards by starvation and thirst, in the Grand Cañon region. Subsequently he took a prominent part in the development of mining and smelting in the Great Basin. One of the first trained metallurgists in the region, he contributed greatly to improvement of the crude smelting-methods then in vogue. In 1875, he filled an engagement in northern Mexico, with unsatisfactory results to himself. Returning to Utah in 1876, he took charge of the Winnemucca works in Bingham Cañon. Engaged by the Lewissohns to operate their smelter in Meaderville, near Butte, he came to Montana in 1879. In 1882, he took charge of the new mill of the French Société Anonyme des Mines de Lexington, becoming general manager of its entire property in 1884. Feeling the need of change and rest, he resigned this position at the end of 1887, and spent the year 1888 traveling in Europe and elsewhere. In 1889, he went to Chili, Bolivia, and Peru, to report on the Potosi, Huanchaca and other mines.

Mr. Wartenweiler was best known later, by reason of his connection with the Exploration Company, and his activity in placing large American mines with English and other foreign investors. In this work he was associated with Henry Bratnober, and the two did much to build up confidence in American mines abroad. Working in connection with French experts, he sold in France the Boleo mines of Baja California; and his name is associated with many other profitable ventures. An excellent judge of mines, he was conscientious in his opinions and cautious, making few mistakes.

He was of a genial, kindly disposition, and is widely remembered for his charitable acts and his generous helpfulness to friends. He joined the American Institute of Mining Engineers in 1875. In 1911, his health broke down, doubtless as a result, in part, of some of the hardships of his very active career; and he died in California Dec. 13, 1912.

CHARLES C. REUGER.

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[1] Until Feb., 1914. [2] Until Feb., 1915. [3] Until Feb., 1916. [4] Until Feb., 1917.

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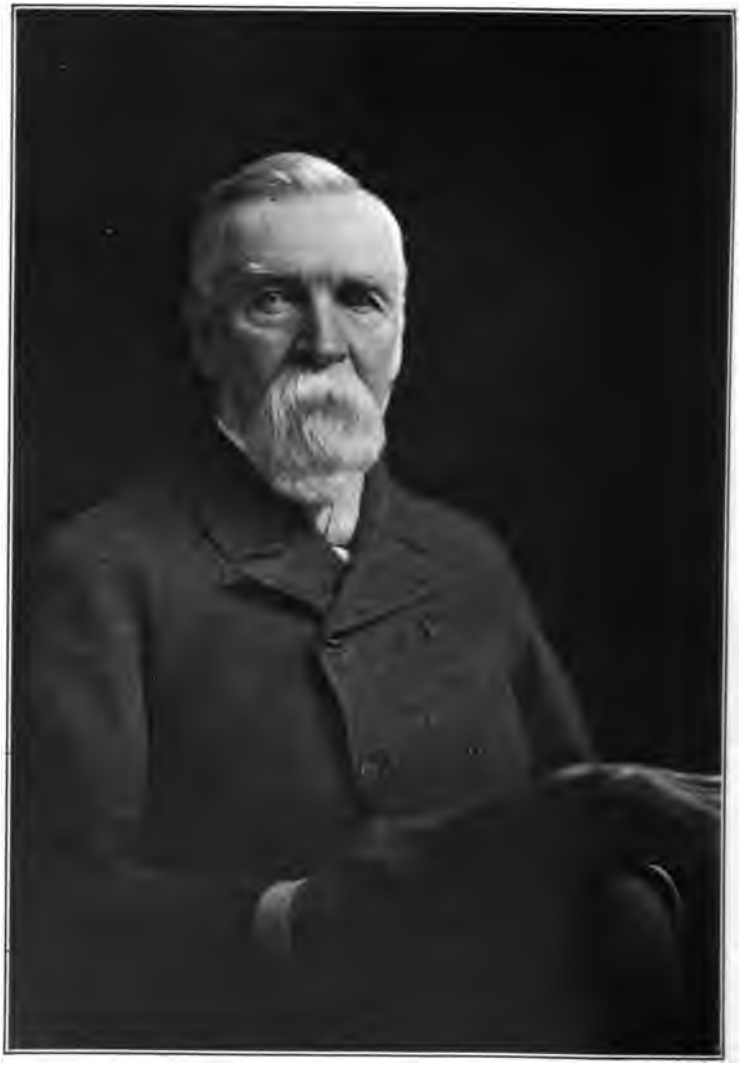

G. Fritz

# Biographical Notice of John Fritz.

BY ROSSITER W. RAYMOND, NEW YORK, N. Y., AND HENRY STURGIS DRINKER, SOUTH BETHLEHEM, PA.

(Butte Meeting, August, 1913.)

ON Mar. 28, 1913, the Board of Directors of the American Institute of Mining Engineers unanimously adopted the following Minute:

JOHN FRITZ, one of the most distinguished of American mechanical and metallurgical engineers, won that position by the force of innate genius, indomitable industry, unstained integrity and unfailing sympathy, and generosity towards his fellow-men.

Self-educated in the hard school of practice, he appreciated nevertheless the advantages of technical instruction and discussion, and evinced this appreciation both by his membership and lively interest in this and other similar societies, and by his munificent gifts to engineering education at Lehigh University, and his long and faithful service as a Trustee of that institution.

As one of the foremost of those American engineers who, through their brilliant inventions and practical skill, developed here the modern iron blast-furnace and rolling-mill, and introduced and perfected the Bessemer process and other improvements in the manufacture of steel, Mr. Fritz contributed mightily to the chief departments of that industrial progress which characterized the Nineteenth Century.

Proud of his great achievements, we cannot but rejoice over his long and fruitful life, crowned with a peaceful death; but our praise and thanks are mingled with sorrow, as we recall the kindly face which we shall see no more on earth, and the loyal friendship and spontaneous good-will which led the love of his generation, and the reverence of the generation which followed, to regard him universally as "Uncle John Fritz."

That summary, though only an outline, is a good portrait; and in the present sketch, we shall do no more than supply some details of the likeness, and some elements of the background.

John Fritz was born Aug. 21, 1822, in Londonderry, Chester county, Pa. His father, George Fritz, a native of Hesse Cassel, was brought to this country by his parents in 1802, with three brothers and a sister, to whom were subsequently added three daughters born in America. The family settled in Pennsylvania, and "grew up with the country." George Fritz married in 1821 the native-born daughter of a Scotch-Irish Presbyterian immigrant of 1787, and they had seven children—four girls and three boys, of whom John was the first. He was named after his grandfather, the foreign form, Johannes Fritzius, being modernized and Americanized into John Fritz. Thus he was descended from stanch and sturdy stock on both sides. His ancestors came here at a period when faith in the new Republic and the future

development of its domain under free institutions, brought to its shores the bravest and most enterprising of pioneers. It was the era of dauntless, independent individualism, and it produced among us a generation of strong men, whose personal gifts and ambitions could be developed freely in the stimulating atmosphere of liberty and opportunity.

The *Autobiography of John Fritz*, published in the autumn of 1911, bears unconscious testimony to the effect of this environment upon innate genius. His father, a mill-wright and mechanic, could not be content with farming, but repeatedly followed the call of the trade which he loved better; and the three sons, inheriting his talent and his predilection, after dutifully following the plough in their youth, abandoned it for the pursuit of mechanical engineering, in which, educating themselves without the aid of technical schooling, they all achieved high position.

Another influence, not to be overlooked, was that of the large family, with its necessary development of mutual affection and happiness. It was a sad thing for John Fritz, brought up in such an atmosphere, that to him and his beloved wife, during their long life together, only one child was given—a daughter, who died at the age of seven; but it may be fairly imagined that this experience had something to do with the fatherly and brotherly affection which he lavished upon the sons of others. If he had had, like his father, many children of his own, perhaps there would not now be so many to call him gratefully "Uncle John Fritz!"

It should be added that both his ancestry and his early life endowed him with splendid health and strength, so that his body was at all times able to respond to the demands of his spirit.

Finally, we cannot omit to mention (what John Fritz was wont, on all occasions, to emphasize) the moral influence of his God-fearing father and mother upon his whole life. Under that influence, added to all the rest, he became the strong, gentle, simple-hearted, high-souled man we knew and loved, combining with his own inborn genius the warm Irish heart, the steady German head, and the American courage and elasticity of endeavor.

Like other American boys, he had the benefit of some schooling; but his own epigrammatic summary, "Five days in the week, for three months in the year, is too short a time for the study of Bennett's Arithmetic," tells the whole story. The schooling of those days could only show the door, and give the key to those who would enter. Perhaps, after all, our modern systems accomplish little more!

In 1838, at the age of sixteen, he became an apprentice in the

trades of blacksmith and machinist—the latter comprising repairs of agricultural and manufacturing machinery, including the simple blast-furnaces of that day. The old apprentice-system, applied to such miscellaneous work, has largely passed away; but some of us veterans can testify that it turned out the best all-round mechanics we have ever known. Above all, it gave to the apprentice himself the opportunity to find and follow the line for which he was best adapted. That is what it did for John Fritz, who, at the end of his apprenticeship, returned to work for a time on the paternal farm, with his mind made up to engage somehow in the manufacture of iron, with special relation to its use on railroads. This early decision was illustriously justified by his subsequent career.

It was not until 1844 that he succeeded in making an entrance upon this career, by getting employment in a rolling-mill at Norristown, Pa., then in process of erection. After it started, he was put in charge of all the machinery, and soon discovered by annoying experience many weak spots in design and construction which he long afterwards remedied either by his own inventions or by those which he adopted and introduced. Among these were the two-high rolls and their cog-gearing, which he determined to abolish, if he ever got a chance. But meanwhile, not wasting his time in visions of future achievements, he seized the present opportunity to master thoroughly the thing nearest to him, outside of his immediate task. This happened to be the puddling-furnace. Many ambitious young mechanics have attended night-school after a day of hard work; but few have done so when the curriculum meant another shift of manual labor. John Fritz worked through a long day at his job as superintendent and repairer of machinery, and then spent the evening in the exhausting work of a common puddler, studying, while he rabbled or drew the glowing charge, the apparatus and the process. Months of such toil and thought made him at last not only a master-puddler, but also an expert, qualified to improve the old construction and practice. This accomplishment, however, he merely stored for the time when he should be able to use it, and, meanwhile, turned his attention to the heating, rolling, and finishing departments of the mill, with each of which, by the same method of actual practice at night, he acquired a similarly thorough familiarity. It is needless to say that, before long, John Fritz was the superintendent of the whole works, with the hearty support, not only of the proprietors, who knew his value, but also of the workmen, whose comrade and friend he had been—as, to the end of his life, he continued to be.

But, having learned what was to be learned in that particular busi-

ness, he accepted in 1849, with the sympathetic approval of Messrs. Moore and Hooven, his employers at Norristown, a position in connection with a new rail-mill and blast-furnace, erected at Safe Harbor, Pa., by Messrs. Reeves, Abbott & Co. The salary was smaller ($650 a year, instead of $1,000!); but he wanted to learn all about blast-furnace practice and the manufacture of rails. His strenuous and successful work at Safe Harbor was cut short after a few months by an attack of fever and ague, which forced him to give up, for a time, all active labor. During this interval, he made a trip to Lake Superior, and saw with wondering eyes the great Cleveland and Jackson iron-ore deposits in the Marquette district. After his return, he tried in vain to interest Pennsylvania capitalists in Lake Superior iron-mines, as a source of supply even for Pennsylvania. He was told that he might as well dream of bringing iron-ore from Kamschatka as from Marquette—to which he replied that, within ten years (this was in 1852), iron-ore from Lake Superior would be sold in Philadelphia. One-half the Jackson mine could have been bought then for $25,000!

But if his friends and former employers could not trust him as a prophet, they appreciated his character and experience as a mechanical engineer; and he was engaged in 1852 to superintend the rebuilding and improvement of the Kunzie blast-furnace, on the Schuylkill, about 12 miles from Philadelphia. This involved the new method of manufacturing pig-iron with anthracite, instead of charcoal or coke, as fuel —a scheme which had just been proved practicable by David Thomas and William Firmstone[1] in the Lehigh Valley. Mr. Fritz, though not the designer of the new furnace, was soon called upon to remedy defects in the original design, and managed to do so to the satisfaction of the proprietors, and without losing the friendship of the engineer whose opinion he had contradicted. After the furnace had been put in blast, and was running successfully, his desire to learn all about operation, as well as construction, led him to pursue his old habit of prowling about at odd times, day and night; and in this way he discovered, "by accident," as he says—but such accidents do not happen to everybody!—one of the most important principles of modern blast-furnace practice, namely, that of the "closed front," replacing the old fore-hearth and those frequent interruptions of the blast for cleaning out the crucible, known as "working" the furnace. This was, at the time, a revolutionary change of practice. The principle was afterwards embodied and made more effective by the water-cooled cinder-notch patented by Lürmann, and now universally employed. But,

[1] See *Trans.*, iii., 152, and xxix., 901.

while Mr. Fritz cannot be said to have anticipated that invention, as a means of facilitating the purpose in view, he was apparently the first, in this country at least, to recognize the importance of that purpose, and to carry it out in another way. When Lürmann's agent was trying to introduce his improvement in this country, the favorable opinion of John Fritz was one of the strongest arguments at his command.

In 1853, having got the Kunzie furnace machinery into good running order, and learned what was to be learned about the operation of the furnace itself, Mr. Fritz joined with his brother George and others in building at Catasauqua a foundry and machine-shop to supply blast-furnaces and rolling-mills. But in the following year he was invited, through David Reeves, who, as a proprietor both at Safe Harbor and at Kunzie Furnace, had learned to trust him, to go to the Cambria Iron Works, Johnstown, Pa., as General Superintendent.

This may be regarded as the turning-point of his career. His preparation for it had occupied sixteen years, during which he had mastered every part of the manufacture of iron into commercial forms—the blast-furnace, the foundry, the puddling-furnace, the heating-furnace, and the rolling-mill—while he had also learned the higher art of commanding the enthusiastic loyalty of workmen, and the highest art of all, perhaps—that of securing the confidence of employers.

All these patiently acquired qualifications were immediately demanded and tested in his new position, and the lack of any one of them would have been probably fatal to his success. The Cambria Iron Company was in a bad way administratively, financially, mechanically, and metallurgically, although, to his hopeful vision, "Cambria was destined to be the greatest rail-plant in the world." He had to meet successively the problems of technical authority and responsibility, temporary repair and reform of an old plant, improvement in quality of product, and the procurement of means for new and needed construction. When these problems had been so far solved that the mill "was running about as well as could be expected, and making some money,"[2] the property was attached under judgments upon former claims. Fritz persuaded all parties to allow the work to go on, and Fritz was the only man upon whom all parties could agree as an agent to protect the rights of all. As he says:

"Every evening, after the day's work was finished, I received the rails, first to secure the pay of the men, and secondly, in the name of the railroad company (the purchaser), to see that it got them."[3]

[2] *Autobiography of John Fritz*, p. 101.
[3] *Autobiography*, p. 103.

Under this *modus vivendi*, operations went on under the shadow of impending bankruptcy, until a reorganization with adequate capital was decided upon. This was not easily effected, under the complicated and discouraging financial circumstances of the company; and it is fair to say that confidence in the technical ability, good judgment, integrity, and loyalty of John Fritz, on the part of capitalists who knew him and his record, was the influence which turned the scale in favor of the enterprise. The capital was subscribed, and operations were resumed. But Fritz's troubles were not over. He was determined to put into the works, at the first opportunity, a three-high roll-train, in accordance with his prophetic vision of earlier years; and this plan was vehemently opposed by many of the stockholders, who were supported in their position by the opinions of leading iron-masters in all parts of the country, and the declarations of the laboring "heaters" and "rollers," who were, on principle, hostile to anything new. Fritz had to overcome this double antagonism; and it was by the sheer force of personal character that he secured, at last, authority for the execution of his plan. Against the denunciation of critics and the warning of friends, he introduced the three-high rolls into the Cambria Company's mill, laying thereby the foundation not only of unexampled prosperity for that establishment, but also of an improvement which was rapidly adopted throughout this country and the world, and has been justly called the last great step of progress in iron-manufacture preceding the Bessemer process.

But this triumph was followed by further trials. The day after the success of the three-high rolls had been demonstrated in the Cambria mill, the mill itself was destroyed by fire. Fortunately, the demonstration had been conclusive, so that, if the fire was the work of an enemy, it came too late to defeat the new invention. Fritz was equal to the emergency. Inside of thirty days, he had the mill running again, though without a roof to cover it; and it was one of the proudest recollections of his after-life that he subsequently erected a building 1,000 ft. long by 100 ft. wide, with trussed and slated roof—the finest rolling-mill building, at that time, in the United States—without interrupting the running of the mill which it covered, and without injury to a single person. He says of this work, finished in December, 1857:

"I was on the building while every member of every truss was being raised and put in place. This was the most trying ordeal that I ever had. The falling of a member of the truss, or a bolt, nut, wrench or tool of any kind, striking a man on the head, would cause instant death; and no person but myself knew what a load was off my mind when the last truss was in place, and I came down off the building for the last time." [4]

[4] *Autobiography*, p. 120.

It goes without saying that, in the progressive reconstruction of the Cambria works, Fritz introduced many improvements which he had conceived in previous years, when as yet there was no opportunity to realize them—improvements in puddling-furnaces, gearing, boilers, etc. One of his most characteristic and radical measures was the abandonment, in connection with the roll-trains, of light coupling-boxes and spindles, and a special "breaking-box," holding the rolls in place—all of which were intended to break under special strain, so as to save the rolls from fracture. The continual breaking of these weak parts was to him a source of greater delay and loss than was likely to occur in their absence. He said he "would rather have a grand smash up once in a while" than be thus annoyed all the time! And this remained his guiding principle as a mechanical engineer throughout his life. The structures and machines designed by him have been occasionally criticized, from the standpoint of theory, as unnecessarily costly at the outset; but, so far as we know, none of them ever failed in service. His trusses are still standing; his engines are still running; and perhaps his abundant "margins of safety" have proved to be worth more than they cost.

After six years of continuous hard work with the Cambria Iron Company, Mr. Fritz accepted in July, 1860, the position of General Superintendent and Chief Engineer of the Bethlehem Iron Company.

The works of this company, designed and erected by Mr. Fritz, were so far completed by September, 1863, as to begin the rolling of rails made from the product of its own blast and puddling furnaces. It is impossible, as well as unnecessary, to narrate here the history of his connection with this enterprise. A few features of it, however, deserve mention, by reason of their relation to the general progress of iron-metallurgy.

The first of these was the introduction of high-pressure blast in the iron blast-furnace. The iron-masters of the Lehigh Valley region were startled indeed, when they learned that Fritz was blowing air at 12 lb. per sq. in. into his furnaces, and was prepared even to blow at 16 lb. in an emergency. This method of overcoming by main force the internal difficulties which had previously been treated with so much old-fashioned skill, was the beginning of the new blast-furnace practice, in which rapid running, immense product and high blast, while creating fresh problems of blast-furnace management, have superseded many of the old ones.

Fritz's horizontal blowing-engines were much criticized at the time; but they have run continuously, day and night, for more than thirty years, blowing at from 10 to 12 lb. pressure, and frequently

more. He was so well satisfied with the result of his innovations in blast-furnace practice that he designed a larger furnace, with an engine that would supply a 20 to 30 lb. blast. But, to his great regret, the directors of the company were too conservative to authorize this experiment.

During the Civil War, the Government needed a rolling-mill somewhere in the South, in which rails torn from the track, twisted and deformed by Confederate raiders, could be re-rolled for renewed use. It was probably the advice of Abram S. Hewitt, whose patriotic services and wise counsel as an iron-master were so highly valued by Mr. Lincoln, which led to the selection of Mr. Fritz as one who could procure the necessary machinery and secure the erection of the mill with the least possible delay. He was surprised in March, 1864, by his appointment to this place with almost unlimited powers. His commission under the War Department declared that "any arrangements" he might make would be "fully carried out" by the Government, and continued:

"The early completion of this mill is important and indispensable to the advance of the Army; and all persons who may engage to furnish machinery or material therefor are directed to do so, to the exclusion of all other business!"

Mr. Fritz immediately prepared the plans and secured the necessary machinery for the mill, which was built at Chattanooga, Tenn., and of which his brother William was made Superintendent. William Fritz had been employed at Cambria and at Bethlehem until 1861, when he enlisted in the Union Army, and in 1864, he was on furlough, recovering from a serious wound. He ran the Chattanooga mill successfully until the end of the war.

The part taken by John Fritz at the Bethlehem works in the application and improvement of the Bessemer process in this country, was no small one. He was one of a notable group, comprising his brother George Fritz, then Superintendent of the Cambria Works, Robert W. Hunt, William Jones, and Alexander L. Holley, which used to meet frequently for the discussion of serious practical difficulties not communicated to the general public, or even to the technical societies and journals. He said afterwards of these conferences:

"We did not meet as diplomats, to find out what each other wanted; but we met as a band of loving brother engineers, trained by arduous experience, young, able, energetic, and determined to make a success. I doubt if ever five natural brothers were more loyal to each other than the five brother engineers above named. What each of us knew was common to all. Upon one occasion we all met at my house and talked over our troubles in detail; and they seemed so grave that some of us doubted if we could ever make the Bessemer process a financial success."

It is worthy of notice that these young engineers were all rail-makers; and it was in the manufacture of rails, more than in any other department, that the Bessemer process produced its widest and deepest effect throughout the civilized world, by its revolutionary improvement of the conditions, distances, speed, and economy of transportation. The troubles of which Mr. Fritz speaks in the foregoing quotation were those encountered in making good steel rails; and the chemists, physicists, and metallurgists would never have solved them without the aid of the practical rail-makers, of whom John Fritz was a leader and type.

During nearly thirty years of work with the Bethlehem Iron Co., Mr. Fritz, supported by the faith and courage which he inspired in other men, made that enterprise one of the most famous in the world —the Mecca of engineer-pilgrims from abroad and the pride and pattern of American practice. The introduction of open-hearth furnaces and of the Thomas basic process; the progressive improvements of strength, simplicity, and automatic handling in the rolling-mills; the adoption of the Whitworth forging-press; the manufacture of armor-plate; the erection of a 125-ton steam-hammer; and innumerable other improvements in the manufacture of iron and steel, owe their present perfection in large degree to his inventive genius, practical resourcefulness, and patient study. The stamp of his mind may be found on almost every detail of construction and operation throughout a wide range of processes and products.

In 1892, at the age of seventy, he retired from the responsible and arduous work at Bethlehem, which had occupied more than the latter half of the fifty-four years since his apprenticeship began. For nearly twenty years longer he lived to enjoy, as few men have been permitted to do, the fame and the friendships which he had amply earned. Indeed, he had received world-wide recognition before his retirement, and that event elicited numerous public expressions of the pre-existing fact. This Institute, of which he had been a loyal member since 1872, elected him its President in 1894;[5] the American Society of Mechanical Engineers, which he had joined in 1882, made him an Honorary Member in 1892, and President in 1895; the American Society of Civil Engineers, of which he became a member in 1893, conferred honorary membership upon him in 1899; the Iron and Steel

[5] Mr. Fritz made the following contributions to the *Transactions:*

| Title. | Volume. | Year. |
|---|---|---|
| Remarks on the Fracture of Steel Rails | III. | 1873. |
| Remarks on the Bessemer Process | XIX. | 1890. |
| Early Days of the Iron Manufacture (Presidential Address) | XXIV. | 1894. |
| Remarks on Rail-Sections | XXIX. | 1899. |

Institute of Great Britain made him an Honorary Member in 1893, and a perpetual Honorary Vice-President in 1909; and the recently organized American Iron and Steel Institute elected him an Honorary Member in 1910. Meanwhile, he had received the Bronze Medal of the U. S. Centennial Exposition in 1876; in 1893 the Bessemer Gold Medal of the Iron and Steel Institute; in 1902 the John Fritz Medal (the fund for which was established by subscription, to honor his eightieth birthday, by awarding a gold medal annually "for notable scientific or industrial achievement"—the first medal being bestowed with enthusiastic unanimity upon John Fritz himself); in 1904 the Bronze Medal of the Louisiana Purchase Exposition, in connection with which he served as Honorary Expert on Iron and Steel; and in 1910, the Elliott Cresson Gold Medal of the Franklin Institute of Philadelphia, "for distinguished leading and directive work in the advancement of the iron and steel industries." And he received *honoris causa* the following academic degrees: Master of Arts, from Columbia University, in 1895; Doctor of Science, from the University of Pennsylvania, in 1906; Doctor of Engineering, from the Stevens Institute of Technology, in 1907; and Doctor of Science, from Temple University, in 1910.

But these official distinctions could not tell fully the story of love and praise which pressed for the utterance which it found on two memorable occasions—celebrations of his seventieth and eightieth birthday anniversaries, in which hundreds of his friends and professional colleagues participated. The first took place at Bethlehem in 1892, and the second at New York in 1902. On the latter occasion, as has been said above, he received the first "John Fritz medal." Both are reported in the Appendix to his *Autobiography*, and need not be described here.

The conferment of honorary degrees by institutions of learning upon this self-educated workingman was a recognition not merely of his professional achievements, but also of his wise and generous aid to the cause of technical education, some account of which may fitly close this story of his life.

Lehigh University, located in the Lehigh Valley of Pennsylvania, was founded in 1866 by a Pennsylvanian—Asa Packer, who knew and appreciated the great qualities of John Fritz, and who named him as one of the original Board of Trustees. Established nearly half a century ago, at the period when men were beginning to feel that the material development of our natural resources and the extension of our transportation-facilities called for professional training as thorough as that previously given to the so-called "learned professions," this

institution had in its Board of Control, from the beginning, the strong common sense and the superlative engineering ability of John Fritz. For a wholly self-educated, self-cultured man, he was remarkably broad in his conceptions of education. Only a few years before his death, in commenting to a friend on the antagonism manifested by another distinguished iron-master to all forms of classical training, Mr. Fritz said:

"I think a well-educated man ought to know something of Greek and Latin. If I had a son I would see that he had some knowledge of those languages in addition to his more practical studies."

This from a man who, but for his natural culture and natural clarity of vision, might have been expected to look down on all but so-called "practical" education!

While not a wealthy man in the modern millionaire-sense of the term, Mr. Fritz, who though generous was thrifty, had laid aside and enjoyed a comfortable competence in his old age; and one day in the spring of 1909 he astonished President Drinker of Lehigh by saying,

"In my will I have left Lehigh University a certain sum of money to be expended in your discretion. I now intend to revoke that bequest, and instead of leaving money for you to spend after I am gone, I'm going to have the fun of spending it with you and Charley Taylor" (Mr. Taylor being a co-Trustee of Lehigh with Mr. Fritz, and an old and valued friend—a former partner of Andrew Carnegie). "I have long watched the career of a number of Lehigh graduates, and I have been impressed by the value of the training they have received at Lehigh. But you need an up-to-date engineering laboratory, and I intend to build one for you." [6]

Mr. Fritz acted as his own architect; designed the building, (substantially on the lines of the large shop he had built at the Bethlehem Steel Works); selected, purchased and installed the superb testing-equipment; and renewed his youth in the task, which was a great pleasure to him. At his death, it was found that (after making generous provision for his near relatives, and for bequests to the Free Library of the Bethlehems, to St. Luke's Hospital at South Bethlehem, to Temple College at Philadelphia, to the Methodist Hospital at Philadelphia, to the American University at Washington, and to other charitable purposes) he had bequeathed his residuary estate, estimated to amount to about $150,000, to Lehigh University, as an endowment-fund for the maintenance and operation of this Laboratory.

Mr. Fritz retained much of his vigor and activity up to the autumn of 1911. He took frequent trips alone to Philadelphia and New

[6] *Autobiography of John Fritz.* Account of Fritz Engineering Laboratory, p. 216.

York, and attended many gatherings of his old engineering friends and associates. In the spring of 1911, he decided, at the urgent solicitation of friends, to put into shape the notes of incidents in his life which he had been making for years. His first thought was to put this material into the hands of some competent literary man who should be employed to frame it into a biography; but he finally decided, largely on the insistence of those same friends, to undertake the task himself; and this was done during the summer of 1911 in his office at Bethlehem. The large pile of penciled notes, made in his own handwriting on yellow slips, was arranged chronologically by his nephew George A. Chandler, who, as an engineer, had had a close life-long association with Mr. Fritz; then Dr. Drinker, who was admitted to participation in the task, procured a competent stenographer; and they, with Mr. N. M. Emery, another friend, spent day after day, during the summer vacation-season, on the task. First, the crabbed desultory penciled notes were read aloud, and commented on by Mr. Fritz—every now and then with the injection of some delightful reminiscence or story—all being taken down by the stenographer, of whose presence Mr. Fritz soon became unconscious, as she was an unobtrusive, most competent little woman. As soon as this mass of matter had been typewritten, it was all read over again to Mr. Fritz, who again corrected, commented, and amplified. It was then turned over to the publishers (William H. Wiley claimed this privilege as a labor of love), and again the galley-proofs were similarly read, and the matter improved in Mr. Fritz's painstaking way. Finally the paged proofs were all read to him. The *Autobiography* was absolutely his own individual work. All that the devoted friends who were admitted to participation in its preparation did, was (as Dr. Drinker expressed it), to do the "cooly work," to perform the manual operations of authorship; the literary work, the direct forcible expression, the loving reminiscences, the jocund incidents of home- and mill-life are all the work of Mr. Fritz.

And then came the beginning of the end. This literary work finished, the laboratory built, his affairs in good order, our dear old friend began to fail. He suffered from recurring attacks of bronchitis during the autumn and winter, and finally an abscess formed on his chest, resulting from an infection from a germ of a previous severe attack of pneumonia. This abscess was opened by his physician, Dr. John H. Wilson, in February, 1912. Mr. Fritz, in his weakness, shrank from physical pain; so the spot was frozen by the application of chloride of ethyl before the knife was applied. When the patient heard the hissing of the gas, he turned languidly in bed towards Dr.

Drinker, who stood by him, and said, "Doctor, that sound reminds me of my first Bessemer blow!"

In March, 1912, his medical attendants expressed the opinion that unless he would submit to a drastic operation for the removal of pus on his chest, blood-poisoning would set in and death must soon follow; and Dr. Drinker was appealed to by the family to exert his personal influence as a friend to persuade Mr. Fritz to submit to the operation. In this he was successful; and the operation was performed April 15, 1912, by Dr. William L. Estes, Mr. Fritz's old and intimate friend, with Dr. Edward Martin, of Philadelphia, as consulting surgeon, and Dr. John H. Wilson as physician.

At this time Mr. Fritz again gave evidence of his characteristic sense of humor under any and all conditions. Every precaution was of course taken to ease the patient, and the surgeons arranged to bring from Philadelphia a special operator with apparatus to administer nitrous oxide, before subjecting him to the influence of ether. When Dr. Drinker explained this to him, Mr. Fritz said, "All right, but don't let them pull out any of my teeth"—the joke being that he had not a natural tooth left. This from a man in a state of extreme weakness, following weeks of suffering!

The operation was highly successful in averting the immediate threatened danger. Mr. Fritz wished to live; and his life was prolonged until February 13, 1913, when he died quietly, without apparent pain, passing away in sleep.

His funeral, held at Bethlehem on February 17, was attended by a large concourse of his friends; and he lies at rest in the beautiful Nisky Hill cemetery of his home town, beside his only daughter, who died in childhood, and his beloved wife.

So lived and died a great man—strong, wise, brave, invincible; a good man—simple, generous, tender and true; a loving husband; a loyal friend; a public-spirited citizen; a real philanthropist, giving "himself with his gift!" To us who miss and mourn him now, the man shines even more illustrious than the famous engineer.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.
[SUBJECT TO REVISION].

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# Notes on the Occurrence of Some of the Rarer Metals in Blister Copper.

BY A. EILERS, NEW YORK, N. Y.

(Butte Meeting, August, 1913.)

A NUMBER of the copper refineries in this country have lately separated some of the rarer metals from the slimes in the refinery tanks One of these has furnished me the following table of recoveries from given amounts of blister.

I publish this to show the proportions, to gold and silver and to each other, in which these rarer metals may be expected to occur in copper from certain localities.

TABLE I.—*Recoveries of Rare Metals from Blister Copper.*

| | Blister. Tons. | Gold. Ounces. | Silver. Ounces. | Platinum. Ounces. | Palladium. Ounces. | Selenium. Pounds. | Tellurium. Pounds. | Bismuth. Pounds. | Nickel. Pounds. |
|---|---|---|---|---|---|---|---|---|---|
| Garfield | 5,000 | 14,400 | 174,000 | 17.09 | 59.14 | 2,800 | 277 | 305 | 2,000 |
| Per 100 tons blister | ..... | 288.0 | 3,480 | 0.342 | 1.183 | 56.0 | 5.54 | 6.1 | 40. |
| Steptoe | 3,000 | 5,070 | 16,500 | 30.48 | 132.10 | 3,303 | None | 10 | 1,920 |
| Per 100 tons blister | ..... | 169.0 | 550 | 1.016 | 4.402 | 110.1 | ..... | 0.33 | 64. |
| Omaha | 800 | 2,880. | 184,720 | 14.60 | 51.89 | 213 | 537 | 151 | 7,552 |
| Per 100 tons blister | ..... | 360 | 23,090 | 1.825 | 6.486 | 26.6 | 67.1 | 18.6 | 944. |
| Mountain, Cal | 150 | 2,127. | 16,485 | 1.98 | 0.91 | 54 | 5 | 41 | 172 |
| Per 100 tons blister | ..... | 1,418 | 10,990 | 1.320 | 0.607 | 36.0 | 3.3 | 27.3 | 11.5 |
| Tacoma | 800 | 17,496 | 69,680 | 5.68 | 26.62 | 336 | None | 46 | 6,160 |
| Per 100 tons blister | ..... | 2,187. | 8,710 | 0.710 | 3.327 | 42.0 | ..... | 5.7 | 770. |
| Aguascalientes | 1,100 | 5,302 | 740,300 | 4.58 | 2.49 | 1,870 | None | 44 | 132 |
| Per 100 tons blister | ..... | 482. | 67,300. | 0.416 | 0.226 | 170.0 | ..... | 4.0 | 12. |
| Cerro de P. | 2,000 | 3,400 | 198,000 | 6.38 | 11.77 | 275 | None | 270 | 640 |
| Per 100 tons blister | ..... | 170. | 9,900 | 0.319 | 0.589 | 13.7 | ..... | 13.5 | 32. |
| Mt. Lyell | 800 | 3,716 | 57,640 | 4.99 | 10.99 | 336 | None | 34 | 1,328 |
| Per 100 tons blister | ..... | 464.5 | 7,205 | 0.624 | 1.374 | 42.0 | ..... | 4.3 | 166. |

GARFIELD, UTAH:—The blister copper comes principally from the Bingham porphyry copper mines.

STEPTOE, NEV.:—All the blister copper is produced from the Nevada Consolidated mines. (Also porphyry deposits.)

OMAHA, NEB.:—Blister copper comes principally from concentrated copper-lead mattes shipped from the different lead-silver plants of the Rocky Mountain region to the converters at the Omaha lead refinery.

MOUNTAIN, CAL.:—Ores occur in connection with diorite (?).

TACOMA, WASH.:—The blister is produced from Pacific Coast and Alaska copper ores, all of which occur in connection with diorite dikes and masses.

AGUASCALIENTES, MEXICO:—Blister is produced from the smelting of silver and gold ores with mostly low-grade copper ores, coming from all parts of Mexico and from many different geological occurrences and connections.

CERRO DE PASCO, PERU:—Blister from copper and lead-copper mattes produced at the Cerro de Pasco works. Ore originates in veins occurring in limestone near andesite masses.

MT. LYELL, TASMANIA:—Blister is from the well-known low-grade copper deposit of Tasmania. The geological occurrence is not known to me.

I am aware that the above recoveries do not represent the complete contents of the rare metals named, and I suspect that there are others in the original ores, which are lost in the successive metallurgical processes. Nevertheless, an imperfect record is better than none. Furthermore, it may serve as an incentive to our chemists for further research.

It may also direct the attention of some of our geologists, who work on the lines of chemical and physical influences in the original deposition of ores, to trace the reasons why the proportions of the rarer metals to each other are so different in different ores. It seems to me that geological conditions—broadly considered—must be to a great degree responsible.

As to the financial outcome of the extraction of the rare metals named, it must be said that, while a few have a good market, the majority have not. For them, markets must be made by new uses in the arts.

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# Topographic Maps for the Mining Engineer.

BY E. G. WOODRUFF, WASHINGTON, D. C.

(Butte Meeting, August, 1913.)

FEW authors of treatises and papers on engineering subjects have given adequate attention to topographic maps. The statement applies especially to mining engineering in all branches. Even those who have discussed such maps have treated the subject only in a general way. Therefore it is proposed to outline in this paper a few of the uncommon yet important relationships which topographic maps and their by-product, structure-contour maps, have to modern methods of mining investigation and development when the facts depicted on such maps are properly interpreted. Topography, as suggested by the etymology of the word, means a detailed description of particular places. Written descriptions have been found less effective than the pictorial representations, therefore attempts have been made in various ways to picture the surface features of places. Lines and shading have been used, hachures drawn, and, finally, the contour topographic map. To a large extent this style of map is the result of the demands of engineers. It is designed to meet their needs far more than those of the man untrained in engineering. In fact, the average man obtains a better idea of the topography of an area from the hachure system than from contour maps. Since the maps have been made chiefly for the use of the engineer, they ought to meet his demands on the one hand, and, on the other, should be used by him to the fullest extent. Such maps, when properly made, accurately portray a portion of the earth's surface and should present the topography better than a personal examination of the area without a map could present it, because the person making such examination views only a limited portion of the surface at a time, and estimates distances only roughly with his eye. Let the engineer consider carefully the fact that in many instances the topographic map is superior to a personal examination of the surface features of a property because the map is based on careful instrumental measurements, whereas the eye gives imperfect data because it can be used merely for the estimation and

not for the measurement of distances. Such estimation is frequently unreliable, since it depends upon the engineer's physical condition. When weary in the evening, after a hard day's tramp, he views a cliff or steep slope which he may have to climb in an entirely different way than he views it when he is fresh in the morning. The long mountain slope appears different in the ascent than during the descent. It is not uncommon for the traveler approaching the steep mountain front from the plains to feel that he is gradually descending, whereas he is actually going up a gentle slope. The experience is so common that, in irrigated regions along the mountains, the feeling is expressed in the conclusion that water runs up hill. These familiar experiences will no doubt suffice to demonstrate the statement just made, that, in some ways at least, topographic maps give better information than field examination.

That engineers rely upon topographic maps is emphasized by the following table, compiled under the direction of the Chief Geographer from the record of the U. S. Geological Survey to show the great number of maps purchased annually and also the growth of their use:

| Fiscal Year Ending June 30. | Maps Distributed. |
|---|---|
| 1897 | 101,974 |
| 1898 | 151,950 |
| 1899 | 168,641 |
| 1900 | 342,645 |
| 1901 | 327,603 |
| 1902 | 416,301 |
| 1903 | 440,422 |
| 1904 | 475,324 |
| 1905 | 566,016 |
| 1906 | 522,936 |
| 1907 | 595,915 |
| 1908 | 474,868 (including 369,521 sold). |
| 1909 | 597,361 (including 444,230 sold). |
| 1910 | 593,622 (including 478,737 sold). |
| 1911 | 684,129 (including 517,777 sold). |
| 1912 | 658,240 (including 529,656 sold). |

This table shows that more than half a million maps are now purchased annually for work in the mapped portion of the United States, which is 38 per cent. of the total area. At this same rate, if all the country were mapped, the demand during the year would have been for 1,400,000 maps. Furthermore, it seems very conservative to consider that at least half of the maps purchased during the preceding year would have been used also. On this assumption—namely, that the additional maps might have been sold, and that half of those sold during the preceding year were used—we infer a potential de-

mand for the use of 2,000,000 topographic maps. This conclusion does not seem unreasonable, because as people are educated to the use of topographic maps there is a greater demand, and as contiguous areas are completed the demand increases.

Both surface and subsurface features are now represented by contour maps, the former by the topographic maps and the latter by the somewhat less known structure-contour maps. Surface work is directly related to topographic maps and only indirectly related to structure-contour maps, whereas with underground work the reverse is true. Therefore, in this paper the two kinds of maps will be treated more or less independently. Furthermore, since most mining engineers are familiar with topographic maps and their uses, this paper discusses only certain data, presented on topographic maps but generally overlooked by the engineer because he fails either to read the map properly or to use the information which it contains.

One of the common problems in both the development and the operation of a mining property is to control the water supply which is to be used, and to protect the property from any excess that is not needed. In many plants the flood water of a nearby stream is a menace. It takes years of measurements to determine the high-water stage of a stream, but a close approximation may be made in another way. With the size of the drainage basin given and the daily rainfall known, the stream's flood stage can be roughly estimated. In the United States the observations of the Weather Bureau have given us data on the rainfall for most parts of the country. The other factors, such as the size and characteristics of the drainage basin, are shown on topographic maps. In this way two of the important elements in the solution of the water problem can be obtained. But more data can be actually obtained by proper interpretation of the map, especially with regard to flood. If the slopes are steep the run-off occurs rapidly and the streams are quickly flooded, but if the slopes are gentle the flow of the water is more gradual and lower flood stages may be expected. These considerations are not theoretical, but actual. They were real to the engineers of the Burlington railroad a few years ago when tracks were washed out by floods and bridges were overflowed. The railway's engineers attacked the problem just as indicated above. Rain-gauge measurements had shown the extremes of rainfall which were to be expected in a given area in southeastern Nebraska. They knew the character of the soil, forestation, and similar factors which control the run-off. Of course, the run-off as a result of the rainfall must pass under the bridges of the railroad. These bridges must be of a certain size in order to

permit the water to flow beneath, so as to avoid washing out the grade. The engineers recognized that, with their general knowledge of conditions and with specific data on the amount of rainfall and on the size and topography of the basin to be drained, they could closely estimate the maximum amount of water which might be expected to flow under any given bridge. As previously stated, they knew the amount of rainfall, but they did not know the second factor—namely, the topography; hence the railroad went to the expense of making a topographic map of the basins which drain towards the railroad in order that they might know the second factor, and thus construct their bridges properly.

A knowledge of the topography of a drainage basin is applicable in another way. Snow slides have wiped out more than one mining camp. With the fragrance of the pine enveloping him in the balmy days of summer, when claims are usually examined, the engineer does not fully appreciate the dangers from a winter's accumulation of ice and snow above the property in question. Possibly, as stated above, his point of view cannot give him the proper relations of slopes, which might warn him of the danger Yet many of us have seen those streaks on the mountain side bared of their normal verdant cover by the avalanche and know that slides are a menace. We have seen too that, as the avalanche plunged into the narrow valley below, the width of the area devastated was less, but the destructive power increased. Do all geologists and engineers, however, know that the topographic maps present data for the proper location of a camp to prevent its destruction by snow slides? The snow must accumulate somewhere, and it can slide only on steep slopes. The size and slope of the catchment basins and the angle of slopes over which the snow will move are shown on topographic maps. Reliable results can be determined from these data.

Commonly, topographic maps are used as a general guide and accurate surveys are made subsequently as a basis for detailed work; but in many cases the second and more expensive survey is unnecessary, because the accurate location of points can be determined from data shown on topographic maps. In this work the map is used as a base and needed refinements can be added to it. It may be that the map available is drawn with 50-ft. contours, whereas a 10-ft. interval is required for the work in hand. If so, the map may be accepted and supplemental contours and other refinements may be added. Ordinarily, the geologist takes his topographic map as a good guide, and a standard map is deserving of this confidence, but even good things have their limitations. On a 25-ft. contour map the elevation

between the two lines is generally estimated. In this case the error cannot be more than 25 ft. Likewise, horizontal positions are more or less a matter of estimation within certain limits, but the engineer is constantly demanding accurate measurements, not approximations. Good topographic maps like those of the U. S. Geological Survey furnish a basis for this accurate work.

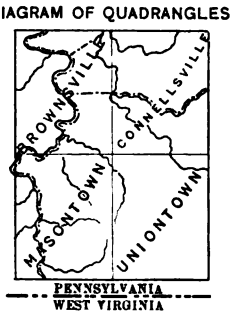
To obtain these precise data it is necessary for the engineer to repeat some of the operations of the topographer at the points where such data are desired. To do this work it has been found convenient to place the topographic sheet on a suitable-sized plane-table board, and use it as an original sheet; or, better still, to have a photolithograph printed on good drawing paper on a scale suitable for the work in hand. The engineer proceeds to the field equipped with his sheets, telescopic alidade, and stadia rod. The bench marks and triangulation stations, shown on the sheet, give him points from which he can establish his locations. This is an important feature. Here are useful data presented on topographic sheets and seldom considered, yet they furnish a basis for accurate work. They comprise some of the best data on the sheet, such as triangulation stations, houses, water towers, oil derricks, etc. Some points are shown only by bench marks, and signals should be re-established. If this is done for the engineer when he enters the field, he proceeds to the point whose location and elevation are desired and orients his table by the magnetic or three-point method, locates his point by intersections, and determines his elevation by vertical angles. After he has determined the location and elevation of his station he records the observations about the outcrop of the vein or bed. So far, the engineer has worked alone and he can continue to do so if necessary, but if he is accompanied by an assistant who can serve as a rodman, or, better still, who can operate the plane table, the engineer, serving as a rodman, traces the outcrop and holds the rod at points to be located. The plane-table operator sights to each stadia station and determines the location and elevation accurately. With these data he can plot the outcrop of a vein, if that is what he is tracing, though it is exposed intermittently, and determine its position with regard to land lines which are also shown on the sheet. Furthermore, he can obtain data from which he can determine the pitch of the vein. For the sake of illustration, a simple problem is assumed by supposing that the engineer has found three exposures of a vein, two at about equal elevations on opposite sides of a valley, and a third in the bottom of the valley, 500 ft. lower. If the three fall in a line the vein is vertical, but if the valley point falls to either side of the line joining the other

two the vein dips in the direction of that point in an amount equal to the distance of the point from a line joining the two points on the sides of the valley. In the case assumed above, if this distance of the point from the line is equivalent to 100 ft. the vein dips 100 ft. in 500, or 20 ft. to the hundred.

There are many additional applications of the data other than contours, but this discussion is not prolonged to include them.

The remaining part of this paper relates to structure-contour maps, or maps that might well be called strata-form maps because they portray the form of the strata. Their discussion is introduced in this paper because they are very closely related to surface-contour maps in their construction, interpretations, and uses. Such maps are in demand in coal fields, and especially in oil and artesian water fields, but are also applicable to some metalliferous areas. The methods of construction of such a map vary with conditions which are receiving consideration in some geologic magazines; therefore this paper begins with the assumption that the maps have been made.

These maps, illustrated in Fig. 1, show the structure of some distinctive stratum, such as a coal bed, oil sand, or any persistent bed. Contours are drawn on equal elevations of the bed just as surface contours are drawn through points of equal elevation of the surface. Synclines are depressions just as valleys are depressions, and anticlines are ridges in the strata just as mountains and hills are land ridges. To illustrate the relation of structure contours to topographic contours, let us suppose that we have strata folded into a syncline and an adjacent anticline, and that a definite unbroken stratum is the surface rock over both. Then the structure and surface contours are coincident if the same datum plane is used for both. Sea-level is generally used in the construction of both kinds of maps. The structure lines and topographic contour lines agree exactly. Now, if a portion of the anticline is eroded and the débris partly fills the valley without attendant crustal movement, the structure contours remain the same, but the surface contours are shifted. On the crest of the anticline the surface contours show lower elevations than the structure contours, whereas the reverse is shown in the valleys. In the construction of the map, however, the portion of the structure contour map representing the crest of the anticline is generally shown by broken lines to indicate that the bed does not actually remain and that its position has been assumed. In the adjustment of the assumed eroded surface the contours are shifted from their former position of coincidence to new ones, depending on the amount of erosion or deposition. If the syncline and anticline trend north-



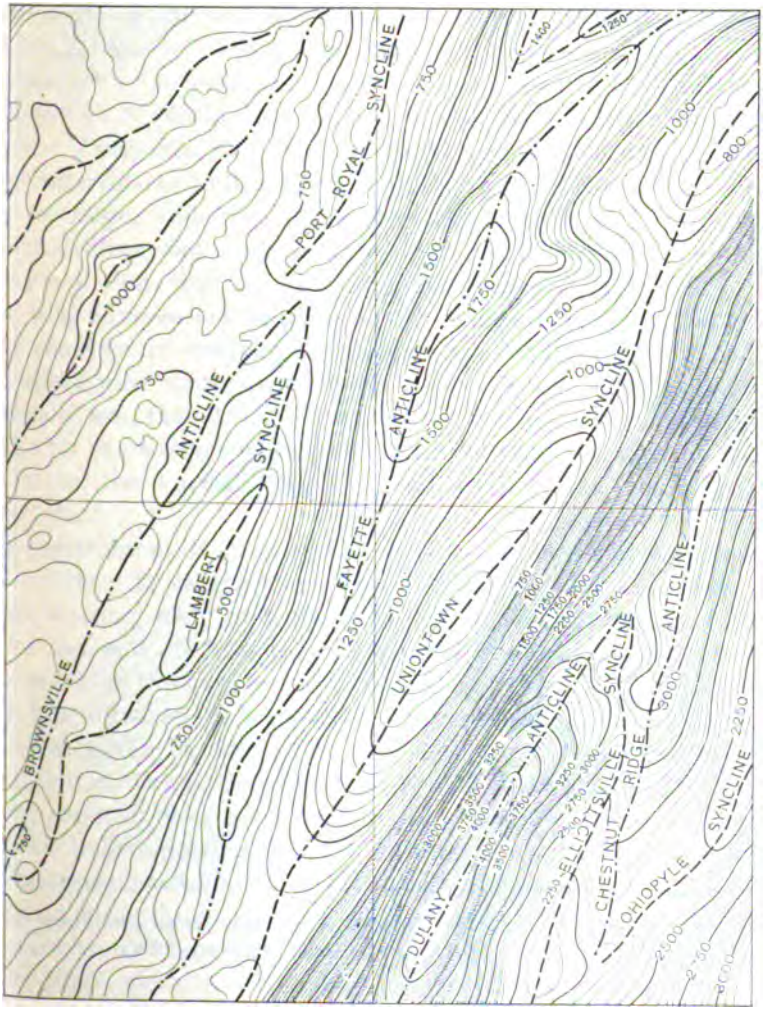




FIG. 1.—STRUCTURE-CONTOUR MAP OF FOUR QUADRANGLES IN SOUTHWESTERN PENNSYLVANIA. From *Folio No.* 94, *Geologic Atlas of the United States.*

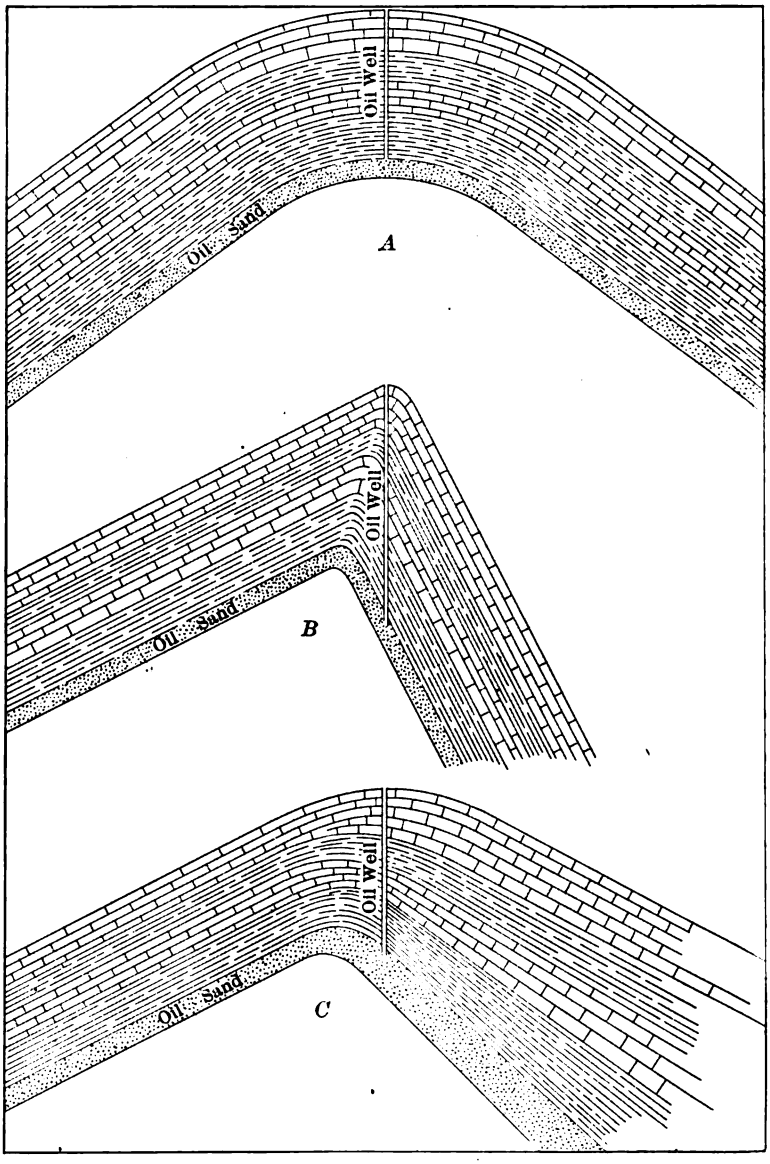



FIG. 2.—CROSS-SECTIONS OF HYPOTHETICAL ANTICLINES DRAWN TO SHOW THE RELATION OF AN OIL SAND TO THE SURFACE ROCKS.

south, the structure contours extend in the same direction; but if the syncline is filled so that the surface slopes to the north or south, the topographic contours extend east-west across the structure contours. The difference in elevation of any two lines at the point of intersection is the distance of the contoured strata from the surface. The depth at points between contour lines can be determined by interpolation. Thus, if a point lies midway between two surface contours, 2,100 and 2,150 elevation, its altitude is assumed to be 2,125, and if the same point is two-fifths the distance from the 1,150-ft. structure contour to the 1,200-ft. contour its elevation is assumed to be 1,170, and the depth of the bed below the surface is the difference between 1,170 and 2,125, or 955 ft. The uses of such maps are apparent and their accuracy is surprising. One such map, made by M. R. Campbell, of the U. S. Geological Survey, is published in the *Masontown-Uniontown Folio*, No. 82. The structure lines are drawn on 50-ft. contour intervals to indicate the position of the Pittsburg coal west of Laurel ridge, and the Upper Freeport coal east of that ridge. Diamond drilling and mining subsequent to the construction of the map have shown that the maximum error is only a few feet. Of course, it is unnecessary to point out the value of such a map to the mining engineer in studying a mining problem. Supposing he has both surface and structure-contour maps printed on one plate, as they usually are, then from the one sheet he can select the possible locations for the surface works, can infer the depth to the coal if it is a coal mine, the lowest point for drainage purposes, the slope for haulage, etc.

The construction of maps of this kind has often given surprisingly useful results. It is therefore urged that, whenever possible, the engineer dealing with such problems construct corroborative maps both for use and to reassure himself that he is correct. Two examples in which the engineer's judgment may be erroneous are given in Fig. 2 to illustrate this necessity. Supposing an oil well is to be sunk to reach the highest point in the oil sand of an anticline. If the anticline is symmetrical, as in *A*, Fig. 2, the well should be sunk upon the crest of the surface part of the anticline. If, however, the anticline is unsymmetrical, a location upon the previous supposition would be unsatisfactory, as shown at *B*. A structure map on oil sand would show the location of the highest part of the oil sand. In the construction of one such map the engineer found the structure in the immediate vicinity of possible oil location to indicate that the beds formed a fairly symmetrical anticline, but consideration of the data obtained in the surrounding area showed conclusively that the

crest of the contoured bed had been forced to one side at a depth of several hundred feet below the surface and was not directly under the crest, as the surface indications along the anticline would have led him to infer (*C*, Fig. 2). From these considerations the writer concludes that structure-contour maps are valuable to an engineer if constructed for him, but if forced to add details for himself he will receive great value from them both in making the proper field observations and in his subsequent interpretations. Of course, such work is hopeless without a topographic map; therefore, the preparation of topographic maps is advocated because of their usefulness in assisting the engineer to obtain a correct conception of the surface and also as a basis for the study of subsurface structural relations.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.
[SUBJECT TO REVISION.]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Butte meeting, Aug. 18 to 21, 1913, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1913, when Vol. XLVI. of the *Transactions* will go to press. Any discussion offered thereafter should preferably be in the form of a new paper for publication in Vol. XLVII. (with suitable cross references in both volumes).

# The Use of the Microscope in Mining Engineering.

BY FREDERICK W. APGAR, JAMAICA, N. Y.

(Butte Meeting, August, 1913.)

THE valuable results that have followed the application in recent years of microscopic methods of research to problems of ore genesis have been significant, but possibly the recognition of their practical importance is not as general as might be, and while, as a scientific method of investigating ores and rocks, the advantage derived through the use of the microscope is unquestioned, its utility as an aid in solving mining problems arising during actual every-day mine examination and operation may not be fully appreciated.

Perhaps the most logical method of showing some of these practical applications of the microscope will be to consider first the prospect stage of a mining venture, then the uses to which the instrument may be put during the operation of a working and developed mine, and, finally, to discuss some of the problems of a metallurgical nature that may arise during milling and smelting the ore.

The valuation of mining property as a preliminary to acquiring ownership or determining the advisability of investing time and money in an enterprise is one of the most important and delicate problems that come before the mining engineer. The many and variable factors that have to be taken into account, as, for example, the geology of the district, availability of supplies, probable mining costs, transportation facilities, and the quality of ore, require a careful balancing of costs against recoverable values.

Primarily, however, a study of the local geology is of first importance, since, in a prospect with little development work, the question of character and probable quantity of the ore and its value can only be approximated with any degree of accuracy by a thorough investigation of geologic conditions, and a general knowledge of ore deposits. Unless these are favorable and pay ore reasonably certain, other factors need not be considered. The character of the outcrop, evidences of mineralizing agencies in the adjacent rocks, and geological structure, are the data of chief significance.

Now, an early question which will present itself is: What sort of a deposit have we to deal with? Is it a fissure vein, for instance; is it a replacement, or is it possibly a contact-metamorphic deposit?

At first thought it might seem as though the broad general structural characteristics would serve to differentiate these classes, and usually an idea may indeed be gained in this way which will be proved by subsequent investigation to be correct. Not always, however. Let us take, for example, the first two types, fissure veins and replacement deposits. Normally these have few points in common. The former has relatively well-defined walls, and comparative regularity of thickness and character; the latter appears in ramifying masses and irregular mineralized areas throughout the containing rock.

Fissure veins are formed when mineral-bearing waters or gases traverse a pre-existing opening in the rock where some dynamic disturbance has caused fracturing and faulting. The channels for underground circulation so formed are gradually filled by mineral matter deposited from the solutions.

Replacement deposits, on the other hand, are formed by chemical interchange, or metasomatism. No open space for deposition exists except such minor cracks or fractures as are necessary to permit the solutions to penetrate the rock. The process is really one of substitution. Certain of the rock constituents are dissolved and, at the same time, metallic sulphides or foreign minerals are deposited. The reaction is simultaneous, molecule by molecule, solution of one and precipitation of the other, until finally the original grain is wholly replaced by a new constituent.

Now consider a certain type of replacement ore body often met with where rocks have been subjected to intense dynamic stress which has produced fractures along more than a single plane. The effects of the disturbance are distributed through a series of narrow parallel cracks, closely spaced in the zone of maximum movement and gradually separated by wider and wider intervals toward the outer limits. Under favorable conditions a complete replacement of the narrower tabular sheets of rocks toward the center of the zone might develop a solid mass of ore in which the original fissuring would preserve a structure closely resembling true crustification, while, toward the outer limits, a plate or wall of barren rock would separate adjacent areas of mineralization. In cases of this kind it might easily happen that errors of interpretation would result in misleading conclusions regarding genesis, and the presence of valuable ore might remain unsuspected.

Crustification, or banding, is one of the typical features of mineral

deposition in pre-existing openings, but the type of deposit just described differs little in its general aspect from a true fissure vein.

And now, having indicated the problem, let us inquire how the microscope aids in its solution:

By study and investigation of many various types of replacement ore bodies geologists have detected certain common peculiarities, and have come to recognize certain criteria as being characteristic of metasomatic conditions. It is through observations of this sort, some structural and some microscopic, that questions of origin and genesis may be determined.

On examining a normal igneous rock in thin section some of the component minerals will be found in well-developed crystal form; in other words, they are idiomorphic toward others which show only an irregular outline.

In order that a pyrogenetic mineral may develop crystallographic faces on all sides it must have formed freely while suspended in a liquid of approximately its own specific gravity and without interference from neighboring crystals. Therefore the well-bounded minerals are believed to be early crystallizations from the magma. Iron oxides, apatite, and titanite, for example, are often idiomorphic.

But when we examine a rock, either igneous, metamorphic, or sedimentary, that has been mineralized, and find that the new mineral, introduced long after the rock was formed, has complete crystal outline, it is believed to indicate metasomotic replacement, since this theory of molecular interchange seems the only reasonable explanation. Had the new mineral started growth in the interstices between grains and forced them apart, the fact would be demonstrated by strain shadows developed in the adjacent minerals; or had the crystal formed in an open cavity, it would have been imperfect at its point of attachment to the walls. This evidence then, complete perfect crystal development of the new mineral in a rock, furnishes one important criterion of replacement. Its value, however, varies with the kind of rock in which the complete crystals occur, and other qualifying factors must be determined before reasonably safe conclusions can be reached.

When dealing with igneous rocks it must first be ascertained with certainty that the mineral is really secondary, introduced after consolidation, and crystalline form, alone, can be used as a criterion only with those minerals not known to form under igneous conditions.

With other more typical ore minerals the problem is somewhat involved, since, under certain conditions, though rarely, they may be original. But if it is found that the mineral is thickly disseminated

in the neighborhood of a crack and is not prominent in the normal rock, the fact of its secondary nature is established, and its relations to other grains may then be utilized to determine replacement; or, if the mineral cuts or intersects a quartz phenocryst in acid lavas, the criterion is of value, since quartz is here an early crystallization.

Metamorphic rocks present some difficulty, because even the original nature of the rock itself is obscure in many cases, and, unless the secondary minerals were introduced after the change and transformation were effected, very few indications will be found by which their origin may be determined.

But, if the new crystals intersect typical metamorphic minerals, or cut across the schistosity, and are entirely free from strain and fracture, it is reasonably certain that deformation of the rock was produced prior to mineralization, and the usual criteria may then be applied.

With the sedimentaries the problem is simpler. Most ore minerals are too soft and friable to resist the extreme mechanical attrition during the formation of the sands and rock, and it is sufficient to prove only that the mineral has not grown by forcing the layers or bands of rock apart, but has really formed at the expense of the rock material.

Another criterion of value lies in the preservation of original rock structures. A dolomitic limestone, for instance, may be replaced by silica and still preserve the minute, rhombic, dolomite crystals in their original form; yet polarized light will show that they are now merely a quartz aggregate.

Still another microscopic criterion that is a rather definite indication of replacement is the presence of perfect pyrite crystals or of quartz crystals, surrounded by a mass of the same or other ore minerals. The inclosed crystals may or may not be due to replacement, but the surrounding mass proves the gradual substitution of ore for rock. It must be demonstrated, however, that the deposit is not a magmatic segregation.

Suppose now we take up another problem connected with mining in the prospect stage or in early development. To vary the treatment, this discussion will apply more particularly to the fissure-vein type of deposit. Let us assume that a shaft has been sunk on the vein and that conditions have been exposed somewhat below the actual outcrop. A question of vital importance to the success of the mine, and one often submitted to the petrologist for an opinion, is the probability of extension with depth. What sort of conditions or what mineralogical changes may be expected at lower levels?

The factors to be determined and the questions which depend for their solution upon observations of mineralogic relationship and structural characteristics of the vein, are, first, the nature of the primary mineralization, and, second, the probable extent of enrichment through secondary processes.

Primary ore, while reasonably certain to extend to great depth, is not always of commercial grade, and pay ore is more likely to result from a reworking of the deposit by later solutions. Descending surface waters penetrate the vein and cause changes in the upper zone through oxidation and solution, with redeposition as secondary sulphides at lower levels.

The original formation of a vein is by no means the end of the operation; alteration of some kind is always going on. It is doubtful if any marked change could take place, however, without the presence of water, and, unless aqueous solutions have free access, this change is very slow. It is evident, then, that the extent of alteration depends largely upon the character of the vein material as regards permeability to solutions. Another factor of importance is the mineralogic nature of the gangue. For example, a gangue prominently calcic retards secondary enrichment, probably through neutralizing the active organic or mineral acids in solution.

Microscopic examination reveals all this, sometimes clearly, sometimes less definitely; but, even if an original mineral be wholly removed, traces often remain, or characteristic alteration products are produced by which the original may be identified. Permeability of vein and wall rock is determined by evidence of brecciation, fracture, or distortion of the constituents.

Figs. 1 to 5 illustrate the mineralogic changes that may be found in a vein from the surface downward. The sections were made from samples from the Otter vein, Patagonia district, Ariz.

Most minerals are known to be capable of formation under many different agencies, but there are a few which, by their very presence in an ore deposit, serve to indicate certain definite physical conditions, and, where these are present, conclusions as to genesis are simplified.

Tourmaline, for example, is not known to form from aqueous solutions in the ordinary way, but is a typical pneumatolytic mineral which has been formed from vapors and under high temperature. Garnet is another, but it is particularly indicative of contact metamorphism.

Ores of this type, formed by deposition from gases and vapors, usually above the critical temperature and pressure of water given

off by a rock magma during solidification under deep-seated conditions, are rarely of economic importance, and, where their original character is still preserved, with primary sulphides at or near the surface, it is rather useless to look deeper for profitable ore.

In a study of any class of ore deposit an examination of the associated rocks is always necessary. Wherever mineralization has been extensive enough to form profitable deposits the rocks will be found considerably altered, not by ordinary surface change, weathering and oxidation, but through reactions occasioned by the mineral-bearing solutions themselves. Where rocks are fresh, mineral deposits will not, as a rule, be found worth developing.

Figs. 6 and 7 are micro-sections illustrating rock alteration.

In discussing the field of usefulness of the microscope during active mining operations it will be evident that the problems are of a somewhat different nature from those we have been considering. Questions of genesis and origin are no longer of real economic importance. The ore is exposed to some depth, and has been explored along the various levels and has been blocked out. Its quantity can be estimated with some accuracy and its metallic content ascertained by systematic sampling and assay; and since the zone of secondary enrichment may have been passed through, the primary mineralization may be exposed for study.

The success with which microscopic methods may be used to aid in solving the new problems is, however, just as definite, but in this case the work is more one of identification than one of interpretation. The structural geology of the district and of the mine become important factors.

The most economical system of stoping, the proper location for shafts and tunnels, the distance between levels and other purely engineering problems depend upon geologic structure, and upon the nature of ore and rock formations.

These factors are best determined by a careful survey of accessible workings and the compilation of accurate mine maps showing all geologic features. A certain dike, fault, or ore body, for example, may be plotted and correlated through similarity of microscopic characteristics with another exposure on a different level. Its strike and dip can then be computed, and its probable location at a desired point determined.

With these data the miner is enabled to reach the particular formation with the least possible excavation; or avoid it, should this be necessary. But, if the identification be faulty and general appearance alone be depended upon, it may happen that two separate and distinct

FIG. 1.—VEIN QUARTZ NEAR OUTCROP, SHOWING FRACTURE IN VEIN INDICATING DYNAMIC STRESS, RECEMENTED BY SECONDARY QUARTZ AND HEMATITE. × 35.

FIG. 2.—VEIN COMPOSITION AT DEPTH OF 20 FT. REMNANTS OF PRIMARY CHALCOPYRITE SURROUNDED BY SECONDARY HEMATITE. × 35.

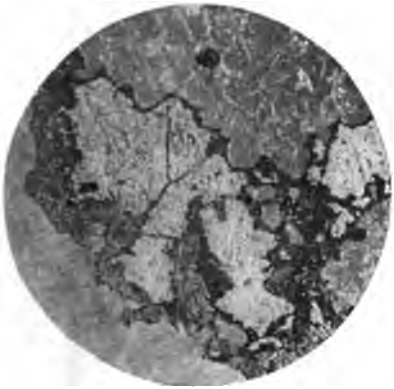


FIG. 3.—VEIN MATTER AT DEPTH OF 40 FT. PRIMARY CHALCOPYRITE PROMINENT, SURROUNDED BY RIM OF OXIDES. × 35.

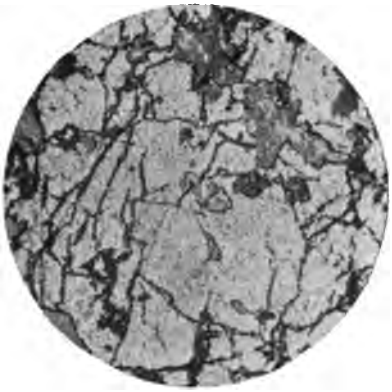


FIG. 4.—VEIN AT 50 FT. BENEATH THE SURFACE. SHOWS ABUNDANCE OF CHALCOPYRITE GREATLY FRACTURED, WITH SLIGHT ENVELOPE OF OXIDES. × 35.

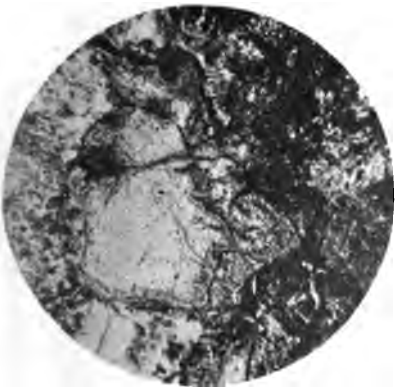


FIG. 5.—VEIN AT DEPTH OF 60 FT. PRIMARY CHALCOPYRITE SURROUNDED BY SECONDARY SULPHIDE, COVELLITE, AND AN OUTER RIM OF OXIDES; EVIDENTLY THE BEGINNING OF SECONDARY SULPHIDE ZONE. × 35.

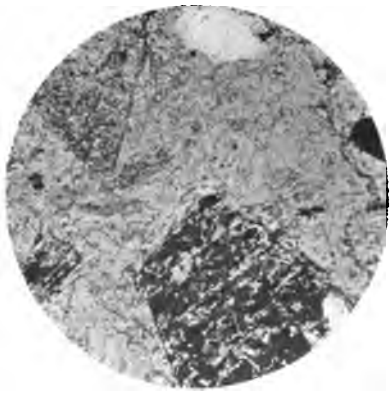


FIG. 6.—QUARTZ-PORPHYRY, GROUND-MASS DEVITRIFIED, BIOTITE CHANGED TO CHLORITE AND FELDSPARS KAOLINIZED. × 35.

FIG. 7.—SYENITE SHOWING MICROSCOPIC CRUSHED ZONE. × 35.

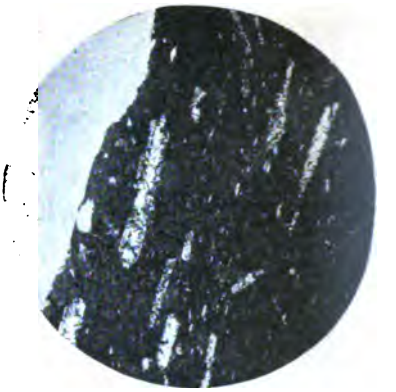


FIG. 9.—MICRO-SECTION OF A 2-IN. STRINGER AS SHOWN IN FIG. 8. FELDSPARS SHOW IDIOMORPHIC DEVELOPMENT, THE GROUND-MASS MERELY A BASIC GLASS; FLOW STRUCTURE MARKED. × 35.

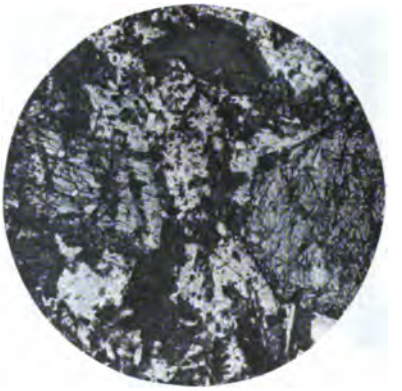


FIG. 10.—MICRO-SECTION FROM A 20-FT. DIKE OF TYPICAL CAMPTONITE INTRUSIVE IN THE FRANKLIN LIMESTONE AT FRANKLIN FURNACE, N. J. AUGITE, BIOTITE, AND TITANITE IN A FELDSPAR AGGREGATE. × 35.

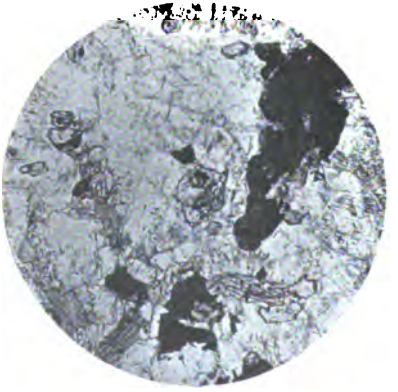


FIG. 11.—SECTION OF MICRO-DIORITE FROM SOUTHERN ARIZONA. HORNBLENDE, AUGITE, APATITE, AND MAGNETITE IN A FELDSPAR GROUND-MASS. × 35.

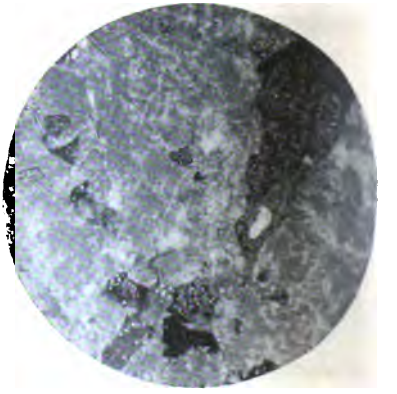


FIG. 12.—SAME AREA AS FIG. 11 BY SURFACE ILLUMINATION, SHOWING CRYSTAL OF NATIVE SILVER IN THE MAGNETITE. × 35.

features will be assumed to be identical. Location on this basis may prove to be in error by hundreds of feet. These problems become particularly important in regions where the structure is complex.

Figs. 8, 9, and 10 illustrate changes in rock texture produced through differences in the rate of cooling.

The identification of rocks and ores, then, is one of the fields of usefulness for the microscope in the operation of producing and developed mines, and it was the accuracy with which rocks could be identified and correlated by petrographic methods that led to one of the first practical applications of it in an actual mining problem.

FIG. 8.—STRINGERS OR OFFSHOOTS FROM THE INTRUSIVE MASS PENETRATING CRACKS IN THE LIMESTONE ADJACENT TO THE MAIN DIKE.

The particular question involved related to the identity of certain formations in the Eureka district, Nevada, and to the definition of the terms "vein" and "lode" as used in the U. S. Mining Statutes.

A body of mineralized limestone in Ruby Hill, near Eureka, had been located and mining operations conducted for some time. Unfortunately, a dispute arose. One company, which had followed down a more or less connected vein within the limestone, claimed ownership of a certain mass of ore on the 5th level, by virtue of the law of extra-lateral rights; the other company contended that the limestone itself was a lode as defined by the statute, and therefore

that they were entitled to that portion of the ore located on their side of a compromise line between the two claims.

During litigation it developed that a definite foot-wall of quartzite formed one boundary of the mineralized zone and that a shale and limestone formed a definite hanging-wall. The rock embraced between the two differed in its characteristics from all other limestone in the vicinity. It was extensively altered; crushed, brecciated, and recemented by calcareous matter and silver-lead ores. No traces of stratification remained.

These facts, demonstrated by microscopic methods, helped to establish that the mineralized Prospect Mountain limestone itself constituted a lode or vein of ore in place.

Besides the usual results to be derived from this sort of work, quite unexpected facts of profitable import are sometimes brought to light. A case that came to notice not long ago serves to illustrate the point:

Having had occasion to determine the nature of a mineral occurrence in southern Arizona, a microscopic investigation of the ore was undertaken. Samples of the neighboring rocks also were taken, to help in the solution. One of these proved to be a diorite, of normal appearance at first sight, but, on more careful examination, it was discovered that, throughout the rock were some peculiar opaque white crystals, perfectly developed, and generally surrounded by magnetite. They were identified as native silver. An assay of the rock, after careful sampling, gave an average silver content of 7.5 oz. per ton. The silver value might easily have been overlooked except for its accidental discovery in thin sections. (See Figs. 11 and 12.)

The last phase of the subject which will be considered relates to metallurgical treatment of the ore after reaching the surface. In many cases the success of a mine depends on the solution of this very problem, how can the ore be most economically treated to extract its values. Here again the microscope may profitably be utilized to assist in the solution.

The nature of the work differs from either of the two previous classes. The important things to be determined relate to the texture of the ore, the nature of the mineral aggregate, the size of grain; in short, to the answer to the question how, and not why, are the minerals related to one another.

For example, let us consider a recent milling problem at a Canadian gold mine. Upon microscopic examination of the ore it was found that most of the gold was closely associated with pyrite, as an inclusion or intergrowth, in a finely divided state. An appreciable proportion, however, occurred in the quartz gangue separate from

other minerals. It was justifiable to anticipate that, with this proportion, amalgamation would accomplish a good recovery, but that, with the other and principal part of the gold, amalgamation would prove ineffective without extremely fine grinding. Hence a combination with the cyanide method suggested itself as a solution of the problem.

Actual mill tests along this line proved the value of these observations. A preliminary study might easily have saved a considerable amount of money spent in unprofitable experiments.

Or take the case of disseminated iron in schists and gneisses, a class of ores that is coming into prominence. Their value and importance lie wholly in the ease of concentration through magnetic separation. But if, instead of the normal magnetite, part of the iron is present as martite, a mineral which is only slightly magnetic and yet which resembles magnetite closely in general appearance, it may readily be seen how the ore might prove a very unprofitable thing to handle.

Polished sections studied with vertical illumination would reveal the mineral character and would suggest the difficulty of concentration, notwithstanding the fact that a chemical analysis might show a favorable percentage of iron.

When deciding upon methods of concentration for any ore, microscopic examination will, in general, indicate which process will be most likely to give the best results. If two or more components are found to be very closely associated and minutely intergrown, chemical treatment, as a rule, will prove most effective, since fine grinding usually introduces difficulties; while an ore with a comparatively coarse-grained fabric will prove more susceptible to mechanical separation.

One recent development in the preparation of ores for the market, the sintering of fine concentrates, flue dust, and other by-products to render them amenable to furnace treatment, is worthy of notice. Many devices have been invented to accomplish this result, and, when comparing the qualities of the sintered product from the various processes to determine the most available treatment for a certain class of material, microscopic characteristics will be found of no little value in arriving at conclusions.

In this way may be determined, for instance, whether the material is sufficiently porous to permit the ready access of furnace gases, or whether the cohesiveness of the product is due to the formation of refractory silicates which would retard reducing action.

The study of mattes, slags, and speiss is another field, but one in which the problems are within the province of the metallurgist rather

than the mining engineer; nevertheless it is an unusual problem that may not be at least simplified by microscopic investigation.

In conclusion, it may be of interest to note that petrographic methods were first applied less than 60 years ago, and, when we realize the great development of the science and its wide application to various lines of research since Dr. Sorby in 1858 published his first paper on the microscopic structure of crystals and of rocks, the possibilities of future development may be the better appreciated.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.
[SUBJECT TO REVISION.]

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# The Application of Electric Energy to Mining in the Cœur d'Alenes.

BY J. B. FISKEN,* POST FALLS, IDAHO.

(Butte Meeting, August, 1913.)

BEFORE touching upon the immediate subject of this paper a few facts of a historical nature as to the general application of electricity to mining might be of interest.

As to what was actually the first application of electricity to mining, opinions will no doubt always differ. My experience does not go back far enough, nor are the records to which I have had access of any value as covering this point.

Sydney F. Walker, a man whose pioneer experience in the application of electricity to mining purposes dates back as far as 1876, gave an interesting *résumé* of the subject in an address before the South Wales branch of the Association of Mining Electrical Engineers at its 1912 session, and, unless his statements are shown to be wrong, they should be accepted as facts. According to Mr. Walker, the ignition of charges of blasting powder was the first operation in mining to be accomplished with the aid of electricity. Very frequently this energy was furnished by a static machine and occasionally by a magneto generator. Later, this was followed by the application of electricity to bell signals, and we are told that this application was looked upon skeptically and regarded as an expensive experiment by the mining managers of those early days. At the Paris Exposition of 1878, there was exhibited an electric lamp, which suggested future possibilities to the mining engineers of those days. The first successful application of electric lighting to underground excavations that I have been able to find any record of is that of M. Blavier. In the year 1879 he installed two Serrin lamps and successfully illuminated one of the large underground chambers in the Angers slate quarries. Another early application of electricity to the lighting of mines is that in the large chambers in the salt mines of Maros-Ujvár, in Hungary; these have been regularly and successfully lighted elec-

* Non-member.

trically since 1880. About this time the incandescent lamp became a commercial success and its application to the lighting of underground workings became general.

The first electric mine locomotive of which I have been able to find any record was used in Germany in 1882 in the royal coal mine at Zaukeroda, in Saxony. This application demonstrated that electrical energy could be applied with profit in this department of mining. Several years elapsed, however, before any application of electrical energy to mine haulage was made in this country, but about 1888 an electric locomotive, for strictly mine use, was constructed by W. M. Schlesinger, of the Union Electric Coal Co., in Pennsylvania.

In July, 1888, the first electric hoist to be used successfully in this country, at least, if not in the world, was installed by the Aspen Mining & Smelting Co. in its mine at Aspen, Colo. In this case the motor was a 7.5-h-p., 500-volt street railway motor built by the Sprague Electric Railway & Motor Co. It was used for hoisting on an incline, and the venture was so successful that another similar outfit was installed in the same year.

The first general application of electrical energy to mining purposes was probably in 1888, when the electric power plant in the Comstock lode, Virginia City, Nev., was put in commission. It is interesting to note that in this case there were six 120-h-p. generators, each generator being connected independently to an 80-h-p. motor, all six motors being belted to the same shaft. It is also interesting to note that the generators were constant current and the motors series wound.

The year 1889 brought numerous applications of electric energy to mining purposes. Among other installations may be noted those at the Ashland (iron) mine, at Ironwood, Mich.; the Calumet and Hecla; the Forbes Reef Gold Mining Co., Transvaal, South Africa; and the Hillside Coal Co., Scranton, Pa.

In 1890 the application of electrical energy to mining purposes became more general, and in this year may be noted the first installation in the Cœur d'Alêne country. But the industry was waiting for the time to come when long-distance transmission would be an accomplished fact. Prior to this time there had been various short transmissions. The first of these for mining purposes was in the Loire valley, France, at the works of Compagnie de la Ferronière. This was successfully accomplished prior to the year 1881, and I much regret that lack of time has prevented me from obtaining full details of this interesting installation.

In 1889 a transmission system was installed in France between Bourganeuf and Les Jarrauds, 9 miles distant, where there was an available water power. The capacity of the generating plant was 130 h-p., the transmission being single phase at a voltage between 3,550 and 3,750. The generator and motor were identical and were separately excited. In the same year there was installed, also in France, for the Chevrant Paper Mills, at Domène, a transmission line of 3 miles. The capacity of the generating plant was 300 h-p., the transmission being single phase at a voltage of 2,850. While these transmissions were not used for mining purposes, they are interesting on account of being among the pioneer lines.

In the fall of 1890 the very interesting transmission line of the Telluride Power Co. was put in successful operation to supply power to the Gold King mine at Telluride, Colo. The length of the transmission line was 2.6 miles. The capacity of the generating plant was 100 h-p., though provision was made for an ultimate capacity of 800 h-p. The motor, a synchronous machine, was like the generator in every respect. The voltage of the transmission was 3,000.

About the same time there was installed on Rock Creek, El Dorado county, Cal., by the American River Syndicate, a plant to supply power to its mine. The length of this transmission was 2 miles. The plant had an ultimate hydraulic capacity of 130 h-p. and consisted of a Brush generator of 100 h-p. and a 70-h-p. Brush motor. The transmission was what was known as high tension in those days, probably about 2,000 volts.

The literature published about this time, 1889 and 1890, furnishes intensely interesting reading to the electrical engineers of this twentieth century. Late in 1889 Mr. Edison published in the *North American Review* his famous article condemning the use of alternating-current systems of high voltage (probably not in excess of 2,300 volts). This article created a sensation at the time, and was bitterly resented by the advocates of the alternating-current systems of transmission and distribution. B. E. Sunny, at a meeting of the Chicago Electric Club in November, 1889, criticized the Edison article, and the following resolution was carried:

"*Resolved*, That it is the sense of the Chicago Electric Club, that the objections to and the general condemnation of the high pressure systems of lighting set forth by Mr. Edison in an article in the November number of the *North American Review* are not sustained, and that the growth and popularity of the service furnished by this class of currents are in themselves a contradiction of Mr. Edison's conclusions."

Where would we be to-day if Mr. Edison's contentions had been taken seriously?

Lemuel William Serrill, M.E., in an article on long-distance power transmission published in the *Electrical World* of Nov. 16, 1889, says:

"The constant potential method of distributing power will, in the present state of the art, give better satisfaction than either the constant current or alternating systems, although much may be expected from both of these, especially the latter, when brought to a better state of perfection through further development."

And, in arguing against the opinion held by Mr. Edison and many other prominent engineers, says:

"The day is not far distant when ten thousand volts will be handled as safely as the high pressures now employed in our steam engines."

In a paper on Operation of Electric Motors with High Tension Currents, read by Elmer A. Sperry before the Chicago Electric Club in February, 1890, mention is made of 3,000 volts as being the highest practicable voltage, and only direct-current systems are discussed, no reference being made to alternating-current systems.

Just one more quotation and we pass on. Robert B. Porter, the Superintendent of the Eleventh Census, in an open letter issued in December, 1889, says:

"The census of 1880 makes no mention of the industry of generating and distributing from central stations current for the uses of Light and Power. The investigation of this industry for the Eleventh Census will be the first official census report made on the subject in this or any other country."

The situation, then, with reference to the application of electric energy to mining purposes at the beginning of the last decade of the nineteenth century was about as follows: Several short transmission systems, none over 10 miles in length, were in use. Some of these were direct-current series systems, some direct-current multiple systems, and a few alternating-current multiple systems. In the alternating-current systems, the motors, as a rule, were of the synchronous type, although Tesla had invented his induction motor, and to a limited extent it had been applied.

Such was the situation when, in August, 1891, electrical engineering fraternities were startled by the news cabled from Frankfort, Germany, that the long-distance, high-tension transmission line between Lauffen and Frankfort, 112 miles apart, had been successfully tried out. Two hundred horse-power were transmitted during a rain storm at a potential of 13,000 volts and with entire success.

Immediately thereafter the voltage was increased by steps, and on Nov. 2, 1891, another cabled message conveyed the information that a successful test had been made with 27,000 volts difference of potential.

This was the first use made of the *drehstrom* (revolving current), or, as it is now called in this country, the three-phase system. This system was invented by Dolivo von Dobrowolsky and made possible the use of self-starting alternating-current motors. The general arrangement of the plant and the lines established a standard which does not differ materially from the standard in use to-day. The remarkable work done at that time by the European engineers made possible the electrification of the Cœur d'Alêne mines.

Now let us glance briefly at the history of the early applications of electricity in the Cœur d'Alêne country: I believe the first installation was the one for the Last Chance mine, in 1890. This consisted of two Edison bipolar generators of 20 kw. each, operated by water power, which were used to supply a three-wire lighting system in the mine. This plant was in regular operation until 1892, when, owing to financial difficulties, it was shut down. The mine itself went into the hands of a receiver in 1883, and Robert Cheyne got possession of the lighting plant. He moved it to a point near the dividing line between Wardner and Kellogg, where he operated it with water power under 140 ft. head during high-water seasons, and with steam at other times. He sold the plant to the Bunker Hill & Sullivan Co. in 1897, which later dismantled it.

The next installation was at the Black Bear mine and consisted of two Edison bipolar generators of 150 kw. capacity. This plant was used for operating Marvin drills, compressor pumps, and lights. I have so far been unable to find out when this plant was dismantled or what became of it.

Again we come back to the Last Chance mine, in which a second Edison plant was installed, with a capacity of 100 kw. This plant furnished power for Marvin drills and lights in the mine and also for lights in Page's Hotel, the Catholic church, and a few residences. This plant, I believe, passed into the hands of the Bunker Hill & Sullivan Co.

In 1892 there was installed in the power house above Burke a 250-h-p. Edison bipolar plant of 1,200 volts direct current, and the power from this plant was transmitted to Burke, where it was used for pumping purposes. This plant was in use till 1898, when the present plant was installed.

The first installation at the Standard mine was made in 1896, and consisted of a 35-kw. alternating-current generator and a Thomson-Houston 45-kw. 500-volt generator for operating the mine railway. The latter machine, I believe, is in operation yet, but I do not know what has become of the former.

In 1897 the Bunker Hill & Sullivan Co. moved the Last Chance plants, which it had acquired, to its power house, adjoining the mill, and in addition installed a 75-kw. 2,300-volt Monocyclic generator. Water power was used, under 360 ft. head, the water being flumed from Wardner. Later a Russell tandem-compound engine was installed to help during periods of low water. This was followed in 1898 by the installation of a 55-kw. 550-volt four-pole G. E. generator driven by a hydraulic turbine operating under 46 ft. head. This plant was used during the driving of the Kellogg tunnel, and later for general haulage in the mine and about the yards.

In this same year the 1,200-volt direct-current plant at the Burke power house was abandoned and a 300-kw. 2,300-volt 40-cycle generator was installed. It will be of interest to note that later the Federal Mining & Smelting Co. increased the speed of this machine to get a frequency of 60 cycles and now it is run in multiple with the Washington Water Power Co.'s plant.

The next installation was made by the Bunker Hill & Sullivan Co., which in 1901 increased the capacity of its lighting plant by the addition of a single-phase 50-kw. 2,300-volt generator and a Russell engine similar to the one referred to above.

No further additions were made until Aug. 25, 1903, when the Washington Water Power Co. commenced furnishing service in the district, and since that time the additions have been too numerous to chronicle.

When the mines were first operated in the Cœur d'Alêne district, water power was used to a considerable extent, and the heavily timbered hillsides afforded an abundance of fuel when water power was not available. But with the increasing development of the district came the necessity of using more water for concentrating purposes and timber for mining timbers, and the operators were face to face with the alternatives of shipping in coal for making steam or securing electric power from some source.

A group of the operators in the Cœur d'Alêne district decided to have electric power, and, in furtherance of this resolution, R. K. Neill, in behalf of those interested, on Aug. 1, 1900, received from Frederick Post a deed to the water power and the necessary real estate at Post Falls.

Plans were started looking towards the development of this power and the construction of a transmission line, but it early became apparent to those interested that their business was mining, not supplying electric power, and in January, 1902, the Post Falls property was

transferred to the Washington Water Power Co. under an agreement that it would supply power to the mines.

No time was lost in carrying out this agreement and the construction of the transmission line was commenced immediately. As it was a physical impossibility to develop the Post Falls power within the time agreed upon, two 2,250-kw. generators were installed with the necessary step-up transformers at Spokane, and on Aug. 25, 1903, the line was put in commission as far as Kellogg, thus supplying power to the Sweeney mill, and the Bunker Hill & Sullivan mine and mill. Work was then pushed on the remaining portion of the line, and early in November the Burke sub-station at the terminus of the line was cut in.

Meantime work had been started on the development at Post Falls, and on July 12, 1906, that station was put in commission and run in multiple with Spokane.

It is interesting to note that the transmission line to the Cœur d'Alênes was one of the earliest long-distance lines in this or any other country to furnish regular service for 24 hours a day.

The apparatus of the generating and receiving ends, and the line, were designed for 60,000 volts, but arrangements were made so that the plant could be operated at 45,000 volts. The latter voltage was carried at the start and, in fact, till Feb. 2, 1905, when the change was made to 60,000 volts, at which voltage it has since remained.

In designing the line the problems were all new, and there was an entire lack of standard construction of any kind.

The business grew; our friends in the Cœur d'Alênes country seemed to like our power, and they liked it to such an extent that it became necessary in 1907 to build an additional line as far as Cataldo.

The first line was built from Spokane, as already stated, around the south end of Lake Cœur d'Alêne and across the Indian Reservation, but the second line was built from Post Falls and the shorter route through the Fourth of July Canyon was followed.

In 1909 the second line was extended from Cataldo to the Sisters mine, where the present switching station is built. To enable greater facility in transferring the load from one line to the other, the switching station at Cataldo, in which oil switches were installed, was built, and air switches were installed at different points to enable the individual loads to be so transferred.

In building the second line, which has nearly four times the capacity of the first, an effort was made to eliminate from it those features which had been found to be undesirable in the first line.

To guard as far as possible from interruptions to the service, these lines are all patrolled once a week, and more frequently if necessary. Patrol stations with a patrolman at each station are maintained on each line, at the following points: Post Falls; Opportunity, a short distance east of Spokane; Rockford Junction, about 6 miles northeast of Rockford; Medimont; Wolf Lodge; Cataldo; Kellogg; Mullan Junction; and Mullan. At each of the patrol stations an air switch is installed, for the purpose of testing the line to locate any trouble that may come on.

Considerable difficulty was experienced at first, owing to the lack of inertia in the generating plant. It was found that a mining load was one in which the changes of load were large and sudden, and it is hardly necessary in a gathering of mining men to point out how very necessary a constant speed is to efficient concentration. The memory of the old hoist at the Hecla mine will not be forgotten soon by those who were almost sweating blood, I might say, to try and keep the speed steady.

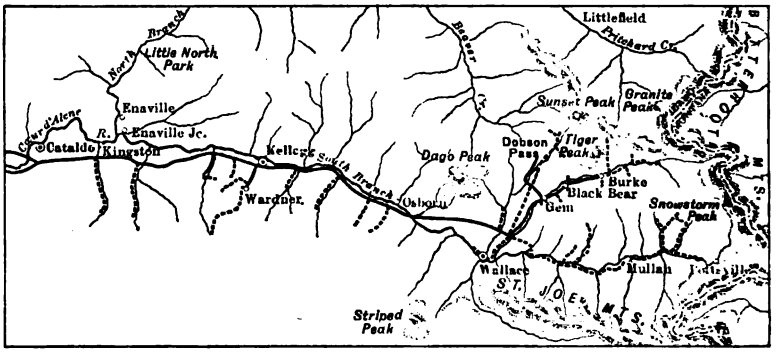


FIG. 1.—MAP SHOWING ELECTRIC TRANSMISSION LINES OF THE WASHINGTON WATER POWER CO.

As most of you know, the speed of an alternating-current motor varies almost directly with the speed of the generator; and, when the Hecla hoist would start at the same time that a compressor picked up load somewhere else, a howl would go up from the mill foremen, which was fully justified, and the restrictions placed by the Washington Water Power Co. on the type of motor to be used for hoisting were also fully justified.

The installation of the splendid apparatus now installed for hoisting at the Hecla relieved the situation, but the greatest relief has come from the fact that the Cœur d'Alêne load is now handled from Little Falls and Post Falls, connected in multiple. The stored energy

in, or the fly wheel effect of, the rotating parts in these two plants is so great that all restrictions as to the types of motors to be used have been removed. The plant at Spokane, being relatively a small one as compared with the others, is not now used for loads outside of Spokane.

The map, Fig. 1, gives some idea of the development of the Washington Water Power Co.'s system in the Cœur d'Alênes. The full line represents the lines as they were in the beginning of 1904, and the dotted lines show the extensions which have been added since that time.

In 1903 there were seven sub-stations, with a total transformer capacity of 3,350 kw. To-day there are eleven sub-stations and a transformer capacity of 13,900 kw.

The original plan followed was to step in each sub-station from 60,000 volts to 2,300 volts and to have sub-stations at frequent intervals; the practice now is to install fewer and larger sub-stations, and from each sub-station serve a larger territory. This result is achieved by using 6,900 volts from the sub-stations and stepping down with open air transformers to 2,300 volts at the customer's plant.

The uses to which electric energy is applied in mining are many, the principal ones being illumination, milling, hoisting, air compressing, pumping, ventilating, hauling, signaling, firing shots, shop power, etc. Some progress has been made towards reheating compressed air, but this as yet is in the experimental stage. Probably the greatest increase in the future will be in hoisting. In addition to the Hecla hoist, already referred to, electric hoists of 150 h-p. each are in successful operation in the Hercules, Gold Hunter, and Bunker Hill and Sullivan mines, and the operators of several other mines are at present investigating the subject. This follows as a natural sequence to the greater depth being obtained in mines below the lowest tunnel level.

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# Development of the Basic-Lined Converter for Copper Mattes.

BY E. P. MATHEWSON, ANACONDA, MONT.

(Butte Meeting, August, 1913.)

In a discussion of a paper on "The Basic Process as Applied to Copper Smelting," by Percy C. Gilchrist, read before the Society of Chemical Industry, London, Jan. 5, 1891,[1] Prof. W. C. Roberts-Austen asked Mr. Gilchrist whether he thought that the substitution of a basic lining for acid lining in the Manhès process would afford anything like the service which it had been shown to render in the metallurgy of iron.

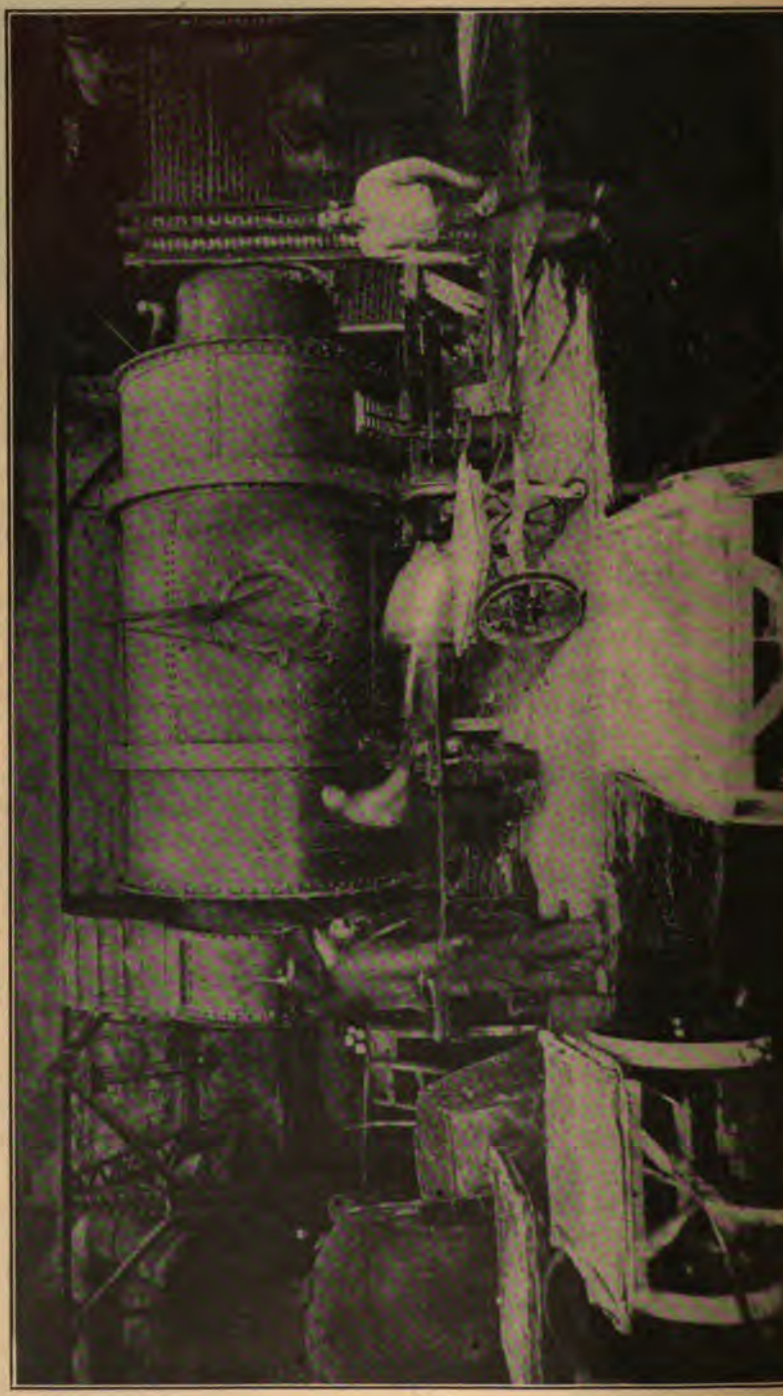
Claude Vautin stated that he had experimented for over two years with basic linings for Bessemer converters for copper mattes at Cobar, but had given up the attempt on the score of cost. Mr. Gilchrist in his reply stated that he did not believe in applying any system of Bessemerizing to copper.

About the time of the presentation of the paper mentioned, Herman Keller, Superintendent of the Parrott smelter, in Butte, was experimenting on a large scale with converters lined with magnesite brick. He gave up the idea on account of the cost of linings and because no particular advantage was observed. My belief is that his tuyeres and converters were too small.

A short time afterward similar experiments were tried at the Great Falls plant of the Boston & Montana Co., and at the Old Works of the Anaconda Copper Mining Co. in Anaconda. These were abandoned on the score of cost and the lack of advantages. The same cause of failure, in my opinion, holds here. The Anaconda Copper Mining Co., however, adopted the magnesite brick lining for its tilting casting machines.

Fig. 1 is a view of the first tilting furnace casting copper. It was made from an old Brueckner roasting cylinder. Fig. 2 shows an elevation and vertical and horizontal longitudinal sections of the second tilting furnace, built in 1897.

[1] *Journal of the Society of Chemical Industry*, vol. x., No. 1, pp. 4 to 16 (Jan. 31, 1891).



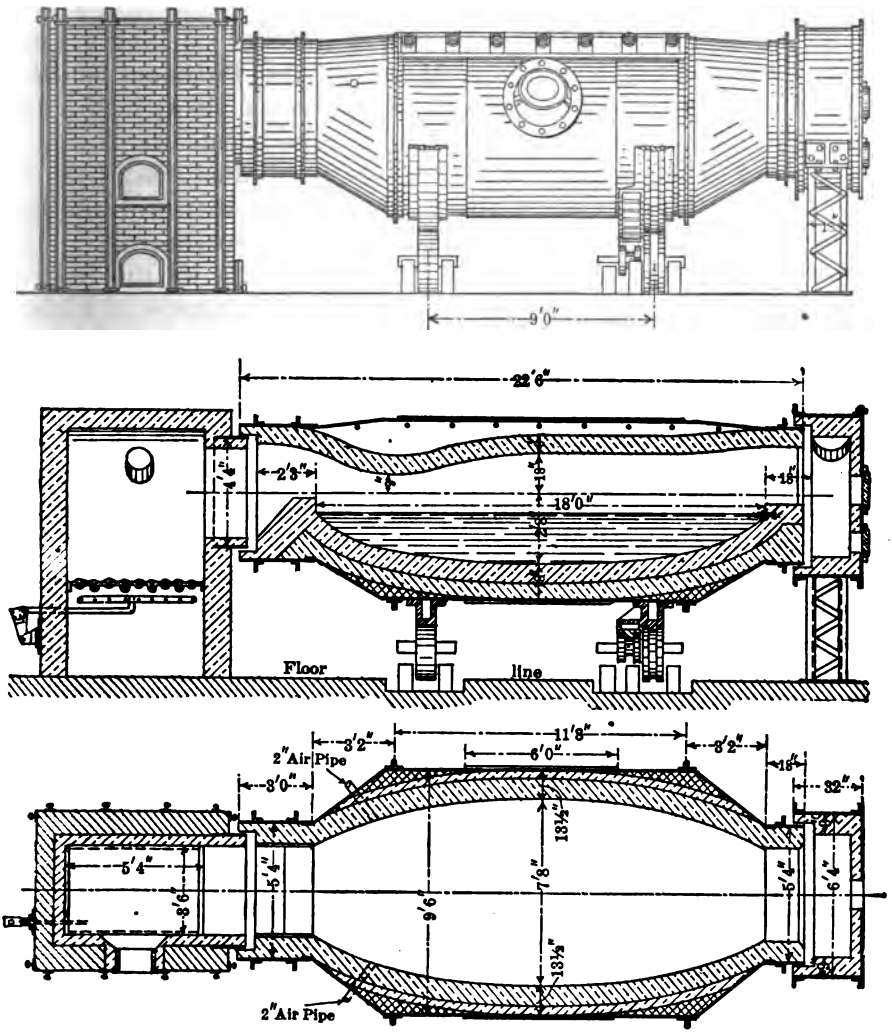



FIG. 2.—ELEVATION AND SECTIONS OF THE SECOND TILTING FURNACE OF THE ANACONDA COPPER MINING CO.

E. A. C. Smith, while temporarily in charge of these casting machines, tried the experiment of blowing the copper in the casting machines, but the return of the head of the department put a stop to the experiment, and Mr. Smith put the idea away till a more favorable opportunity presented itself.

About 1903 Baggalley began his experiments of smelting ore in a basic-lined converter in Butte, Mont., at the Pittsmont smelter. He was followed by Knudsen at the Sulitjelma plant, in Norway, in 1907. About the year 1905 similar experiments were tried at several smelters, notably the plant of the United States Smelting Co., at Midvale, Utah. Mr. Smith, who was then with the Baltimore Copper Co., found his opportunity to experiment, and his superintendent and manager, Mr. Pierce, gave him all the help in his power, the result being the construction at Baltimore of a basic-lined converter for leady copper mattes, along the lines of the old tilting anode furnaces of Anaconda. The experiment gave promising results, so the American Smelting & Refining Co. took up the process and introduced it with success on leady copper mattes at its lead refineries at Perth Amboy and Omaha. Then Messrs. Smith and Pierce persuaded the company to try it on straight copper mattes at the Garfield, Utah, plant.

In the meantime the Anaconda Copper Mining Co., at the Washoe Reduction Works, Anaconda, lined a shell of the standard trough pattern with magnesite brick. The results were excellent, so they gradually replaced all the acid lining with magnesite brick. At Great Falls the same company's experts developed a large converter along the lines of the upright shell, and, as this type is easier to build and keep in repair, it has become the standard during the past year. Practically all the Bessemerizing of copper mattes in the United States is now done in basic-lined converters. The main points to be observed for successful operation of the basic linings are: not to exceed a temperature of 2,100° F.; not to have tuyeres smaller than 1.25 in. (1.5 in. is the preferred size); to drive in punch rods the full size of the tuyere opening, immediately after pouring copper; to maintain in the converter as large a mass of matte and slag as possible to prevent sudden changes in temperature and overheating of the lining; to employ slag containing preferably about 25 per cent. of silica.

*Magnesia in Basic-Lined Converter Slags.*—The following test was made with a view to finding out whether the cutting action of converter slag on a magnesia brick lining bore any relation to the silica content of the slag.

Thirteen slags were selected from the daily samples sent to the

laboratory. These varied in silica from 22.4 per cent. to 37.8 per cent. and were carefully analyzed for MgO. The results show no relation between silica and magnesia content and are given in the following table:

TABLE I.—*Analyses of Slags.*

| Cu. | $SiO_2$. | FeO. | CaO. | MgO. |
|---|---|---|---|---|
| Per Cent. | Per Cent. | Per Cent. | Per Cent. | Per Cent. |
| 2.2 | 37.8 | 46.3 | 1.1 | 0.6 |
| 2.2 | 37.2 | 46.6 | 1.0 | 0.6 |
| 3.80 | 35.1 | 48.3 | 1.0 | 0.6 |
| 2.80 | 34.5 | 49.3 | 1.0 | 0.6 |
| 3.20 | 33.5 | 50.3 | 1.0 | 0.5 |
| 3.80 | 31.6 | 51.4 | 1.4 | 0.5 |
| 1.80 | 30.8 | 52.5 | 1.3 | 0.8 |
| 2.20 | 28.9 | 56.0 | 1.0 | 0.5 |
| 2.40 | 28.0 | 56.5 | 0.9 | 0.5 |
| 3.00 | 25.8 | 56.6 | 1.0 | 0.5 |
| 4.60 | 24.6 | 56.2 | 1.0 | 0.7 |
| 4.60 | 22.4 | 58.3 | 1.3 | 0.7 |
| 4.20 | 23.4 | 58.1 | 1.1 | 0.4 |

To Smith and Pierce belongs the credit of taking a long-discarded idea and developing it into a successful process. The great advantages of the process are: decreased cost of lining; the ability to use large units in coverting, with consequent economies in labor, power, and repairs; neatness and cleanliness of plant, abolishing the danger, from dust, to the health of the lining crew.

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# An Assay for Corundum by Mechanical Analysis.

BY W. SPENCER HUTCHINSON, BOSTON, MASS.

(Butte Meeting, August, 1913.)

It is the purpose of this paper to describe a method used to determine the corundum contents of samples of hard crystalline gneiss containing both corundum and red garnet. A chemical analysis of the rock for alumina would not fix the amount of corundum because at least three other constituent minerals of the rock, mica, feldspar, and garnet, contain alumina combined with silica.

Following are the principal minerals found in the samples, together with their specific gravity and hardness as given by Dana:

| | Specific Gravity. | Hardness. |
|---|---|---|
| Quartz | 2.6 | 7 |
| Mica and biotite | 2.75 to 3.1 | 2.7 |
| Feldspar | 2.6 to 2.7 | 6 |
| Garnet | 3.15 to 4.3 | 6.5 to 7.5 |
| Corundum | 3.9 to 4.16 | 9 |

The corundum occurred in crystals of a gray or bluish gray color, in minute grains and coarser up to 0.5 in., and no massive or block corundum was observed. The larger crystals were commonly quite slender, well defined, and the normal hexagonal form clearly developed. They lay with the structural planes of the gneiss, and sometimes appeared prominently in relief on weathered surfaces. In some cases these crystals had suffered partial or total alteration to mica, margarite, or other soft minerals, a condition illustrated by sample D, which appeared to be studded abundantly with corundum on the flat faces. Garnet, a striking feature of nearly all the samples, was present in small rounded crystals of a pink to red color.

The samples were first broken to pass a No. 4 mesh screen. This was done with a hammer on an iron plate, particular care being taken to avoid the breaking of the crystals finer than need be to pass the screen. A portion of the sample weighing 500 g. was then taken and, with a little water added, was scoured round and round in an iron

bucket with a wooden block, until the coarse crystals of corun were cleaned of adhering pieces of mica. The bucket was the peatedly filled with water and much of the fine light mica floate The remainder of the sample was dried and sifted, making six between No. 4 and No. 14 mesh; each portion was picked ove hand and the pure corundum crystals separated and weighed. rejected remainder was then reunited with the fine portion o sample and all crushed to pass No. 30 mesh.

All was next concentrated by panning in a gold pan to prod garnet-corundum concentrate. The bright red color of the g served as a reliable indicator and made possible the production clean concentrate. The tailings were panned repeatedly, usuall or eight times, until the additional amount of concentrate recov was so small as to be negligible. This concentrate was assume contain practically all the corundum and garnet in the original ple—certainly all that could be recovered by any known mi treatment. It also contained a small proportion, one-fifth or sixth, of quartz and feldspar.

The dried concentrate was next put under an electromagnet. picked out the garnet; but, as in most of the samples the garnet present in amount several times the corundum, there was a rel increase in the proportion of quartz and feldspar in the residue, w still contained all the corundum. To determine the actual qua of the latter, it was separated by floating the grains of quartz feldspar in a heavy solution. The solution, of 3.1 specific gra was made up of a mixture of potassium iodide and mercuric io in water, and an apparatus was devised consisting of an upright tube with a bell top and a closed bottom, the lower 2 in. detach and connected by a rubber sleeve with a pinch cock. To this filled with the heavy solution, the residual corundum concentrate added and stirred in, allowed to settle, the pinch cock closed the clean corundum removed from the lower tube, washed, d and weighed.

The following is a tabulated statement of some results selected the samples tested:

| Sample. | A. | B. | C. | D |
|---|---|---|---|---|
| | Per Cent. | Per Cent. | Per Cent. | Per C |
| Coarse corundum | 1.2 | 0.6 | 3.8 | 0.6 |
| Fine corundum | 1.3 | 1.8 | 4.8 | 0.4 |
| Total corundum | 2.5 | 2.4 | 8.6 | 0.4 |
| Garnet | 6.4 | 8.0 | 4.9 | 1.1 |

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# The Kennedy Mining District, Nevada.

BY PAUL KLOPSTOCK, KENNEDY, NEV.

(Butte Meeting, August, 1913.)

THE Kennedy mining district is situated about 55 miles in a southerly direction from Winnemucca, and about the same distance southeast from Battle Mountain: two towns situated on the main line of the Southern Pacific railroad.

## *History.*

Prospectors were first attracted to the district by the reported discovery of gold by Chinamen in Cinnabar creek in the early 90's. The prospectors came from the rich placer workings in and about American canyon; but the new diggings were not found sufficiently rich to justify their being worked as placer properties, and the prospectors started work in a number of places, looking for the veins which had fed the canyon. In 1893, Charles E. Kennedy, a prospector who had been attracted by the reported discoveries in the vicinity, left Hawthorne and came to the present site of the town of Kennedy. Within a short time he discovered the vein known as the Imperial, and proceeded, with the aid of some friends, to develop it. The vein was readily traced for a distance of about a mile, and about $10,000 worth of high-grade ore was shipped to the Utah smelters. At about the same time the Lawler Brothers, who had been prospecting in the district for some time, located the "Gold Note group," which showed some extremely rich ore. Within a very short time they extracted from the Gold Note vein 150 tons of ore, which was shipped to the Utah smelters, and netted about $105 per ton above all expenses. Their good fortune attracted attention to the camp, and within a short time a town was laid out, with a population of about 500 people. It was named in honor of Kennedy, who had made the first discovery of pay ore in the district.

A great deal of work was done, many veins were discovered, and the camp assumed a prosperous position. George S. Nixon, at that time cashier of the First National Bank at Winnemucca, was attracted to the district through the steady shipment of ores to the

smelters, and with James Wardner, who had come down fro
Cœur d'Alêne region of Idaho, decided to install a mill for the c
treatment of the ores of the district, and especially of those fro
Imperial mine, on which they obtained an option. A 20-stam
(plate amalgamation) was built in 1895, designed to handle th
milling ores of the camp. The mill was in successful operati
about a year, during which time two other small ones wer
structed. All the time steady shipments of high-grade ore
going to the smelters, besides the 8,000 tons of ore treated
mills. No precise records of the total production of the camp
this time are available. It is generally estimated that from
when the camp was first discovered, up to the end of 1896, t
proximate returns were: from sale of ores to sampling work
smelters, $175,000; from milling, $100,000.

About this time it became evident throughout the district th
oxidized zone was very shallow and in no place extended belo
ft., at which depth a heavy sulphide ore was found. Inasmu
this experience became general, and the mills were unable t
the ores, the camp soon became depopulated and, with the exc
of a few prospects, active work almost ceased. It became e
that the mills had been built before there was enough ore in si
justify them; and also with almost entire disregard of the ch
of the ores to be treated.

In the beginning of 1900, A. E. Lasher, who at that tim
making a trip in Nevada, heard of the Kennedy district, a
making an examination, decided that sufficient ore had been e
in the district to justify both its proper development and the r
eling of the Imperial mill. He interested some of his friends
project, and a syndicate was soon formed, known as the Wynn-
syndicate, whose object was the purchase of several of the prop
inclusive of the Imperial mill. The sale was eventually nego
and a leaching plant was added to the 20 stamps; so that by t
of 1901 the plant was in operation and the camp again preser
lively appearance. The Gold Note group had been previousl
chased by J. A. Blossom, of Battle Mountain, and was shipping
grade ores, which had been cut in the Union tunnel at a depth
ft. This work, together with the Wynn-Lasher operations,
employment to about 100 men. The mill was working about 5
of ore per day, the production coming from the Klondike, C
and Borlasca mines. No effort was made at any time to mill
the Gold Note ores. Owing to metallurgical difficulties an
failure to keep the development work ahead of the mining, a
as to financial difficulties, the Wynn-Lasher syndicate was com

to suspend operations in 1905. A short time prior to this the Gold Note mine closed down because of litigation following the death of Mr. Blossom, and again the entire district was almost deserted. During the period last mentioned the Gold Note mine produced about $60,000 worth of high-grade shipping ore. The production from milling and other incidental sources did not exceed $150,000.

In the latter part of 1909 the district was brought to the attention of the writer, who, after several examinations, concluded that the district required a consolidation of the various interests in and about the camp and that, with proper development and equipment in and around the different mines, and a remodeled mill, the district would enter into a period of prosperity. In accordance with these views, efforts were made in the beginning of 1910 to effect the desired consolidation, which was eventually consummated in September, 1911, with the formation of the Gold Note Mining & Milling Co., which now controls about 75 per cent. of the mineralized territory of the district and owns the Imperial mill, buildings, etc.

At the present time this company is not operating on an extensive scale, but is confining itself to thoroughly resampling the workings on its properties (some 3.7 miles of underground work), and thoroughly testing the ores. Plans have been made for the complete overhauling of the plant, and it is expected by the early summer to employ about 100 men on the property.

*Geology.*

The town of Kennedy is situated in the center of the district, which runs for a distance of 5 miles north from the town, and 1.5 miles south. The district is about 1.5 miles wide, from east to west. From a commercial standpoint, the district is embraced practically in a belt of eruptive rocks which begins to the south of Gold Note mountain, and then runs north for a distance of 2.5 miles to Water canyon, where the porphyries are capped by a flow of basalt. Beyond Water canyon no extensive evidence of mineralization is evident. Quartz-porphyries, grano-diorite, gneiss, and diabase are mostly associated with the ore-bearing veins, while the southern portion of Gold Note mountain is chiefly composed of andesite.

In and close to the town of Kennedy the gneiss is very much in evidence; through it several large rhyolitic dikes have been intruded, causing considerable fracturing and shearing. An examination of a hand specimen of the rhyolite revealed particles of calcareous slate, a rock which is found outcropping some 3 miles south of the town of Kennedy. The dikes vary from 6 to 20 ft. in thickness and can be traced for two miles or more. Very often the material of the dike has been sufficiently mineralized to form good pay ore.

In and about the vicinity of the Gold Note mine and ru north, the quartz-porphyries and grano-diorite form the pre nating rocks. They appear to rest on the gneiss, and in places penetrated it. In places the quartz-porphyries are intrusive th the grano-diorite and diabase, and *vice versa*. This section c country has been greatly disturbed by a series of rhyolitic intru which run northwest. Along their planes of intrusion, and inci to them, the veins in the Gold Note group of mines are found. veins can be traced for distances ranging from 0.5 to 1.5 mi their strike, usually northwest. They dip to the south at a ranging from 30° to 65°. They vary in width from 1.5 to 5 f appear to have been formed after the rhyolitic intrusions or c dent with them. They traverse without any regularity the dif rocks encountered in the district, and pass from the porphyries grano-diorite and then through to the gneiss. Their walls are and well defined. Considerable shearing is indicated by the sli sides, and clay selvages between the gangue and the wall rocks

From work done in the Union tunnel on Gold Note mount cross-cut which has cut 5 of these veins in the 650 ft. it has bee and from which about 4,500 ft. of drifting has been done, it writer's belief that these veins and others (about 15 have been exp are the results of ore deposition incidental to the rhyolite. The lite in its upward movement met serious resistance and sprea causing fractures through the different overlying rocks, which b filled with the present vein material. Judging from their un strike, their dips should converge at a depth of about 400 ft. the present lowest level (450 ft.).

The gangue consists of silica, particles of rhyolite, and some c The whole is brecciated and carries a metallic content of abo per cent. of iron (both as the oxide, hematite, and the sul pyrite). Below the water-level a little zinc, copper, and lead sionally appear.

Some faulting is evident in a number of places; this, how causes no serious displacement. The largest throw is due to a fault which causes a displacement of one of the veins of 4 feet.

The chief faults are directly under the ravines and gulches an probably caused by replacement.

### *Character and Value of Ores.*

At the surface, and to depths ranging from 50 to 125 ft. on th of the veins, the ores are entirely oxidized; the following analy submitted as representative of these ores: Insoluble, 75; iron, 9 to 15; sulphur, 3 per cent.; Au, 0.75; Ag, 12 oz. The values

from $8 to $200 per ton, depending on the vein from which the ore may come, and the average value is about $14 per ton.

Beginning at the bottom of the oxidized zone, a secondary sulphide is found, caused chiefly through leaching and redeposition at and below this level. The secondary ores vary from 50 to 75 ft. in depth, below which the primary sulphides are encountered. The enriched ores carry less silica and more iron and sulphur, as is attested by the following analysis: Insoluble, from 30 to 35; iron, 30; sulphur, 25 per cent.; at times, small amounts of copper, lead, and zinc are found, usually accompanying high-grade ores. The values here range from $2 to $60 per ton and are somewhat bunchy.

The primary sulphides are found directly below the secondary sulphides and average 65 ft. below the bottom of the oxidized ores. Their average content is represented by the following analysis: Insoluble, 43; iron, 33.5; sulphur, 14 per cent.; Au, 0.52; Ag, 10 oz.; lead, trace; zinc, trace; copper, trace; although sometimes there are bunches of lead, copper, and zinc.

It has been noted that there is considerable connection between the values in the secondary sulphide zone and the fault planes; as an illustration, it has been observed, during repeated and systematic sampling, that the values were very bunchy directly under the gulches and in open ground. Where the vein was distant from the faults the values were uniformly higher.

### *Ore Treatment.*

The Imperial Mill.—This mill, the largest in the district, was constructed in the fall of 1894 and completed in the early part of 1895. It contains twenty 850-lb. stamps and plates for amalgamation. All were constructed by Fraser & Chalmers. An electrical department was added in 1896–97 in order to experiment with electro-amalgamation; this proved unsuccessful. In the fall of 1901 the mill was acquired by the Wynn-Lasher syndicate. A stone addition was built and a cyanide plant added, designed for a straight leaching process to handle the oxidized ores of the district. The treatment of these was as follows:

Slaked lime was fed to the ore in passing to the batteries, where a 1-lb. solution of KCN was used in crushing to 30 mesh. The ore next passed over plates; the sands and slimes were separated in cones. The slimes, passing on into leaching tanks, settled for three hours. The cleared solution was decanted and passed onward into zinc boxes, then to sump tanks, from which it was pumped into storage tanks and restandardized. The heads of the solution carried from $0.75 to $1.25 per ton in Au and Ag; the tailings, from 3 to

5 cents per ton; the extraction varied from 90 to 95 per cent. o
assay value of the ore. Only 0.01 per cent. of the values was l
the slime tailings; 60 per cent. remained in the solution, as no
was used.

The sands were leached for 24 hours in a solution of 4 lb. of
to the ton of water used and were then washed twice, the sol
passing through the zinc boxes after being filtered through th
tom of canvas cloth. The extraction in the zinc boxes was
the same as with the slimes.

When a clean-up was carried out, the zinc slimes were place
tank, washed in sulphuric acid solution, settled, and the acid sol
was removed. The slimes were washed twice, settled, dried, fl
and melted into bullion, which was not separated. It was the
either at Selby's or to the mint. The cyanide consumption
about 1 lb. per ton of ore treated.

A series of tests, made during the last few months on the sul
ores, demonstrates that fine grinding of the sands is required.
chanical agitators will be installed, together with a filter pres
the same method of treatment will be applied in general as is pra
at the Nevada Hills property, at Fairview, Nev.

The capacity of the mill will shortly be increased to 125 to
day, which is now justified by the established ore reserves
another small mill will be built to take care of the ore coming
leasing operations soon to be inaugurated in the district.

*General Remarks.*

The district, particularly the mineralized section of it, has
been opened by over 4.5 miles of underground workings. Mor
50 separate and distinct veins have been exposed, and have
proved for distances ranging from 1,000 ft. to 2 miles along
strike, and to depths ranging from 50 to 450 ft. on their dips.
veins vary from 1.5 to 7 ft. in width, and to date (February,
ore has been placed in sight which has been estimated to
200,000 tons. Considerable ore is on the different dumps, an
an active and intelligent campaign of development work there
reason why large bodies of good milling ore cannot be pr
blocked out, insuring a long life to the district, which should
active producer by the end of this year.

Mill construction will be started with the opening of
together with active leasing operations, and it is expected th
district, which has passed through so many difficulties, will
again become active, and join the ranks of the permanent pro
camps of Nevada.

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# Mining Cost Accounts of the Anaconda Copper Mining Co.

BY H. T. VAN ELLS, NEW YORK, N. Y.

(Butte Meeting, August, 1913.)

THE following is a brief description of the cost accounts in effect at the mines of the Anaconda Copper Mining Co.

The accompanying chart, Table I., shows the distribution of labor, materials, and redistributable accounts to the various operating and repair accounts.

## OPERATING.

### *Development.*

Owing to the fact that every mine has its peculiarities in formation and methods, the question of arriving at what charges should be included under the head of "Development" is one that has occasioned considerable thought and comment, and in the cost sheet of the Anaconda Copper Mining Co. this account has been sub-divided to show the expenses of breaking ground, tramming, timbering, shaft sinking, up-raising, and cutting stations and skip chutes. It has always been the policy of the company to charge the entire cost of this work to operations monthly, whether the mines are producing or not, also the cost of the opening up of new mines; and only surface construction and additions are charged to capital, excepting original machinery installed for operating pumping plants. Explanations of the charges to the various sub-accounts follow:

*Breaking Ground.*—Work performed in driving cross-cuts from the station, or cross-cuts from the drifts, and diamond-drill work, for the purpose of discovering or developing ore bodies.

*Tramming.*—Cost of tramming waste produced by cross-cuts mentioned above. When waste is hoisted to another level to be used for filling, the charge to this account stops at the station, and when hoisted to surface the charge includes tramming from the collar of the shaft to the dump.

*Timbering.*—Cost of timbering the cross-cuts mentioned above includes the cost of handling and framing such timbers, but does not include the wages of station tenders and top carmen.

No repairs and renewals are included in above costs, as they are charged to repair accounts. This is true of all the following operating accounts, except as otherwise stated.

*Sinking and Up-raising.*—Charges under this heading are classified as follows:

Breaking Ground.—Includes all charges for sinking and up-raising, when shaft timber is used.

Tramming.—Cost of tramming waste. When waste is hoisted to surface, in addition to the wages of the shovelers there is included the cost of tramming the waste material from the collar of the shaft to the dump, and when hoisted to another level only the wages of the shovelers is included.

Timbering.—Cost of timbering shafts and up-raises, expenses of handling and framing timbers, and cost of compressed air used in hoisting timbers when up-raising.

Draining.—Cost of draining or unwatering shaft, including repairs to suction hose and repairs to and handling of sinking pumps; also cost of power used in operating pumps.

Hoisting and Lowering.—This includes expenses of operating auxiliary hoist while sinking shaft, repairs to sinking cages, buckets, shaft-sinking engine, etc., and power used by engine, not including repairs to main hoisting equipment.

The statistics shown under this heading are: number of feet advanced and cost per foot, and cubic feet excavated and cost per cubic foot.

*Cutting Stations and Skip Chutes.*—Charges under this heading are classified as follows:

Breaking Ground.—Cost of breaking ground for stations, and cost of excavating and putting in skip chutes, except where a station is cut for pumps and used exclusively for that purpose.

Tramming.—Cost of tramming waste and putting in new turnsheets and repairs to same while cutting station.

Timbering.—Cost of timbering stations and skip chutes and expenses of handling and framing such timbers.

Hoisting and Lowering.—Cost of hoisting and lowering, repairs to sinking cages, buckets, shaft-sinking engine, etc., and cost of power used by engine.

A separate statement is made for each station or skip chute cut, and under Statistics is shown the number of cubic feet excavated and cost per cubic foot.

*Draining.*

The cost of draining or unwatering mines includes wages of miners

engaged in digging ditches, putting in water boxes, pumpmen, cost of new drain tunnels, cost of pumping water into drain tunnels, proportionate part of draining by pumping plants based on the gallons of water pumped per minute, cost of diamond-drill work for drainage purposes, and cost of power used in operating pumps.

### *Mining.*

*Breaking Ground.*—Work performed in drifts, up-raises, winzes and stopes, lateral drifts and cross-cuts from laterals to main drift, cross-cuts run for the purpose of obtaining filling, breaking and shoveling waste on surface when used for underground filling, cost of putting in doors for ventilation, and cost of compressed air used for ventilation and drills.

*Tramming.*—Cost of tramming from drifts, up-raises, winzes, and stopes to the shaft, and on surface from shaft to ore bins, tramming waste on surface for underground filling, and on levels the cost of tramming from shaft to waste chutes; cost of shoeing horses and mules, new track and turnsheets in drifts, lateral drifts and lateral cross-cuts, and cost of electric power used in operating motors.

*Timbering.*—Cost of timbering in drifts, up-raises, winzes, and stopes, lateral drifts, lateral cross-cuts, new ore chutes, raising of chutes from floor to floor, cost of handling and framing timbers used, cost of operating timber hoists for hoisting timber into the stopes, and cost of power used by same.

### *Hoisting and Lowering.*

Wages of engineers on main hoist, auxiliary hoist, underground engines, oilers and wipers, station tenders engaged in handling ore, waste, materials, and supplies, hoisting charges when ore is hoisted through another shaft, cost of compressed air and steam power used by hoisting engines, and cost of steam furnished for heating air in receivers.

### *Superintendence.*

Salaries of the Superintendent, shift bosses, foremen, and clerks, including residence repairs and maintenance of horses and vehicles used by these employees.

### *Fire Expense.*

This being an extraordinary expense it is considered necessary to show it under a separate account. It includes all charges for controlling and fighting fire underground.

*Miscellaneous.*

Wages of timekeepers and watchmen, and charges from redistributable accounts which cannot be properly charged to other operating accounts, such as assaying, lighting, teaming, change house, surface, shop maintenance, and precipitating underground.

REPAIRS AND RENEWALS.

*Hoisting Equipment.*—Cost of repairing or renewing air receivers, air pipes from receivers to engine, engines, cables, head frame, cages, skips, bell ropes, signal lines, etc.

*Shafts and Stations.*—Cost of repairing, renewing or retimbering shafts, stations, and skip chutes, and repairs and replacements of turnsheets in the stations.

*Levels.*—Cost of repairing or renewing underground timbering, except such repairs as are otherwise stated.

*Draining Equipment.*—Cost of repairing drain tunnels, pumps and accessories, pipes, water boxes, tanks, etc.

*Cars, Motors, and Track.*—Cost of repairing or renewing cars, motors, tracks, turnsheets, switches, etc., whether on surface or underground.

*Timber Hoists.*—Cost of repairing or renewing timber hoists, trucks, etc.

*Air Drills and Hose.*—Expenses of repairing or renewing air drills, hose, tripods, bars, etc., and replacing parts of same, which include all repairs from the connection at the air pipe, except sharpening or repairs to steel for drills.

*Hammer Drills and Hose.*—(Same as Air Drills and Hose.)

*Tools and Utensils.*—Cost of repairing, renewing, or adding to tools and utensils used in operating the mines, including wages of toolmen engaged in carrying tools, and wages of tool sharpeners when engaged in sharpening steel for drills, picks, tools, etc.

*Ore Bins.*—Cost of repairing, renewing, or adding to shaft ore bins and railroad ore bins, including foundations, trestles, approaches and all permanent attachments thereto.

*General.*—Cost of making repairs or renewals not otherwise provided for.

GENERAL EXPENSES.

Items making up this expense, and which are self-explanatory, are as follows:

General office, legal, traffic, telephone, engineering, geological, donations, taxes, and insurance. The distribution to the various mines is made to (1) mines direct where such charges are obtainable, and (2) balance proportioned on tonnage basis.

*Redistributable Accounts.*

*Power House.*—Wages of firemen, coal passers, and ashmen, and all work performed for the purpose of developing steam power; also all charges for repairing or altering buildings, ash flumes, ash houses, water tanks, boiler feed pumps, repairing or cleaning boilers, and premiums paid on boiler insurance. Distribution is made as "Steam Power" to the various operations on the basis of horse-power furnished.

*Compressed Air.*—Wages of compressormen, oilers, and wipers, etc., tending compressors and motors, pipemen and others when engaged in or repairing air pipe lines; cost of repairing buildings, oil houses, compressors, and electric motors used in operating compressors; cost of steam and electric power used for operating compressors. Distribution is made to the various operations on the basis of horse-power furnished.

*Pumping Plant.*—Wages of pumpmen running pumps and all work performed in pumping water from the drain tunnel to surface through tunnels to precipitating boxes; also cost of repairing or renewing pumps and accessories, pump stations, etc.; cost of steam and electric power used for operating pumps. Distribution is made to "Draining" expenses of the different mines on the basis of gallons of water pumped per minute.

*Change House.*—Wages of men employed in taking care of change house, also all work performed in making repairs or alterations thereto, steam furnished for heating, electric current for lighting. Distribution is made to "Miscellaneous" expenses of the different mines on the basis of total number of men employed.

*Shop Maintenance.*—Wages of men employed in making or repairing any kind of tools, steam hammers, or machinery used in shops or connected therewith; also all labor performed in repairing or altering shop building. Distribution is made to "Miscellaneous" expenses. Work performed by shopmen either underground or on surface is distributed daily to the proper repair acccount.

*Electric Light.*—Wages of electricians and helpers employed in the repair and maintenance of light lines and lamps on surface or underground, electric current furnished for lighting, cost of all supplies, such as lamps, shades, globes, carbons, wire, and electrical fixtures. Distribution is made to "Miscellaneous" expenses of the different mines on the basis of number of lights used.

*Electric Plant.*—Wages of men employed in tending motors, generators, switchboards, transformers, etc., cost of repairing or renewing machinery, cost of current purchased. Distribution is made

as "Electric Power" to the various operations on the basis of power used.

*Assaying.*—Wages of assayers, chemists, sample grinders, and helpers employed in assay office; also work performed in repairing or renewing buildings and machinery. Distribution is made to "Miscellaneous" expenses of the different mines on the basis of number of determinations made.

*Surface.*—Wages of men employed in grading and repairing roads, cleaning up yards, etc., and all surface work not chargeable to other specified accounts. Distribution is made to "Miscellaneous" expenses of the different mines on the basis of work performed.

*Teaming.*—Wages of teamsters and stablemen engaged in general teaming, caring for horses and mules, repairing stables, vehicles, and harness. Distribution is made to the various accounts on the basis of work performed.

*Precipitating.*—Wages of men employed in precipitating copper from mine water underground, including building of boxes, ditches, etc., is distributed to "Miscellaneous" expenses of the mine producing such precipitates. Surface precipitating plants are not distributed to expenses of any of the mines, as the expense of operating such plants could not be properly so charged; these expenses are kept separate.

*Miscellaneous.*—Sales of power, air, and light to outside mines and parties are credited to above expenses, and the net cost distributed to power, air, and light of the various operations. The amount realized from sale of scrap iron to outside mines and parties is credited to Sundry supplies under "Miscellaneous" account.

Deductions from "Operating" expenses are made for labor performed and supplies furnished outside mines and parties which have been charged to operating, but no deductions are made for revenues from rents, such as buildings, machinery, shafts, or dwellings, and royalties from ore and precipitates.

## General Remarks.

After the costs as enumerated above have been shown on the cost sheet, the following information is given under the head of "Statistics": Wet tons of ore extracted, per cent. moisture, dry tons extracted, per cent. copper and contents, ounces silver per ton and contents. Also an efficiency statement showing (1) number of days in operation, (2) average number of shifts per day, (3) average number of tons per day, and (4) average number of tons per shift per day.

On account of the number of mines operated by the company some system had to be perfected in order to make a concise and intelligent

cost sheet, and it is believed that with the introduction of the present system this has been accomplished, consisting as it does of only two sheets for each mine, without details of redistributable accounts, which adds approximately two sheets.

The cost sheet as finally prepared gives the total cost of the above accounts, with the cost per ton; classification of labor, divided between operating, shops, and superintendence, with the shifts worked, total cost and cost per ton; and classification of supplies, with the total cost and cost per ton.

All labor distributions underground are made by the foremen, turned over to the timekeepers, completed, and sent to the mine office for tabulation and verification. Each day's payroll is checked with the distribution to the various accounts, which saves a great deal of time at the end of the month. When the last day's payroll and distributions have been checked the monthly payroll and distributions are complete, usually about the second of the following month.

Statements are made monthly showing the extraordinary expenses, if any, which have been included in the operating or repair accounts. Under this head may be classified such items as expenses of mines during shut-down, special repairs to hoisting engine, gallows frame, retimbering of shafts, etc. This information in the past has been of great value in making comparisons on the cost of extracting ore as between periods.

The Statistical department each period prepares a report showing the detailed cost of mining as compared with the former period, and such items as the above are taken into consideration, as is also increase or decrease in costs of development and shaft sinking, increase of wages, fire expense, etc., and the final results show whether ore has been extracted at a lower or higher cost and the reasons therefor.

In the Statistical department all the costs per ton shown are tabulated monthly, and a report is furnished condensing the cost sheet to such an extent that costs per ton of the various items at each mine are seen at a glance, and remarks made covering the reasons for any irregularities.

A monthly statement is prepared showing the cost of production per pound of copper from each of the mines operated, carrying the ores treated through each one of the processes at the smelter, arriving at a result for the treatment and production from such ores. This statement has been made for several years, and such efficiency has been attained that monthly figures check the final results for the year when balance sheets are prepared.

TABLE I.—*Chart Showing Distribution of Labor, Material, and Redistributable Accounts.*

| | Direct Charges. | | Redistributable Accounts. | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Labor. | Materials. | Steam Power. | Compress'd Air. | Pumping. | Change House. | Shop Maintained. | Electric Light. | Electric Power. | Assaying. | Surface. | Teaming. | Precipitat. | General Expenses |
| **OPERATING.** | | | | | | | | | | | | | | |
| Development: | | | | | | | | | | | | | | |
| Breaking ground | — | — | | — | | | | | | | | — | | |
| Tramming | — | — | | | | | | | | | | — | | |
| Timbering | — | — | | | | | | | | | | | | |
| Sinking and up-raising | — | — | | — | | | | | | | | | | |
| Cutting station | — | — | | — | | | | | | | | | | |
| Draining | — | — | | — | — | | | | — | | | | | |
| Mining: | | | | | | | | | | | | | | |
| Breaking ground | — | — | | — | | | | | — | | | — | | |
| Tramming | — | — | | | | | | | — | | | — | | |
| Timbering | — | — | | — | | | | | | | | | | |
| Hoisting and lowering | — | — | — | — | | | | | — | | | | | |
| Superintendence | — | — | — | | | | | | | | | | | |
| Fire expense | — | — | | — | — | | | | | | | | | |
| Miscellaneous | — | — | — | | | — | — | — | | — | — | — | — | |
| Total | — | — | — | — | — | — | — | — | — | — | — | — | — | |
| **REPAIRS AND RENEWALS.** | | | | | | | | | | | | | | |
| Hoisting equipment | — | — | | | | | | | | | | | | |
| Draining equipment | — | — | | | | | | | | | | | | |
| Shafts and stations | — | — | | — | | | | | | | | | | |
| Levels | — | — | | | | | | | | | | | | |
| Cars, motors, and track | — | — | | | | | | | | | | | | |
| Timber hoists | — | — | | | | | | | | | | | | |
| Air drills and hose | — | — | | | | | | | | | | | | |
| Hammer drills and hose | — | — | | | | | | | | | | | | |
| Tools and utensils | — | — | | | | | | | | | | | | |
| Ore bins | — | — | | | | | | | | | | | | |
| General | — | — | | | | | | | | | | | | |
| Total | — | — | | | | | | | | | | | | |
| **GENERAL EXPENSES.** | | | | | | | | | | | | | | |
| Charged direct | — | — | | | | | | | | | | | | |
| Proportioned | | | | | | | | | | | | | | — |
| Total | — | — | | | | | | | | | | | | — |
| Grand Total | — | — | — | — | — | — | — | — | — | — | — | — | — | — |

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.
[SUBJECT TO REVISION.]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, October, 1913, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1913, when Vol. XLVI. of the *Transactions* will go to press. Any discussion offered thereafter should preferably be in the form of a new paper for publication in Vol. XLVII. (with suitable cross references in both volumes).

# New Design of Regenerators for Open-Hearth Furnaces.

BY H. F. MILLER, JR., VERONA, PA.

(Butte Meeting, August, 1913.)

THE major cause of the deterioration of the open-hearth furnace as its length of service increases, is the melting down, or rather the slagging, of the checker-brick, together with the deposition of dirt in the form of iron oxide, etc., which helps to close the openings, as well as to slag the brick by chemical action.

Any construction that will do away with the necessity of having the waste gases come down through the checkers will eliminate the main reason for their deterioration. There are some other well-defined reasons which are subordinate to the main one given above.

Some of the defects of the present type of regenerator chambers are as follows:

1. The checkers cannot be cleaned while the furnace is in operation, nor can they be cleaned at the end of the run without taking out all of the checker-brick. This causes high costs for labor and brick. The labor cost is high because all checker-brick have to be handled twice. The brick cost is high because many of the bricks are spalled in the double handling, rendering them useless because a spalled brick will not stand up.

2. The top courses of brick become coated with iron oxide, etc., melt down, and large amounts of dirt are deposited throughout the checkers, especially at the bottom. The draft of the furnace is decreased by the closing of the top and bottom openings, especially the top ones, and the entire working of the furnace is affected. If an effort is made to clean the checkers the dirt and brick falling down into the holes still further diminish the draft.

3. The distribution of the air or waste gases through the checkers is in many cases uneven, resulting in a lower temperature of preheated air and therefore a slower working furnace. Also, the passage of air and waste gases is so crooked that sometimes the chimney is not powerful enough to overcome the friction.

These items are all important factors in decreasing the possible tonnage of a furnace.

The ideal regenerator chambers have then the following qualities to overcome the defects mentioned:

1. The checker work must at all times be accessible in all parts, so that the chamber can be either cleaned or repaired while the furnace is in operation.

2. The chamber must be of such shape that the checkers may be cleaned perfectly without touching a brick.

3. The checker work should be so placed that there will be a uniform distribution of air or waste gases throughout the chamber and a minimum of dirt deposited on the brick nearest the hearth.

4. The bricks of the chamber should always be in such state that they would practically never have to be removed.

These qualifications the writer believes would be fulfilled by a regenerator chamber built according to the accompanying design, Fig. 1. The explanation of the drawing is as follows:

The chamber forms an enlarged portion of the flue proper, the travel from the downtake to the chimney being in almost a straight line.

The checkers are built on the floor, thus doing away with the unstable tiles and rider walls. The checkers occupy a central portion of the chamber, with an air space at each end as wide as the chamber for a few feet, and then converging at an easy angle to the passageway at each end of the chamber.

This air space plays an important part in the life and working of the checkers: The waste gases going out reach this wide distribution space and are slowed up, and the suspended dust is deposited on the floor, from which it may be scraped out at any time through the seals at either side of the distribution space at both ends of the chamber.

The waste gases go through the chamber with very little further deposit of dirt. On Saturdays, after the last heat is out, the gas may be put on the end that is to be cleaned and in a short while the temperature of the end of the chamber nearer the air valve will be low enough for a man to enter and clean the checkers with compressed air or steam. The gas checkers are cleaned the same way, except that the gas is not admitted and the manhole is opened to supply air.

The gas sewer in this design of checkers is entirely separate from the flue to the stack and is accessible at all times. In addition to this benefit, the sewer may be burned out quickly with hot air by

using compressed air as an aspirator and closing the stack damper from the gas checkers.

It is readily seen that the checkers will last indefinitely under these conditions, with very few renewals of brick and no renewals of the entire checkers at the end of the run.

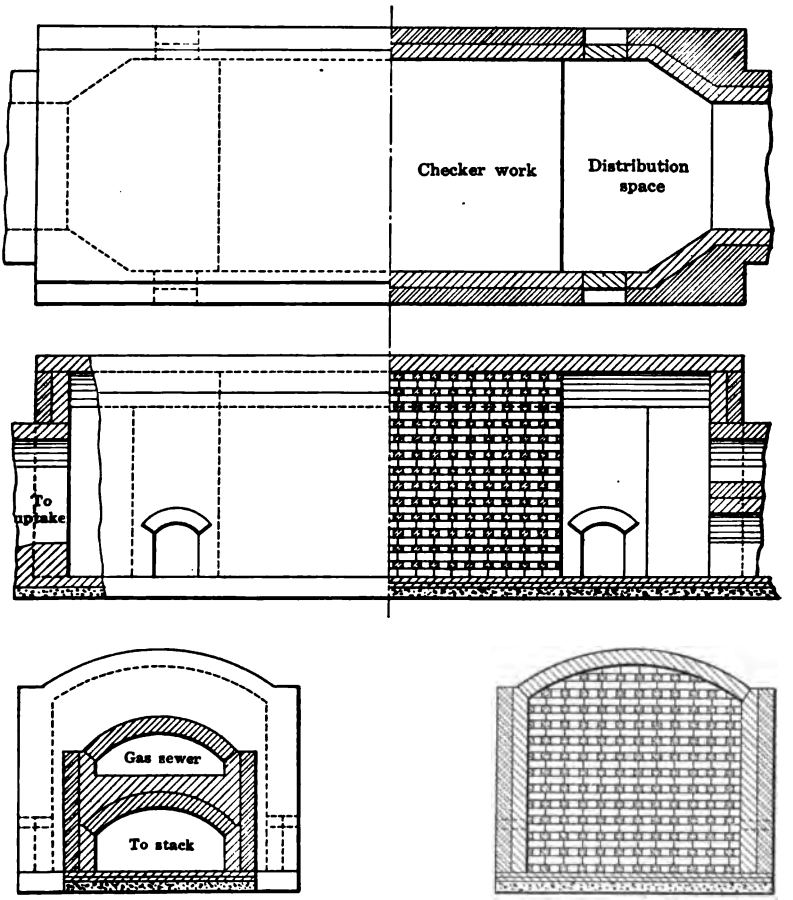


FIG. 1.—NEW DESIGN OF REGENERATOR CHAMBERS FOR OPEN-HEARTH FURNACES.

The wet slag will, of course, be deposited as usual in the cinder pockets.

The results obtained with regenerator chambers built on this design will be as follows:

1. There will be no slowing up of the furnace due to loss of draft because of dirty checkers.

2. The draft will be greater because the lines of draft will be straighter. This is very important in producer-gas work, where the travel of gas should be made as easy as possible. The preheated air or gas will be hotter than with the usual checkers because it will be more evenly distributed through the checkers.

3. The final result, all other things being equal, will be an increased tonnage, a longer run, and faster time made in producing the steel.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.
[SUBJECT TO REVISION.]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Butte meeting, Aug. 18 to 21, 1913, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1913, when Vol. XLVI. of the *Transactions* will go to press. Any discussion offered thereafter should preferably be in the form of a new paper for publication in Vol. XLVII. (with suitable cross references in both volumes).

# Assay of Gold and Silver by the Iron-Nail Method.*

BY E. J. HALL,† NEW YORK, N. Y., AND C. W. DRURY,‡ KINGSTON, ONT., CANADA.

(Butte Meeting, August, 1913.)

THE iron-nail method of assaying has been used for a number of years, but has not met with the approval of all assayers. The method possesses advantages which may be given as follows: (1) no preliminary treatment is required; (2) a lead button of proper size can be obtained; (3) it is economical. On the other hand, it has its disadvantages, which may be summed up as follows: (1) it is not applicable to ores containing large amounts of impurities other than sulphur, as practically all the base metals will contaminate the lead button; (2) the slag is unsatisfactory in the larger number of fusions; (3) the method was known to give low silver results under certain conditions and the gold results were questionable. It was with the idea of determining the extent of this latter disadvantage that the following investigation was undertaken.

It is well known that the fusion obtained in the lead assay is far more satisfactory than that in the "nails method" for gold and silver. The principal difference in the charges as a class is the presence of considerable reducing agent in the lead charge, and a limited oxidizing action in the nails charge, due to PbO.

The possibility of improving the nails fusion by simulating the lead charge was the reason for using argol in some tests, even though there were reasons for expecting this would have an objectionable effect on the results.

## *Preparation of the Gold-Bearing Material.*

The materials containing gold and silver used in the following experiments were prepared rather than selected, in order to have a definite working product excluding undesirable elements. A high-

* An investigation conducted in the Assay Laboratory at Columbia University, New York.

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grade material was made so that any difference in the results obtained from the several methods employed might be more marked.

The material containing the gold was prepared as follows: Some pure quartz was ground to pass 60 mesh, and sized on an 80-mesh screen. Cupels were made of the —60 + 80 material, using sodium silicate (1 of silicate to 6 of water) as a binder. These were very soft and crumbly, so the finer (—80) quartz was tried, with sodium silicate as above. With the finer material the cupels were very firm: they were dried and heated at a red heat for five minutes. Approximately 5 g. of metallic gold was dissolved in aqua regia, heated until all the chlorine was removed, and the solution diluted to 20 cc. The cupels weighed 15 g. apiece, and into each was poured 3⅓ cc. of gold solution. These were then dried and heated at a red heat to break up the gold chloride and precipitate the gold in the metallic state. At the points of contact of the cupel, due to the unevenness of the base and the asbestos on which they were dried, metallic gold, which was afterwards removed, was seen; otherwise the gold was in a very fine condition, not visible with a magnifying glass. The cupels were ground to pass 120 mesh, and no metallic particles were visible on the screen.

*Preparation of Silver Sulphide.*

Cupels similar to those used to absorb the gold chloride solution were made, and in these were placed 3⅓ cc. of silver nitrate solution prepared by dissolving 23.8 g. of silver nitrate in 20 cc. of water. The solution was absorbed in about 2 min., but not as quickly as the gold solution. The cupels were placed in a 20 per cent. sodium sulphide solution over night, and then in a 5 per cent. sodium sulphide solution for 5 hours. They were washed to remove the excess of sodium sulphide, and dried, not above a temperature at which they could be handled with the fingers at any time.

The iron pyrite used in the following experiments was ground to pass a 120-mesh screen. It contained 47.4 per cent. of sulphur (= 88.9 per cent. of $FeS_2$) and had a reducing power equal to 10 g. of lead.

*Assay of the Gold- and Silver-Bearing Materials.*

The assays were made by the crucible method, and a check was also run with each assay. The charge used to assay this material was as follows:

| | Grams. | | Grams. |
|---|---|---|---|
| Gold material | 0.5 | Litharge | 30.0 |
| Silver material | 1.0 | Borax glass | 7.0 |
| Soda | 10.0 | Argol | 2.0 |

For the checks weighed quantities of gold and silver were taken and added to a charge similar to the above.

The results were as follows:

| Test No. | Original Weight. | | Recovered Weight. | | Loss. | |
|---|---|---|---|---|---|---|
| | Gold. | Silver. | Gold. | Silver. | Gold. | Silver. |
| | Mg. | Mg. | Mg. | Mg. | Mg. | Mg. |
| 1 | 23.86 | 73.09 | 23.84 | 71.84 | 0.02 | 1.25 |
| 2 | 23.38 | 72.46 | 23.34 | 71.24 | 0.04 | 1.22 |
| 3 | 23.90 | 72.76 | 23.82 | 71.04 | 0.08 | 1.72 |

Average gold loss..... 0.05 mg.
Average silver loss..... 1.41 mg.

*Assay of Materials by the Crucible Method.*

| Test No. | Uncorrected Assay. | | Corrected Results. | |
|---|---|---|---|---|
| | Gold. | Silver. | Gold. | Silver. |
| | Mg. | Mg. | Mg. | Mg. |
| 5 | 24.08 | 74.20 | 24.13 | 75.61 |
| 6 | 24.12 | 73.72 | 24.17 | 75.13 |
| 7 | 24.08 | 73.76 | 24.13 | 75.17 |
| 8 | 24.04 | 73.40 | 24.09 | 74.81 |

| | Uncorrected. | Corrected. |
|---|---|---|
| | Mg. | Mg. |
| Average gold results, in 0.5 g. of material..... | 24.08 | 24.13 |
| Average silver results, in 1 g. of material..... | 73.77 | 75.18 |

The different sulphur determinations in the slags were made as follows:

*Total Sulphur.*[1]—With 0.5 g. of finely powdered slag, was mixed 3 g. of zinc oxide and 0.75 g. of dry sodium carbonate. The mixture was heated at a dull red heat for 15 min. After cooling, the mixture, which is easily removed from the crucible if the temperature was not too high, is treated with water and brought to a boil, filtered, the filtrate acidified, boiled, and precipitated with barium chloride.

*Sulphide Sulphur.*—Was determined by treating the sample with HCl, absorbing the $H_2S$ gas in ammoniacal cadmium chloride and titrating with iodine solution, using starch as an indicator.

The following experiments were carried on with varying mixtures

[1] W. C. Ebaugh and C. B. Sprague, *Journal of the American Chemical Society*, vol. xxix, No. 10, p. 1475 (Oct., 1907).

of pyrite and silica, to which were added definite amounts of the gold and silver material. The mixtures contained 6, 13, 20, 40, 60, 80, and 88.9 per cent. of pure pyrite, or 6.7, 14.6, 22.4, 44.9, 67.4, 90, and 100 per cent. of the impure pyrite (88.9 per cent. pure), enough silica being added to the pyrite to make 0.5 assay ton.

While these tests were made primarily to determine the value of the nails charge, it was thought advisable to run niter fusions on the same series, particularly in view of the oft-repeated statement that charges requiring more than 20 g. of niter were not satisfactory. The nails charges (Tests A and B) were:

| | |
|---|---|
| Pyrite mixture | 0.5 A. T. |
| Sodium carbonate | 30 g. |
| Litharge | 25 g. |
| Borax glass | 10 g. |
| Nails, four | 10 d. |

Test A contained no argol, while Test B contained 5 g. in each assay.

For the niter tests (Test C) the following charge was used:

| | |
|---|---|
| Pyrite mixture | 0.5 A. T. |
| Sodium carbonate | 15 g. |
| Litharge | 105 g. |
| Silica | 5 g. |
| Borax glass | 7 g. |
| Niter or argol to give the required amount of lead. | |

The silica and so large an amount of flux is not required in the lower pyrite mixtures, but was used to make the flux charge uniform. The number of grams of pyrite in 0.5 assay ton mixture is indicated in Table I.

The fusion of the nails charges which contained no argol was of the usual character. Those containing small amounts of pyrite were fairly satisfactory, but as the percentage of pyrite increased there was less complete dissolution of constituents; the nails were badly corroded, and in some cases to such an extent that they sank beneath the slag. The lead did not collect perfectly in all cases, leaving small particles in the crucible on pouring.

The charges containing argol were much better, but showed the same general effect with the increased amount of pyrite. Complete dissolution seemed to take place, and in no case were the nails corroded to such an extent that they disappeared. The charges poured much better, showing a good collection of the lead. This improved condition is unquestionably due to the decrease in oxidation of constituents in the charge. As this charge is highly basic, containing

TABLE I.—*Results of Tests by the Nail, Nail-Argol, and Niter Methods.*

| Test No. | Charge. | | | Assay. | | Sulphur in Iron Pyrite. | Weight of Slag. | Sulphur in Slag. | | | | | | Sulphur Removed. | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Pyrites (88.9% $FeS_2$) | Argol. | Niter. | Gold. | Silver. | | | Total Sulphur. | | Sulphide Sulphur. | | Oxidized Sulphur.[c] | | | |
| | Grams. | Grams. | Grams. | Mg. | Mg. | Grams. | Grams. | Per Cent. | Grams. | Per Cent. | Grams. | Per Cent. | Grams. | Per Cent. | Grams. |
| 1-A | 0.96 | ..... | ..... | 24.14 | 73.78 | 0.455 | 43.7 | 1.1 | 0.48 | 0.095 | 0.043 | 1.01 | 0.43 | ..... | ..... |
| B | 0.96 | 5 | ..... | 23.68 | 71.46 | 0.455 | 36.5 | 1.0 | 0.365 | 0.33 | 0.117 | 0.66 | 0.248 | 19.7 | 0.09 |
| C | 0.96 | 1 | ..... | 24.08 | 74.10 | 0.455 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 2-A | 2.08 | ..... | ..... | 24.18 | 73.42 | 0.982 | 45.0 | 2.0 | 0.90 | 0.48 | 0.216 | 1.52 | 0.69 | 8.1 | 6.08 |
| B | 2.08 | 5 | ..... | 23.65 | 72.09 | 0.982 | ..... | 1.9 | ..... | 0.84 | ..... | 1.06 | ..... | ..... | ..... |
| C | 2.08 | ..... | ..... | 23.92 *a* | 72.96 *a* | 0.982 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 3-A | 3.2 | ..... | ..... | 23.97 | 72.92 | 1.52 | 48.2 | 2.6 | 1.25 | 0.97 | 0.467 | 1.63 | 0.79 | 17.7 | 0.27 |
| B | 3.2 | 5 | ..... | 24.10 | 72.26 | 1.52 | 38.5 | 2.9 | 1.1 | 1.9 | 0.731 | 1.0 | 0.37 | 26.9 | 0.41 |
| C | 3.2 | ..... | 2 | 24.12 | 74.88 | 1.52 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 4-A | 6.4 | ..... | ..... | 23.88 | 71.02 | 3.03 | 50.7 | 5.4 | 2.73 | 3.07 | 1.55 | 2 33 | 1.18 | 10.0 | 0.3 |
| B | 6.4 | 5 | ..... | 21.36 *b* | 60.78 *b* | 3.03 | 37.5 | 5.7 | 2.13 | 3.7 | 1.61 | 2.0 | 0.52 | 29.7 | 0.9 |
| C | 6.4 | ..... | 9 | 24.14 | 73.82 | 3.03 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 5-A | 9.6 | ..... | ..... | 23.98 | 70.14 | 4.55 | 52.8 | 7.6 | 4.0 | 4.87 | 2.57 | 2.73 | 1.5 | 12.1 | 0.55 |
| B | 9.6 | 5 | ..... | 23.92 | 66.10 | 4.55 | 42.5 | 7.5 | 3.28 | 5.3 | 2.25 | 2.2 | 1.03 | 27.9 | 1.27 |
| C | 9.6 | ..... | 17 | 24.16 | 74.84 | 4.55 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 6-A | 12.8 | ..... | ..... | 23.83 | 67.45 | 6.02 | 57.5 | 8.8 | 5.06 | 6.01 | 3.45 | 2.79 | 1.61 | 16.5 | 0.96 |
| B | 12.8 | 5 | ..... | 24.00 | 63.68 | 6.02 | 48.5 | 8.4 | 4.15 | 6.71 | 3.25 | 1.69 | 0.90 | 31.0 | 1.87 |
| C | 12.8 | ..... | 24 | 24.18 | 74.18 | 6.02 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 7-A | 14.5 | ..... | ..... | 23.99 | 66.13 | 6.86 | 62.4 | 6.8 | 4.25 | 5.5 | 3.43 | 1.3 | 0.82 | 38.0 | 2.61 |
| B | 14.5 | 5 | ..... | 24.12 | 62.56 | 6.86 | 46.0 | 8.8 | 4.0 | 7.6 | 3.49 | 1.2 | 0.55 | 40.5 | 2.80 |
| C | 14.5 | ..... | 32 | 24.09 | 73.90 | 6.86 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |

Uncorrected results of gold material..... 24.08 mg.
Uncorrected results of silver material..... 73.77 mg.

Tests marked A, Nails Assay. Tests marked B, Nails assay + 5 g. argol. Tests marked C, Niter assay.
Tests Nos. 1, 2, 3, 4, 5, 6, 7, corresponded to 6, 13, 20, 40, 60, 80, 89.9 per cent., respectively, of pure pyrite on 0.5 A. T.
*a* Bead No. 2 C froze. *b* In assay No. 4 B a little lead was lost. *c* Oxidized sulphur obtained by difference.

large amounts of sodium carbonate, the sulphides are readily soluble; but if the iron present is converted to oxide this will not be readily taken into solution. It is not permissible to make this slag highly acid to take these oxides into solution, as the sulphides would then fail to dissolve.

It was noticed in the different fusions that the character of the furnace atmosphere, whether oxidizing or reducing, had considerable effect on the charge. If the atmosphere was strongly oxidizing the fusion was not nearly as good as when it was reducing, this apparently being due to the same causes as where argol is present in one case and absent in the other.

The largely increased weight of slag in charges high in pyrite is due partly to the greater amount of iron dissolved, and partly to the material corroded from the crucible on account of the much more basic character of these slags.

From the results, it will be noted that the silver losses increase with the sulphur content. Also, that in charges containing argol the loss is somewhat greater than where the addition of argol was omitted. The principal difference in the slags of these two methods, which could affect the results, is in the percentage of sulphide sulphur present, the charges containing argol having a much greater percentage. This indicates that the silver losses are due to sulphide sulphur.

This explanation seems quite reasonable, as these slags are somewhat of the nature of a matte, particularly where sulphide sulphur is high, and it is well known that mattes are good solvents for silver.

For the same reasons, it was expected that the gold loss would also increase to a considerable extent. It will be noted from the table that the increased loss is not proportionate to the silver, nor does it seem to increase progressively with the sulphur.

In a number of the experiments, the nails were weighed before and after fusion of the charge. It was seen that considerably more iron was removed from the nails than was necessary to form iron sulphide; also, on examining the nails, that they were covered with a dull brown coating. The following figures show the amounts of iron removed from the nails. Tests Nos. 8 and 10 were nail assays; and, to Tests Nos. 9 and 11, 5 g. of argol were added.

| Test No. | Iron in Pyrite. | Sulphur in Charge. | Weight of Nails. | | Loss of Iron from Nails. | Weight of Slag. | Iron in Slag.[a] | |
|---|---|---|---|---|---|---|---|---|
| | | | Before Fusion. | After Fusion. | | | | |
| | Grams. | Grams. | Grams. | Grams. | Grams. | Grams. | Per Cent. | Grams. |
| 8 | 6.0 | 6.86 | 29.2 | 13.2 | 16.0 | 56.2 | 27.95 | 18.1 |
| 10 | 6.0 | 6.86 | 29.4 | 13.9 | 15.5 | 58.5 | 28.7 | 19.1 |
| 9 | 6.0 | 6.86 | 29.7 | 17.7 | 12.0 | 44.5 | 21.5 | 13.1 |
| 11 | 6.0 | 6.86 | 29.1 | 15.5 | 13.6 | 46.4 | 25.1 | 15.3 |

*a* This weight includes the weight of iron on nails as scale.

Some of the coating or scale on the nails was removed and analyzed for total sulphur, sulphide sulphur, and iron. The results were as follows:

| Test No. | | Total Sulphur. | Sulphide Sulphur. | Iron. |
|---|---|---|---|---|
| | | Per Cent. | Per Cent. | Per Cent. |
| 10 A. | Nail quenched in water after removing from fusion... | 3.55 | 1.6 | 58.05 |
| 10 B. | Nail removed from fusion, cooled in air..... | 3.05 | 0.4 | 56.65 |
| 11 A. | Nail quenched in water after removing from fusion... | 3.35 | 1.3 | 60.9 |
| 11 B. | Nail removed from fusion, cooled in air..... | 3.8 | 0.5 | 54.5 |

The scale was considered to be iron sulphide, but this was impossible from the small percentage of sulphur. A portion of the scale from Nos. 10 A and 10 B was ground and did not show any metallic particles. Further, a portion of this scale was magnetic, and the magnetic portion from No. 10 A contained 59.4 per cent. of iron, and the magnetic portion from No. 10 B contained 59.35 per cent. of iron.

The non-magnetic portion from Nos. 10 A and 10 B became magnetic when heated in the reducing flame of a blowpipe, which indicates the presence of ferric oxide ($Fe_2O_3$). The literature gives this as iron sulphide (FeS), but indications are to the contrary, and the point was thought worthy of further investigation, therefore the following tests were made:

Fusion (Test No. 12) was made, using this charge:

| | Grams. |
|---|---|
| Soda | 20 |
| Silica | 8 |
| Borax | 8 |
| Sodium sulphate | 5.5 |
| Nails, two | 29 |

Also, fusion (Test No. 13), using the same charge, except that no sodium sulphate was added.

| Test No. | Weight of Nails. | | Sulphide Sulphur in Slag. |
|---|---|---|---|
| | Before Fusion. | After Fusion. | |
| | Grams. | Grams. | Per Cent. |
| 12 | 29.1 | 22.5 | 0.57 |
| 13 | 29.8 | 26.5 | ..... |

In fusion No. 12, the nails were corroded badly beneath the slag and also the portion above the slag. The scale on the nails (from the part beneath the slag as well as the part above it) was found to contain a magnetic portion, as before, and analyzed 52 per cent. and 71.4 per cent. of iron, respectively.

In Test No. 13, there was also a scale on the nails above the slag, but the portion of the nail in the slag was bright and not corroded.

This scale, as before, contained a magnetic portion, which analyzed 68.6 per cent. of iron.

These results clearly indicate that metallic iron under above conditions reacts with sodium sulphate ($Na_2SO_4$), producing sulphide sulphur and iron oxide, and explain the excessive corrosion of nails during long fusion in an oxidizing atmosphere. Apparently, there is a cycle in the oxidation of sulphide sulphur to sulphate by the air, reduction of sulphate to sulphide by the iron, and reoxidation by the air. The presence of an organic reducing agent interferes with this oxidation, which is the reason for less corrosion of the nails in charges using argol. The production of this large amount of iron oxide accounts for the unsatisfactory nature of the slag, as the slag is not of a character suitable to dissolve quantities of this material, which remains more or less in suspension. Further, it is responsible for rapid attacking of the crucible, which is composed of constituents that are not conducive to a satisfactory slag with the fluxes employed.

*Conclusions.*

1. The nails method gives low results for silver with ores containing more than certain amounts of sulphide sulphur, this loss increasing with the percentage of sulphur in the ore, reaching a maximum of about 10 per cent. in the straight nails charge under the conditions of this test. In the case where argol was used, giving increased sulphide sulphur in the slag, the maximum loss was about 15 per cent.

2. The gold results are irregular and somewhat lower, possibly 1 per cent. The loss does not appear to be affected by the addition of argol to the charge.

3. The method should not be used for silver determinations in high sulphide ores, except for approximate results. When the sulphur in the ore is not much greater than will be oxidized by the PbO, a fair approximation may be obtained. The method will give fairly good gold results on medium- and low-grade gold ores, say one ounce, as the loss is less than the accuracy of ordinary work.

4. The niter method gives good results even with as much as 30 g. of niter present in the charge and a 30-g. button is necessary in assays containing a large amount of niter.

5. The portion of the iron nails which projects into the lead is converted to sulphide, but the greater amount of iron, that in contact with the slag, is removed as iron oxide.

6. The scale on the iron nails is not iron sulphide, but a mixture of ferric oxide, magnetite, and slag.

7. To obtain a good slag by the nails method with slight corrosion of nails, it is necessary to protect the fusion from oxidation. This, however, will lower the silver results.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.
[SUBJECT TO REVISION.]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, October, 1913, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1913. Any discussion offered thereafter should preferably be in the form of a new paper for publication in Vol. XLVII. (with suitable cross references in both volumes).

# Ae 1, the Equilibrium Temperature for A 1 in Carbon Steel.*

BY HENRY M. HOWE, COLUMBIA UNIVERSITY, NEW YORK, N. Y.

(New York Meeting, October, 1913.)

*The Equilibrium Position of A* 1.—Some of the most important data on this subject are collected in Table I.

*Definition of Ae* 1.—Just as we call A 1 of rising temperature Ac 1, and that of falling temperature Ar 1, so we may give the symbol Ae 1 to the equilibrium position of A 1.

*Preliminary Assumptions.*—In deciding what weight to assign the various data, and indeed what data are to count, three assumptions should be made. These are:

1. The corollary from the phase rule, that the temperature of Ae 1 is constant and independent of the carbon content, though, of course, it may be changed by the presence of other elements.

2. That because neither in heating nor in cooling can the transformation begin before the temperature reaches Ae 1, though it may be delayed by lag till after the temperature has passed beyond Ae 1, therefore Ae 1 should be at or below the observed beginning of Ac 1 and at or above the observed beginning of Ar 1, and, *a fortiori*, it should be at or below the observed maximum of Ac 1, and at or above the observed maximum of Ar 1. It may be necessary to take these maxima into consideration in view of the experimental difficulty of detecting these beginnings.

3. Because of the extraordinary degree of lag in the immediate neighborhood of Ae 1, as proved by Brayshaw, little weight should be attached even to the most conclusive evidence that the transformation has not occurred on reaching, or even staying at, a given temperature, because that failure to occur may always be due to lag. Evidence that the transformation has occurred, or has even begun at a given temperature deserves incomparably greater weight. In other words, the temperature may pass beyond Ae 1 without inducing the transformation, but the transformation cannot be induced except by

* A Contribution from the Metallurgical Laboratories of Columbia University. MS. received Feb. 10, 1913.

TABLE I.—*Important Determinations of A 1.*

| No. | Authority. | Date. | Composition of Specimens. C. | Si. | Mn. | P. | S. | Gr. | A 1. Ac 1. | Ar 1. | Gap. | Method. |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | | | | | ° C. | ° C. | ° C. | |
| 1 | Carpenter and Keeling | 1904 | 0.38 | 0.06 | Tr. | 0.03 | 0.01 | ..... | 724 (Beginning) | 724 (Beginning) | 0 | Thermal. Ar 1 faint. |
| 2 | Carpenter and Keeling | ..... | 1.85 | 0.09 | ..... | ..... | ..... | Nil | 720 | 718 | 11 | Thermal. Ar 1 well-marked. |
| 3 | Carpenter and Keeling | ..... | 3.98 | ..... | ..... | ..... | ..... | ..... | 728 | 728 | 0 | Thermal. Ar 1 faint. |
| 4 | Carpenter and Keeling | ..... | 4.50 | 0.12 | ..... | ..... | ..... | ..... | 728 | 715 | 13 | Thermal. Ar 1 well-marked. |
| 5 | Carpenter and Keeling | ..... | 2.63 | ..... | ..... | ..... | ..... | ..... | ..... | 723 | ..... | Thermal. Ar 1 well-marked. |
| 5 A | Carpenter and Keeling | ..... | 2.85 | ..... | ..... | ..... | ..... | ..... | ..... | 723 | ..... | |
| 6 | Carpenter and Keeling | ..... | 2.85 | ..... | Tr? | ..... | ..... | ..... | ..... | 716 | ..... | Thermal. Maximum of Ar 1 well-marked. |
| 7 | Heyn | 1904 | 0.39 | 0.04 | 0.03 | ..... | ..... | ..... | 700 | ..... | ..... | Thermal. Beginning of Ac 1 very faintly marked. |
| 8 | Heyn | ..... | 0.95 | 0.04 | 0.06 | ..... | ..... | ..... | ..... | 708 | ..... | Thermal. Beginning of Ar 3-2-1 very sharply marked. |
| ..... | Heyn | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | 695 | ..... | Thermal. Maximum of Ar 3-2-1 very sharply marked. |
| 9 | Rosenhain (on Brayshaw's steel A 2) | 1910 | 1.14 | 0.09 | 0.40 | 0.014 | 0.018 | Tungsten | 720 | ..... | ..... | Thermal. Maximum Ac 1, beginning at 713° ±. |
| 10 | Rosenhain (on Brayshaw's steel A 2) | ..... | ..... | ..... | ..... | ..... | ..... | ..... | 715 | ..... | ..... | Thermal. Beginning Ac 1. |
| 11 | Benedicks | 1908 | 1.00 | ..... | 0.25 | ..... | ..... | ..... | 725 | 695 | 30 | Thermal maxima. |
| 12 | Charpy and Grenet | 1903 | 0.64 | ..... | ..... | ..... | ..... | ..... | 713 | ..... | 15 | Dilatation. Ends of transformation in heating and cooling, with long holding at stationary temperature. |
| 13 | Charpy and Grenet | ..... | 0.64 | ..... | ..... | ..... | ..... | ..... | ..... | 698 | | |
| 14 | Charpy and Grenet | ..... | 0.93 | ..... | ..... | ..... | ..... | ..... | 728 | ..... | 16 | ..... |
| 15 | Charpy and Grenet | ..... | 0.93 | ..... | ..... | ..... | ..... | ..... | ..... | 712 | | ..... |
| 16 | Charpy and Grenet | ..... | 1.50 | ..... | ..... | ..... | ..... | ..... | 707 | ..... | ..... | Dilatation. Beginning of contraction in heating up slowly. |
| 17 | Brayshaw, No. W 4 | 1910 | 1.15 | 0.21 | 0.31 | .011± | .012± | 0.50 | ..... | 731 | ..... | *Hardening:* After holding at higher temperatures, cooling, holding at 731° and quenching, hardness decreased slowly as the holding time increased, at last approaching that of samples quenched from evidently below A 1. |
| 18 | Brayshaw, No. W 2 | 1910 | 1.16 | 0.10 | 0.37 | 0.014 | 0.023 | 0.48 | 729.25 | ..... | ..... | *Hardening:* Heated direct to quenching temperature, held there, quenched, and hardness determined. No. 18 hardened slightly at 729.25°, not at 728°. No. 19 hardened slightly at 728° but not at 725.5°. |
| 19 | Brayshaw, No. A 2 | ..... | 1.14 | 0.09 | 0.40 | 0.014 | 0.018 | 0 | 728 | ..... | ..... | |
| 20 | Levy | 1913 | 0.23 | 0.039 | 0.05 | 0.013 | 0.010 | 0 | 725 | ..... | ..... | *Hardening:* Heated direct to 725°, held there, and quenched. Hardened slightly. |
| 21 | Levy | 1912 | 0.92 | 0.14 | 0.123 | 0.009 | 0.011 | 0 | ..... | 710 | ..... | Heated to 900°, cooled to the quenching temperature, quenched. Hardened when quenched from 720°, but not from 710°. |

reaching or passing Ae 1. Hence, in determining the position of Ae 1, all data are here rejected which show only that the transformation has not occurred, or has not begun, at any given temperature.

The data collected in Table I., taken at their face, give a range of 31°, from 700° to 731°, because one of Heyn's results (No. 7) indicates that Ae 1 lies at or below 700°, and one of Brayshaw's (No. 17) indicates that it lies at or above 731°. But this uncertainty of 31° is unreasonably great for our present powers of observation. A critical study of the observations allows us to cut this range down to about 8°—viz., from 715° to 723°, and a weighted average of the determinations is 723°, as I will now explain.

*Narrowing the Range of Probable Temperatures for Ae* 1.—From among the data at hand certain ones may be selected as the most trustworthy. These should not include the beginnings of Ac 1 of Carpenter and Keeling, and of Charpy and Grenet, because their original curves are not available; nor Carpenter and Keeling's beginnings of Ar 1 at 724° and 728°, nor Heyn's beginning of Ac 1, 700°, because they are not well enough supported by the thermal curves in view of their conflict with other and well-supported results. Further, there is reason to think Brayshaw's temperature 731° slightly too high, as explained under No. 19 in the note to Table I. All the remaining data are compatible with the theory that Ae 1 lies between 715° (No. 10 of Table I., Rosenhain) and 723° (Nos. 5 and 5A, Carpenter and Keeling). That is to say, all the remaining Ar 1s lie at or below 723°, and all the remaining Ac 1s at or above 715°; and those Ar 1s which lie below 715° may be assumed to do so because of lag, and those Ac 1s which lie above 723° may be assumed to do so because of lag. 715° to 723°, may then be taken as the most probable range.

*A Weighted Average.*—Recognizing that such averages are weak because they represent only the arbitrary weightings of the weighter, which are purely matters of judgment and opinion, we may yet strike one for what it is worth.

The procedure ought not to follow the usual course, for an obvious reason. The determinations of most properties suggest on their face that they are neither too high nor too low. But this is not true of the determinations by means of which we try to find Ae 1. These determinations are always either Ac 1 points or Ar 1 points. But because each of these is subject to an indeterminate degree of lag, an Ac 1 determination means on its face only that Ae 1 is at or below that determination, and an Ar 1 result means on its face only that Ae 1 is at or above that result. That is to say, these results carry weight in only one direction, the Ac 1 results helping to prevent our

assigning too high a temperature to Ae 1, and the Ar 1 results helping to prevent our assigning too low a temperature, whereas a density or like determination carries weight in both directions, and should be taken into account in considering both those which are greater and those which are less than itself.

TABLE II.—*Calculation of the Weighted Average Temperature of Ae* 1.

| No. in Table I. | Ac 1 Data. Ae 1 is at or below ° C. | Ar 1 Data. Ae 1 is at or above ° C. | Weight. | Product. Ac 1. | Product. Ar 1. | Observer. | Method. |
|---|---|---|---|---|---|---|---|
| 17 | ..... | 731 | 2 | ..... | 1,462 | Brayshaw. | Hardening. |
| 18 | 729.25 | ..... | 3 | 2,188 | ..... | Brayshaw. | Hardening. |
| 3 | ..... | 728 | 1 | ..... | 728 | Carpenter and Keeling. | Thermal, beginning. |
| 11 | 725 | ..... | 2 | 1,450 | ..... | Benedicks. | Thermal, max? |
| 20 | 725 | ..... | 1 | 725 | ..... | Levy. | Hardening. |
| 1 | 724 | ..... | 1 | 724 | ..... | Carpenter and Keeling | Thermal, beginning. |
| 5 | | 723 | 5 | | 3,615 | | |
| 9 | 720 | ..... | 3 | 2,160 | ..... | Rosenhain. | Thermal, max. |
| 10 | 715 | ..... | 1 | 715 | ..... | Rosenhain. | Thermal, beginning. |
| 21 | ..... | 710 | 1 | ..... | 710 | Levy. | Hardening. |
| 8 | ..... | 708 | 5 | ..... | 3,540 | Heyn. | Thermal, beginning. |
| 16 | 707 | ..... | 1 | 707 | ..... | Charpy and Grenet. | Dilatation, beginning. |
| 7 | 700 | ..... | 1 | 700 | ..... | Heyn. | Thermal, beginning. |
| 13 | ..... | 698 | ..... | ..... | ..... | Charpy and Grenet. | Dilatation. |
| 11 | ..... | 695 | ..... | ..... | ..... | Benedicks. | Thermal. |

This may be made clearer by arranging the data as in Table II. Here we see that, though most of the results are in conflict with at least some one other, certain of them are not in conflict with any others. For instance, all the temperatures in the Ar 1 column, with the exception of Benedicks's 695° and Charpy and Grenet's 698°, argue that Ae 1 is higher than 700° and thus conflict with No. 7, Heyn's 700° for the beginning of Ac 1, which argues that Ae 1 must be at or below 700°. Again, all the results in the Ac 1 column are in conflict with Brayshaw's 731° for Ar 1, for they all argue that Ae 1 is below 731°, whereas his argues that Ae 1 is at or above 731°.

In striking the average, all these conflicting results should be considered, but those not in conflict should not be taken into account. For instance, Charpy and Grenet's 698° and Benedicks's 695° for Ar 1 are irrelevant, because they convey no suggestion as to how far above these temperatures Ae 1 lies, and they are wholly compatible

with any temperature assigned to Ae 1 within the conflicting limits of 700° below and 731° above. The reason for setting these in Table II. is to show why they and like results outside the conflicting range should not be counted. The highest and lowest heavy black lines in Table II. show the limits of this range.

We might at first believe that we ought to represent each observer by one temperature, the temperature to which his data point. But there is no single temperature to which his data point. His Ac 1s have one teaching and his Ar 1s another and independent teaching. Both ought to count and hence both should be included. The highest position for his Ar 1 and the lowest for his Ac 1 are presumably the most accurate, because the least influenced by lag. These, then, are the ones to count.

Moreover, in case his beginning is so faintly marked that we assign a very small weight to it and yet his maximum tends to support that beginning, then we may well take account of that maximum also, always provided that it falls within the conflicting range, *i. e.*, if an Ar 1 point, that it lies above 700°, and, if an Ac 1 point, that it lies below 731°.

The fact that the beginnings of Ar 1 and of Ac 1 are difficult to detect is provided for, not by ignoring them, but by assigning a light weight to them.

The temperature assigned to Ae 1 should be that which does the least violence to the data as weighted, and this temperature is between 723° and 724°, as indicated by the broad horizontal line midway in Table II. Let us take it as 723°. At this temperature the sum of the weighted conflicts is at the minimum. This is only 3° distant from the most important result with which it conflicts, Rosenhain's 720°. In considering this temperature we should remember that the conflict between it and Rosenhain's 720° is apparent only, because the 0.40 per cent. of manganese of his steel may easily lower its Ae 1 position by more than the 3° of conflict. With this 720° left out of account, only Rosenhain's 715°, Charpy and Grenet's 707° and Heyn's 700° for the beginning of Ac 1 argue that 723° is too high, and only Brayshaw's 731° for Ar 1 and Carpenter and Keeling's 728° for the beginning of Ar 1 argue that it is too low, and of these the former may well be several degrees too high. This weighted sum (700 + 707 + 715 = 2,122 below) + (1,462 + 728 = 2,190 above) = 4,312 is less than if any temperature above 724° or below 723° is selected. 723° is reconciled with all the data other than those just discussed by the reasonable assumption that those above it in the Ac 1 column are above it because of lag, and those below it in the Ar 1 column are

below it because of lag. Indeed, when we have reached this point we see that some temperature, close to 723°, would have to be selected even if no weighting process was followed.

*Summary.*—While there is evidence which tends to put Ae 1 as high as 731° and other evidence which tends to put it as low as 700°, yet a reasonable exclusion of the less strongly supported evidence gives a range of 715° to 723° as that within which Ae 1 probably lies.

A weighted average of the results discussed, including those thus rejected because outside the 715° to 723° range, is 723°.

## *Note to Table I.*

*1 to 6.* Carpenter and Keeling, *Journal of the Iron and Steel Institute*, vol. lxv. (1904, No. I.), p. 224, and *Collected Researches of the National Physical Laboratory*, vol. i., p. 227, Plate 4. Cylindrical specimens ⅝ in. long and ⅝ in. in diameter were heated, without complete exclusion of the air, in general to temperatures between 1,100° and 1,140° in 1.5 hr., held for about 0.5 hr., and cooled thence to 500° C. in about 1 hr. 20 min. For these individual specimens the full analyses are not given. For seven other specimens the manganese content is given, and in each of these it is a trace.

The time-temperature curve was obtained in certain cases by direct reading, and in other cases by differential reading.

The temperature, 723°, which I give for the beginning of Ar 1 of their 2.63 and 2.85 per cent. alloys (their Nos. 27 and 30), is not given directly by them, but read off their curves Nos. 27 and 30, Pl. 4, *Collected Researches of the National Physical Laboratory*, vol. i., p. 227.

*7 and 8.* Heyn, Mikroskopische Untersuchung, etc., *Verhandlungen des Vereins zur Beförderung des Gewerbfleisses* (1904), p. 371, and Plates B and C. The time-temperature curve was taken differentially. Specimens were heated rapidly, without complete exclusion of the air, to 1,100°, whence they were cooled. The cooling to 200° occupied about 42 min. in all cases.

*9 and 10.* Rosenhain, *Proceedings of the Institution of Mechanical Engineers* (1910), parts 1 and 2, p. 688; *Engineering*, vol. lxxxix. (1910), p. 702, column I, Fig. 2. This is the same steel as No. 19. The heating was done in a very high vacuum and apparently with very slow movements of temperature.

*11.* Benedicks, *Journal of the Iron and Steel Institute*, vol. lxxvii. (1908, No. II.), pp. 218 to 219. Cylindrical specimens 50 by 6.5 mm. were heated without complete exclusion of air. Ac 1 was at 725°; Ar 1 at 695°. His Ar 1 represents the temperature of complete arrest, *i. e.*, of maximum heat evolution, and his Ac 1 also is probably the maximum.

*12 to 16.* *Bulletin de la Société d'Encouragement pour l'Industrie Nationale* (1903, No. I.), pp. 480 to 482. The observations were all made by the dilatation method, with extremely slow changes of temperature, about 200° per hour in heating.

For Nos. 12 and 13 the transformation, as determined by dilatation, completed itself, on holding the temperature stationary between 708° and 713° for 1 hr. in heating up. In cooling down the transformation completed itself, on holding the temperature stationary for 0.5 hr., between 708° and 698°. For Nos. 14 and 15 the same conditions applied, the stationary temperature being between 723° and 728° in heating up and between 713° and 712° in cooling down. p. 482.

No. 16. Charpy and Grenet give on p. 480 the temperatures at which the transformation occurs, as indicated by the beginning of the abrupt contraction at Ac 1. The temperatures vary from 707° to 730°. In one case a temperature of 693° is given, but as in this case the contraction was "hardly appreciable," the carbon content being only 0.07, and as in their curve itself no trace of this contraction can be seen, this temperature should in my opinion be left out of consideration.

*17, 18, and 19.* Brayshaw, *Proceedings of the Institution of Mechanical Engineers* (1910, parts 1 and 2), pp. 525, 537, 656 and 670; *Engineering*, vol. lxxxix. (1910), p. 526, column 1.

No. 17 was heated to a variety of temperatures between 760° and 893°, transferred thence to a salt bath in a furnace at 731°, left there for periods ranging from 30 to 240 min., and drawn and quenched in brine at 16° or 18° C. The hardness of the quenched specimen, whether measured by the Brinell or by the scleroscope test, progressively decreased as the sojourn in the second furnace increased, till after 240 min. it had fallen nearly to the hardness of a specimen of the same steel held in the second furnace at 731° without previous higher heating, and then quenched. From the progressive decrease of hardness it is inferred that 731° must be below Ae 1, the slowness of the decrease representing the slowness with which the Ar 1 change from austenite (through martensite?) into pearlite occurs at temperatures only very slightly below Ae 1. The fact that the specimen heated to 731° and quenched thence did not harden is corroborative evidence that 731° is below Ae 1, though of little weight. The slowness of the loss of hardness during the holding at 731° is referable in part to a continuing of the fall of temperature during an important part of the holding. When the hot specimen first entered the relatively cool bath at 731° its rate of cooling would be very rapid; but as its temperature approached that of the bath its further cooling would be slower and slower, finally reaching 731° only asymptotically.

As pointed out under No. 19, Brayshaw's temperatures may be a little too high. It might be thought that his 731° was actually above Ae 1, and that the reason why his specimens quenched after progressively longer and longer stays there were progressively softer and softer was that they were progressively decarburized more and more by the salt bath. This presupposes that he did not remove the effects of surface decarburization by grinding off the decarburized layer. But this explanation should be set aside, because, if it were true, then the specimens held at 731° for various periods without previously passing above 731° also should have undergone surface decarburization, and hence should have lost hardness progressively, because the hardness of even unhardened steel decreases progressively with decreasing carbon content. But in fact the hardness of these specimens remained practically constant. Hence the inference that Brayshaw's temperature was actually below Ae 1.

No. 18. On heating to 728° and quenching was not hardened; on heating to 729.25° and quenching was hardened slightly.

No. 19. On heating to 725.5° and quenching was not hardened; on heating to 728° and quenching was hardened slightly.

These temperatures of Brayshaw's may be a little high, for Rosenhain found the maximum of Ac 1 with this same steel at 720°, and its beginning at about 715°. But Brayshaw found this steel apparently wholly unhardened when quenched in brine after a stay of 30 min. at 725.5°. This failure to harden at all after so long a stay at this higher temperature, 725.5°, is very hard to reconcile with Rosenhain's finding the maximum of Ac 1 at the lower temperature, 720°, by the thermal method, because the long sojourn of Brayshaw at stationary temperature ought to lead to a lower observed position of Ac 1 than that observed in the presumably more rapid rise of Rosenhain's thermal method. Rosenhain is in the highest degree accurate and trustworthy. In view of the evident extreme care and accuracy with which Brayshaw worked, the most probable explanation is that, though his results are closely comparable among themselves, yet his temperature scale is a few degrees too high.

*20 and 21*, cylinders $\frac{5}{8}$ in. in diameter and 0.5 in. long, treated by my assistant, A. G. Levy.

No. 20, when heated to the quenching temperature and held 30 min., then quenched, hardened when quenched from 733° but not when quenched from 725°. Brinell drop hardness No. 20 and 18 respectively.

No. 21, heated to 900°, cooled slowly to the quenching temperature, held 30 min., and then quenched, hardened when quenched from 720° but not when quenched from 710°, the Brinell hardness being 48 and 27.5, in each case by the Brinell drop test.

## *Note to Table II.*

The beginnings of Ar 1, 728° of Carpenter and Keeling and of Ac 1, 700° of Heyn, are weighted lightly because they are not well supported by thermal curves. A further objection to Carpenter and Keeling's 728° is that it coincides exactly with the beginning of Ar 1 for this same steel. In view of the great lag in the immediate vicinity of Ae 1, as proved very clearly by Brayshaw, it seems in the highest degree improbable that the beginnings of Ac 1 and of Ar 1, if found truly by the thermal method with any usual rate of heating and cooling, can coincide. On the other hand the beginnings 723° of Carpenter and Keeling and 708° of Heyn are not only well marked on the cooling curves, but are followed so soon by a very great heat evolution as to deserve great weight. Moreover, this same temperature, 723°, is the beginning of Ar 3 for two of Carpenter and Keeling's alloys. Under these conditions the fairest way seems to be to retain the extreme temperatures, 728° and 700°, but to assign them a very low weight, and to supplement them with these better supported beginnings, 723° and 708°, and to assign these a correspondingly greater weight.

The Rosenhain case may be met in like manner: Because of the greater difficulty of detecting the beginning of Ac 1, his 715° is assigned a relatively light weight, but his maximum of Ac 1, 720°, is assigned a much greater weight.

It might be thought that Carpenter and Keeling's maximum of Ar 1, 716°, also should be given weight. As to this opinions may differ. I think that its weight is already recognized in that assigned to their beginning, 723°, of Ar 1, a weight given in consideration of the good support which these maxima give it. Be that as it may, it would not influence the result of our weighting calculation, because this Ar 1 temperature is below any which the other data allow us to assign to Ae 1.

In considering the relative weight to be attached to the results of Carpenter and Keeling and of Rosenhain we should remember, on one hand, that they used very pure materials, whereas his steel had 0.40 per cent. of manganese, and that their experiments were very extensive; and on the other hand, that they did not work *in vacuo* as he did.

The value of Brayshaw's temperatures is lessened by the fact that Rosenhain, who is extremely accurate and trustworthy, found temperatures somewhat lower in operating on the same steel. Further, Brayshaw's No. 17 had 0.50 per cent. of tungsten and his No. 18 had 0.48 per cent.

The low weight assigned to the data of Charpy and Grenet is due to our lacking the original curve, and to their method of calibration, which may have lowered their observed temperatures materially.

The reasons for assigning so small a weight to the observations of so competent an observer as Benedicks are that his thermal curves themselves are not given; that his numbers are maxima, which are less good evidence than the beginnings of Ac 1 and Ar 1; and that his manganese content is rather high, 0.25 per cent.

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[SUBJECT TO REVISION.]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, October, 1913, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1913. Any discussion offered thereafter should preferably be in the form of a new paper for publication in Vol. XLVII. (with suitable cross references in both volumes).

# Determination of the Position of Ae3 in Carbon-Iron Alloys.*

BY H. M. HOWE AND A. G. LEVY, NEW YORK, N. Y.

(New York Meeting, October, 1913.)

§ 1. INTRODUCTION.—This paper gives the results of our micrographic determinations of the position of Ae3 in a series of 14 hypo-eutectoid steels of varying carbon content, one of them very rich in phosphorus, and of SEe for one hyper-eutectoid steel. We have proceeded chiefly by noting the temperature at which the reabsorption of the pro-eutectoid element is found to be complete after a 30-min. holding, but in part also by noting the temperature at which its precipitation begins. Observations on these same steels by Messrs. Burgess, Crowe, and Rawdon, of the United States Bureau of Standards, are given in a paper[1] issued simultaneously with this, while in a third paper[2] one of us compares these data with those of earlier observers. These three papers are treated as one whole by numbering their sections, tables, and illustrations continuously, and by setting the folding plates of micrographs at the end of the last paper.

THE COMPOSITION OF THE STEELS EXPERIMENTED ON is given in Table I.

§ 2. *Notation and Definitions.*—Following Osmond's development of Tschernoff's notation, in which "Ac3" is the upper limit of the hypo-eutectoid part of the transformation range as observed in heating up, and "Ar3" is that same limit as observed in cooling down, and following the usual custom of calling the upper limit of the hyper-eutectoid part of that range "SE," we may go farther and call this latter limit "SEc" when observed in heating up and "SEr" when observed in cooling down.

Further, we may call the equilibrium position of these lines, *i. e.*, the position which they would have in pure iron-carbon alloys if all

* A contribution from the Metallurgical Laboratories of Columbia University. MS. received May 14, 1913.

[1] Thermal and Microscopical Examination of Professor Howe's Standard Commercial Steels, by G. K. Burgess, J. J. Crowe, and H. S. Rawdon, U. S. Bureau of Standards, p. 1093, this *Bulletin*.

[2] Discussion of the Existing Data as to the Position of A3, H. M. Howe, p. 1099, this *Bulletin*.

TABLE I.—*Composition of the Steels Used in these Investigations.*

| No. or Mark. | Chemical Composition. C. | Si. | Mn. | P. | S. | Source. | Analyzed by | Form for Specimens. | Effect of Prolonging Heating from 30 to 60 Minutes. Lessened the Ferrite? | Made Ferrite Disappear? |
|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | | I. HYPO-EUTECTOID AND EUTECTOID STEELS. *A.* Low Manganese. | | | | |
| VI. | 0.027 | ..... | 0.26 | 0.005 | 0.024 | W. B. Kunhardt | B.[b] and M. | Rolled bar $\frac{5}{8}$ in. diam. | ..... | ..... |
| A. | 0.105 | 0.013 | 0.24 | 0.015 | 0.023 | Pittsburg Testing Laboratory | P. T. L. | Plate 1 in. by $\frac{5}{16}$ in. | ? | No. |
| IV.[a] | 0.214 | 0.039 | 0.05 | 0.013 | 0.010 | W. B. Kunhardt | B. and M. | Rolled bar $\frac{5}{8}$ in. diam. | ..... | ..... |
| IV. A. | 0.227 | 0.039 | 0.05 | 0.013 | 0.010 | W. B. Kunhardt | B. and M. | Rolled bar $\frac{5}{8}$ in. & $\frac{5}{8}$ in. diam. | ..... | ..... |
| IIII. | 0.235 | 0.039 | 0.05 | 0.013 | 0.010 | W. B. Kunhardt | B. and M. | Rolled bar $\frac{5}{8}$ in. diam. | Yes. | No. |
| II. | 0.244 | 0.050 | Nil | Nil | 0.007 | C. F. Burgess | B. | Rolled bar $\frac{5}{8}$ in. diam. | ..... | ..... |
| 61 | 0.382 | 0.027 | 0.22 | ..... | 0.004 | W. Campbell | B. and Morse[c] | Rolled bar $\frac{3}{4}$ in. diam. | ..... | ..... |
| I. | 0.40 | 0.103 | 0.16 | 0.014 | 0.012 | W. B. Kunhardt | B. | Rolled bar $\frac{5}{8}$ in. diam. | ..... | ..... |
| E. | 0.563 | 0.18 | 0.15 | 0.013 | 0.013 | J. A. Mathews | B. and M. | Rolled bar $\frac{1}{2}$ in. diam. | ..... | ..... |
| V. | 0.59 | 0.144 | 0.17 | 0.018 | 0.013 | C. S. Marshall and F. H. Daniels | B. and M. | Rolled bar $\frac{5}{8}$ in. diam. | No | No. |
| G. | 0.73 | 0.141 | 0.07 | 0.012 | 0.019 | W. Campbell | B. and M. | Rolled bar $\frac{5}{16}$ in. sq. | ..... | ..... |
| VII. | 0.92 | 0.14 | 0.123 | 0.009 | 0.011 | J. A. Mathews | B. and M. | Rolled bar $\frac{5}{8}$ in. diam. | ..... | ..... |
| | | | | | | *B.* High Manganese. | | | | |
| 125 | 0.110 | 0.004 | 0.60 | 0.036 | 0.064 | W. Campbell | B. | Rolled bar $\frac{1}{2}$ in. sq. | ..... | ..... |
| III. A. | 0.140 | 0.040 | 1.16 | 0.020 | 0.018 | W. B. Kunhardt | B. and M. | Rolled bar $\frac{5}{8}$ in. diam. | ..... | ..... |
| B. | 0.32 | 0.122 | 0.41 | 0.004 | 0.022 | Pittsburg Testing Laboratory | P. T. L. | Forging $\frac{1}{2}$ in. diam. | ..... | ..... |
| II. | 0.46 | 0.094 | 1.215 | 0.017 | 0.015 | W. B. Kunhardt | B. | Rolled bar $\frac{5}{8}$ in. diam. | ..... | ..... |
| | | | | | | *C.* High Phosphorus. | | | | |
| P. | 0.342 | 0.10 | 0.11 | 0.417 | 0.029 | W. B. Kunhardt | B. and M. | Ingot 6 by 6 in. & Rolled bar $\frac{3}{4}$ in. dia. | ..... | ..... |
| | | | | | | II. HYPER-EUTECTOID STEELS. | | | | |
| VIII. | 1.143 | 0.124 | 0.24 | 0.016 | 0.009 | J. A. Mathews | B. and M. | Rolled bar $\frac{5}{8}$ in. diam. | Yes | No. |
| IX. | 1.45 | 0.14 | 0.16 | 0.009 | 0.006 | J. A. Mathews | B. and M. | Rolled bar $\frac{5}{8}$ in. diam. | ..... | ..... |

FOOT-NOTES TO TABLE I.—[a] IV., IV. A, and IIII., are different bars of the same lot of steel, the carbon content differing slightly as shown. IV. A, was used only for the precipitation experiments represented on Plate 4, the pieces used at 910°, 880°, 850°, and 830° being about $\frac{3}{8}$ in. long by $\frac{5}{8}$ in. diam., and that used at 840°, $\frac{1}{2}$ in. long and $\frac{5}{8}$ in. diam. Steels VI., VII., and IX., were not examined microscopically. [b] B. in this column indicates Booth, Garrett & Blair; M. the makers of the steel; and P. T. L., the Pittsburg Testing Laboratory. [c] *Trans.*, xxix., 731 (1899).

The immediate reason for determining A3 was to provide a basis for our experiments on the influence of heat treatment on the tensile properties and endurance of nine steels of widely varying carbon content. In order to carry out these mechanical tests a very large quantity of each steel was needed, a quantity too great to be had in steels of extraordinary purity. Instead it was necessary to get such purity as can be had commercially. It was this consideration that determined the composition of the steels actually used. Perfectly pure steels, consisting of carbon and iron only, would have been preferred, as giving a base line of wider general interest; but as a basis for experiments in heat treating our specific steels, their own A3 was more appropriate than the A3 of pure iron-carbon steels.

The main series of steels on which these heat-treatment experiments are made consists of seven, of which five are covered by the Ae3 determinations of the present paper. In order to learn the influence of manganese and of phosphorus we have in addition experimented on two steels rich in manganese and one rich in phosphorus. Moreover, in order to fill in the gaps in Ae3 we have tested eight other steels of which only a little could be had. One of these, II, was made by Prof. C. F. Burgess from pure electrolytic iron. It is noteworthy that the results reached with this pure steel agree closely with those reached with the less pure steels which are nearest to it in carbon content.

lag were eliminated, "Ae3" and "SEe," in which "e" represents equilibrium.[3]

§ 3. *Our Procedure.—Reabsorption.* In order to detect the temperature at which the reabsorption of the ferrite apparently becomes complete, we first assembled the ferrite into coarse masses, the disappearance of which could be traced the more readily, by overheating to from 1,200° to 1,450° and cooling thence in the furnace. The structure at this time is shown in Fig. *A* of Rows 4, 5, 6, 7, 8, 9, and 17, Plates 2, 3, and 6.

We next reheated a series of such coarsened specimens of each steel to various temperatures below and above the supposed position of Ae3, so as to cause various degrees of reabsorption; held the temperature stationary for from 30 to 60 min. to give time for reabsorption to complete itself to the degree due; quenched in water to fix the *status quo;* ground the specimens very deeply or sawed them open so as to disclose the undecarburized interior; etched them so as to disclose their ferrite; and examined and photographed them microscopically, with the results shown in Figs. 1 and 3, and the micrographs of Plates 2, 3, and 6. In many cases the specimens after quenching were next reheated to 300° or 400° so as to troostitize their martensite, because the unreabsorbed ferrite can be recognized more confidently when in contrast with the black troostite than when in contrast with martensite. From our general knowledge of the subject we feel safe in assuming that no ferrite released during this troostitizing reheating can be mistaken for that remaining unreabsorbed before quenching.

Proceeding thus by stages 10° apart, we selected as our end of Ae3 a temperature between the highest at which recognizable ferrite remained and the lowest at which none remained; taking into consideration, in the exact selection of temperature, the quantity of ferrite unreabsorbed at the former temperature.

In addition we traced the course of reabsorption in steadily rising temperature as described in § 20 and shown in Row 13, Plate 5, but without record of the temperatures which the various stages of reabsorption represent, and of course without light on the temperature of A3.

§ 4. *Precipitation.*—Conversely, for three of our steels we determined Ar3, the temperature at which the precipitation of ferrite begins in cooling down, by heating a series of specimens to 1,000°, and cooling them severally to one or another of a series of tempera-

[3] See H. M. Howe, Ae1, The Equilibrium Temperature for A1 in Carbon Steel, p. 1067, this *Bulletin.*

tures in the neighborhood of Ae3, and quenching and exa
them in the way already indicated for Ac3, with the results in
in Fig. 2 and by the micrographs of Rows 10, 11, and 12, Pla

§ 5. *The Course of Reabsorption Shows Micrographically.*—Ro
Plate 2 shows the general course of reabsorption in a pre
coarsened 0.235 per cent. carbon steel, under these present
tions, of holding for 30 or 60 min. at the quenching temperat
then quenching. When the temperature has risen from Ac1
(Fig. *D*) the internal ferrite of the various grains has been so
absorbed as to make the initial network structure stand ou
prominently, though the several sides of the polygons seem t
balled up as if by surface tension. At 830°, Fig. *E*, the furt
absorption has reduced the residual ferrite to a series of isolat
apparently unrelated specks. At 850°, Fig. *F*, most of thes
disappeared, and those which remain have shrunk. At 850°
has vanished, and the appearance is that shown in Fig. *F* of E

Under the microscope the bright, slightly yellowish white, r
specks of ferrite contrast much more strongly with the surro
martensite than in these photographs, *e. g.*, Figs. *E* and *F* of
We do not think that one is likely to be mistaken for the othe
a very little practice.

In Row 6 the progress of reabsorption in another steel re
poor in manganese follows the same general course as in
though of course for given temperature the reabsorption
higher-carbon steel goes further than in the lower-carbon one
1. The progress from 780° to 800° of Row 6 is less than is d

The influence of manganese in lowering A3, first noted by O
seems to stand out well in certain micrographs but to be
elsewhere. Thus for given temperature the smaller carbon
of Row 5 should lead to a greater quantity of unreabsorbed
than is found in Row 6, as is shown by the numbers printed i
ately under the several micrographs. That the expected ex
not found we naturally refer to the greater manganese content
5, noting in Fig. 3 also that the relatively high manganese co
this specimen seems to have depressed its Ac3 materially.
Fig. *B* Row 5 seems to have at least as much excess of ferri
Row 6 as the difference in carbon content implies. But w
found these appearances very deceptive, especially on such
magnification.

Micrographs showing the progress of reabsorption in th
steels are held in the archives of the Metallurgical Departn
Columbia University. They do not differ enough from tho
shown to warrant reproducing them.

§ 6. *The Course of Precipitation Shown Micrographically*, Plate 4.—Here the temperature of the beginning of precipitation descends as the carbon content rises from 840° for 0.227 carbon to 800° for 0.40 carbon, and 765° for 0.50 carbon.

Comparing those steels for which both precipitation and reabsorption are shown, the lag seems to be very slight. Or, at least, in view of the rapidity with which the quantity of ferrite due increases as the temperature falls, as shown in Fig. 12, for given temperature the difference between the quantity of unreabsorbed ferrite left in heating up and the quantity precipitated in cooling down is within the probable limits of error of our temperatures. This Steel IV. of 0.227 per cent. of carbon has precipitated about as much ferrite at 840° as it holds unreabsorbed at 840°; and Steel I., of 0.40 carbon, has precipitated about as much ferrite at 800° as it holds unreabsorbed at 800°.

The reason why the lag is so small is evidently that before quenching the temperature was held stationary for so long a time, 30 min., giving an incomparably better opportunity for an approach to equilibrium than can be expected in the relatively rapid heating or cooling of the thermal method. It is because of the slightness of this lag that we take our results to give a reasonably close approximation to the true position of Ae3.

The last of the residual ferrite in the reabsorption tests occurs in a few relatively large grains, but the first of the ferrite to precipitate in the precipitation tests occurs in many and very minute ones, probably because at the moderate temperature to which these were heated before cooling for precipitation, a very large number of crystallization centers persisted, at each of which the precipitation of ferrite started independently in the cooling. Our first trials, heating to from 850° to 900° before cooling for precipitation, gave us results hard to interpret. We then adopted 1,000° as a convenient temperature from which to start the cooling, but a higher temperature, by destroying more of the crystalline centers,[4] might have yielded ferrite in fewer and larger spots.

§ 7. *Graphical Representation.—Steels poor in manganese.* Fig. 1 shows the results of our reabsorption tests for steels poor in manganese, summed up by the line *HLM*, which is reproduced in the other graphical diagrams for comparison. The circular black spots indicate temperatures at which we found unreabsorbed ferrite, the white-centered circles those at which we found none. Note that the very pure electrolytic steel of 0.244 carbon agrees with the slightly less pure

[4] Cf. Benedicks, *Proceedings of the Institution of Mechanical Engineers*, vol. —, p. 703 (May, 1910).

steel of about 0.22 carbon. The thermal results of Messrs. B and Crowe with these same steels are shown by means of S drew's crosses, " ×," and plus marks, " +."

In view of the probability that the eutectoid point is at 0.90 carbon and 725°, and that the break at *L* probably at about 760°, representing Ae2, the line *HLM* represen results closely. Thus in 6 out of our 9 hypo-eutectoid steel line lies within the 10° step in which we found that the reabso of ferrite became complete; in two of the remaining steels it or 3° outside this step; and in the last one it lies 7° below this

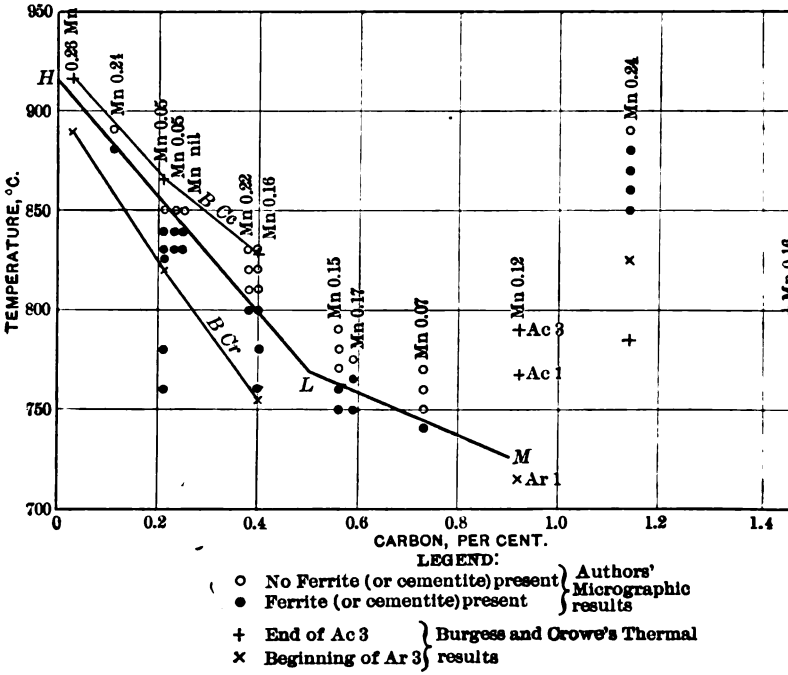


FIG. 1.—THE AUTHORS' MICROGRAPHIC REABSORPTION DETERMINATIONS OF LOW-MANGANESE STEELS, TOGETHER WITH THE THERMAL DETERMINAT A3 BY BURGESS AND CROWE ON THE SAME STEELS.

The agreement is closer than is to be expected under our cond in view of the many sources of error, (1) decarburization, (2) p ence of solidificational segregation, (3) persistence of trans tional segregation, (4) precipitation of " quenching " ferrite,[5] (5) to detect the last traces of unreabsorbed ferrite, and (6) te ture errors from various sources. The first four of these act latively to raise the observed temperature of A3, and the sixt act in the same direction. (See § 31, p. 1112.)

[5] Ferrite which precipitates during the quenching, as distinguished from that le absorbed or precipitated at the holding temperature before quenching. (See p. 10

§ 8. *A Shortage of Recognizable Ferrite* is apparent on comparing the quantity seen in each of the micrographs of Rows 4, 5, and 6, with that written immediately under each figure, and due on the theory that *HLM*[a] is really Ae3. Heyn[6] noted a like but smaller shortage of ferrite in his specimens quenched within the transformation range.

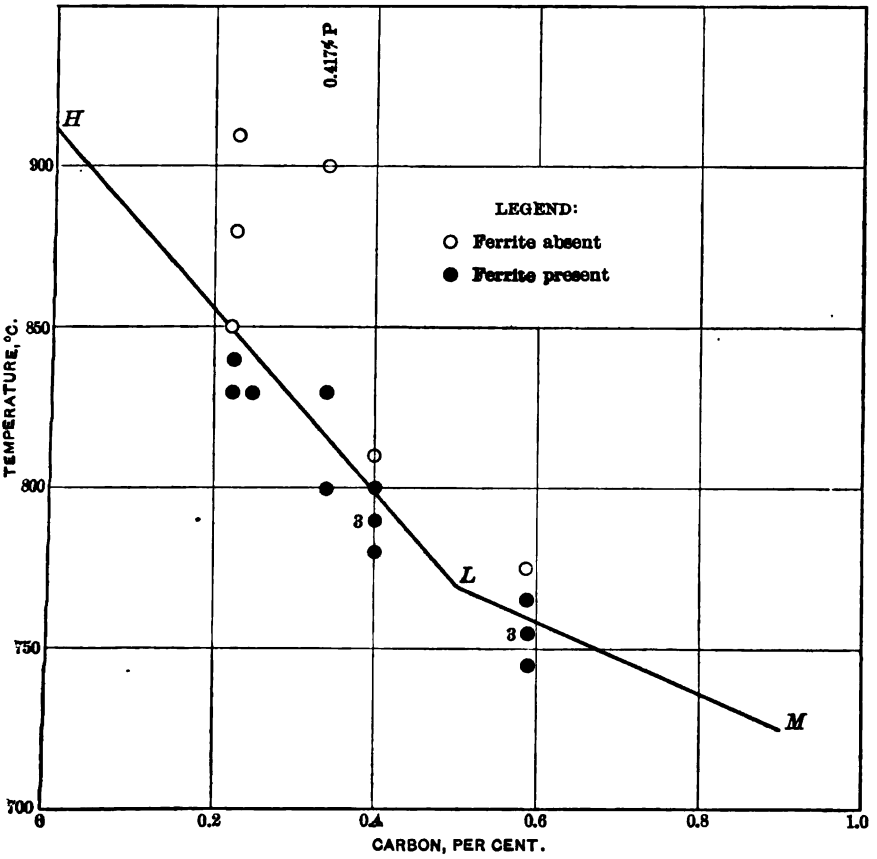


FIG. 2.—THE AUTHORS' MICROGRAPHIC PRECIPITATION DETERMINATIONS OF AR3 IN LOW-MANGANESE STEELS.

Note.—The line *HLM* is reproduced from Fig. 1 for comparison.

§ 9. *Precipitation.*—Though, as shown in Fig. 2, the temperature of Ar3 as found for the three steels experimented on by precipitation and that of Ac3 as found by reabsorption, fall within the same 10° step, yet the results by precipitation were less concordant among themselves than those by reabsorption.

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[a] Or more accurately *AAA*. See § 24 and Fig. 14.

[6] *Verhandlungen des Vereins zur Beförderung des Gewerbfleisses*, p. 378 (1904).

§ 10. *Comparison of Our Micrographic Results with the Thermal Data of Burgess and Crowe.*—Because Ae3 is the upper limit of the transformation range in equilibrium, the thermal data which give the most nearly direct evidence as to its position are the beginning of Ar3 and the end of Ac3, for each of these shows under its own peculiar conditions the actual upper limit of that range, Ar3 for falling and Ac3 for rising temperature. The maxima of Ar3 and Ac3 represent simply the temperatures at which the prior lag of the transformation is wiped out most rapidly, causing temporary maxima of retardation. As to the beginning of Ac3 and the end of Ar3, strictly speaking these coincide with Ac1 and Ar1 respectively; but as habitually reported they represent simply the temperatures at which the breaking up of lag becomes sufficiently accelerated to cause a noticeable retardation of the rise or fall in temperature. Hence for the purpose of checking up the micrographic results, the beginning of Ar3 and the end of Ac3 alone among the thermal data need attention.

These, plotted in Fig. 1, show great lag, with a gap of from 38° to 74° between the beginning of Ar3 and the end of Ac3. The unusual width of this gap may represent unusual sensitiveness in the determinations of the end of Ac3. Our line *HLM* lies from about one-third to about one-fourth way down from the upper edge of this gap. Hence if it is a close approximation to Ae3, this implies that the lag in the thermal observation in cooling is greater than in heating up, as indeed it should be, for well-known reasons.

§ 11. *Steels Rich in Manganese.*—Fig. 3 shows our micrographic results with four steels rich in manganese, and the thermal results of Burgess and Crowe with two of these steels, using the symbols already explained for Fig. 1, from which the lines representing the micrographic position of Ae3, the beginning of thermal Ar3, and the end of thermal Ac3 for our low-manganese steels are reproduced.

Though manganese had but little effect on the position of Ac3 as we found it micrographically, it lowered materially the thermal position as found by Burgess and Crowe, the end of Ac3 by about 30° in both cases, and the beginning of Ar3 by about 60° in one case (manganese 1.16 per cent.), and in the other case (manganese 1.21) by a smaller quantity, which cannot be measured even approximately without rather rash extrapolation.

§ 12. *Acknowledgments.*—This investigation was made in the Metallurgical Laboratories of Columbia University, in part under a grant from the Carnegie Institution of Washington. Our thanks are due to Messrs. W. B. Kunhardt of the Carpenter Steel Co., J. A. Mathews of the Halcomb Steel Co., C. S. Marshall and F. H. Daniels of the

American Steel & Wire Co., Profs. W. Campbell of Columbia University and C. F. Burgess of the University of Wisconsin, and the Pittsburg Testing Laboratory for the specimens of steel; to Messrs. Booth, Garrett & Blair and the Pittsburg Testing Laboratory for the chemical analyses; and to Prof. William Campbell of Columbia University for counsel.

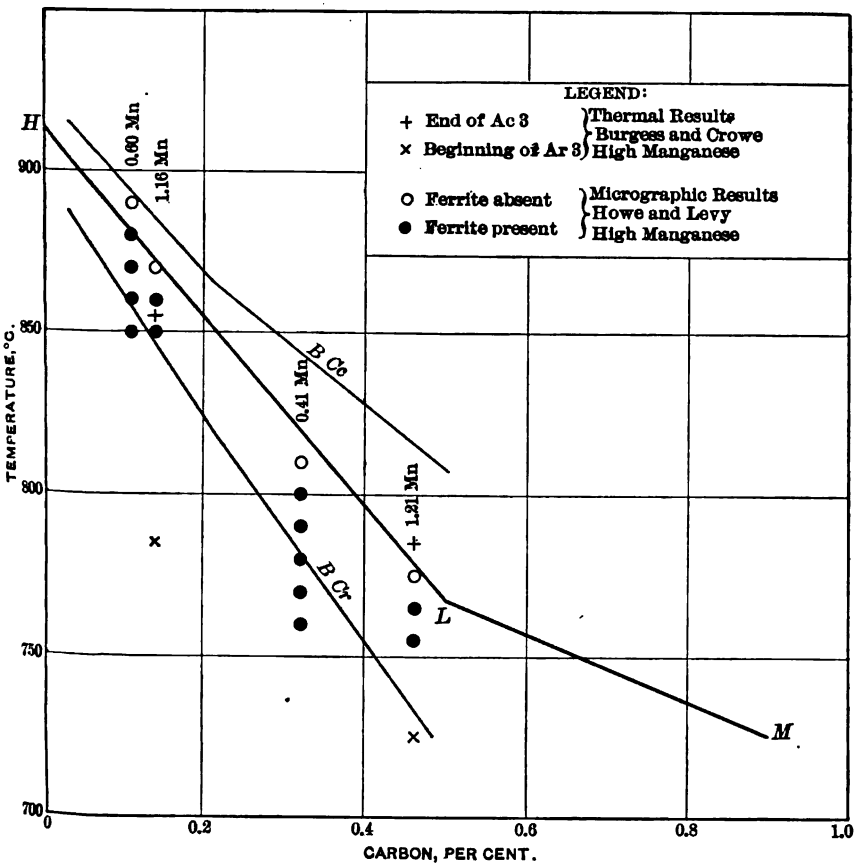


FIG. 3.—HIGH-MANGANESE STEELS. THERMAL A3 OF BURGESS AND CROWE AND MICROGRAPHIC A3 OF THE AUTHORS.

Note.—The lines *BCr*, *BCr*, and *HLM* are reproduced from Fig. 1 for comparison.

To Messrs. Kunhardt and Burgess our obligations are unusually great, for they purposely made special products for us. Mr. Kunhardt's task of rolling this very phosphoric steel into a 12-ft. bar $\frac{5}{8}$ in. round must have been a serious one.

## SUMMARY.

1. We determine micrographic Ac3 and Ar3 in a series of 14 steels, of which 4 are rich in manganese, by quenching from a series

of temperatures in and above the transformation range, and n
that at which the reabsorption of ferrite is complete in heati
and that at which the precipitation of ferrite begins in cooling
Line *HLM*, Fig. 1, p. 1080, sums up our results.

2. Like Heyn, we find in specimens quenched within the
formation range a shortage of recognizable ferrite below that
the position of our line *HLM* calls for. (§ 8, p. 1081.)

3. The results obtained by reabsorption and by precipitation
but the precipitation evidence is less harmonious than the rea
tion. (§ 9, p. 1081.)

4. The micrographic results agree with the thermal ones re
by Burgess and Crowe. (§ 10, p. 1082.)

5. When the ferrite is reabsorbed during steadily rising ten
ture, its network preserves its continuity better than when rea
tion occurs at constant temperature. (§ 20, p. 1089.)

6. Phosphorus raises A3, at least in unforged castings (§ 21, p.

## Appendix 1.

§ 13. *The Initial Coarsening*, for facilitating the detection of t
of the reabsorption of ferrite, was done by heating the specimen
ally ⅝ in. in diameter and from 2 to 4 in. long, to about 1,200°
molten barium chloride in a graphite crucible, heated in a g
nace, within which the crucible and the specimens were
slowly.

From these coarsened specimens wads about ⅝ in. in diamet
0.5 in. long were cut for the reabsorption treatment. This p
nary coarsening treatment was not applied to the specimens t
by precipitation, except in the case of the phosphoric steel.

*Reabsorption.*—The various coarsened specimens were then re
to various temperatures, some believed to be below and some
Ae3, held there for 30 min., and immediately drawn and quenc
cold water.

The reabsorption and precipitation heatings were carried
the apparatus shown in Fig. 4.

The specimen *A* lay in a bath, *B*, of molten chlorides of s
and potassium, or, for the higher temperatures, of chlorides of
and potassium, in a graphite crucible, *C*, 1⅝ in. high and 1.5
diameter at the top, which stood in the middle of the lengt
horizontal steel pipe, *D*, 2.5 in. inside diameter and 30 in
within an alundum tube, *E*, on which a spiral of nichrome
resistance wire, *F*, was wound. This outer tube was encased
in insulating material, *G*. The thermo-junction *J* lay imme
above the crucible, and rested on it.

In order to carry still further the protection against surface decarburization which the bath of fused chlorides offered, an abundance of charcoal was set within the inner tube *D*.

In order to flatten the thermal gradient by lessening end cooling, clay plugs, *H*,*H*, were set within the inner tube about half way between the ends and the middle, and both ends were closed with caps. The thermal gradient of the furnace at 960° is shown by the following determinations to be very flat.

| To the left. | | Middle of length. | To the right. | |
|---|---|---|---|---|
| 2 in. | 1 in. | | 1 in. | 2 in. |
| 958° | 960° | 962° | 959° | 955° |

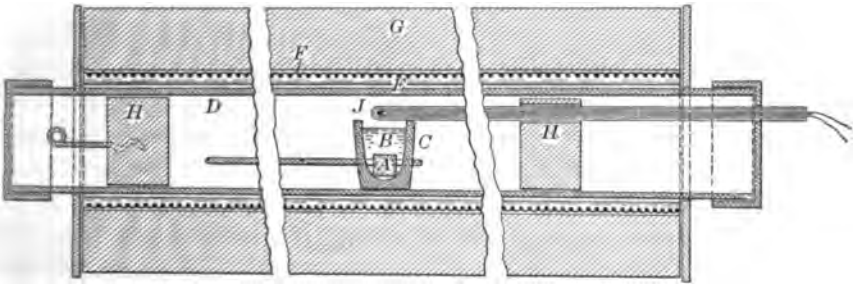


FIG. 4.—TUBULAR ELECTRIC RESISTANCE FURNACE USED FOR MOST OF THE REABSORPTION AND PRECIPITATION EXPERIMENTS.

*A*, Specimen.
*B*, Bath of molten chlorides.
*C*, Graphite crucible, 1⅝ in. high, 1.5 in. diam.
*D*, Steel pipe, 2.5 in. diam.
*E*, Alundum tube.
*F*, Nichrome resistance wire.
*G*, Insulating material.
*HH*, Clay plugs.
*J*, Fused quartz tube containing the thermo-junction.

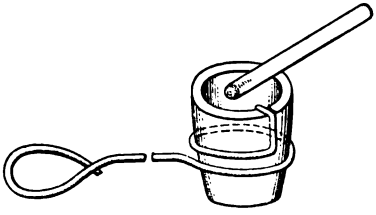

FIG. 5.—DETAILS OF CRUCIBLE AND QUENCHING WIRE.

In order that the crucible could be withdrawn quickly and inverted so as to throw the specimen into the quenching water, it was held in a wire coil as shown in Fig. 5.

Having first brought the middle of the furnace and the crucible with its molten chloride bath to a little below the temperature aimed at, the crucible was withdrawn to receive the specimen to be treated, and immediately replaced exactly beneath the tip of the thermo-

couple. In the reabsorption experiments the crucible and sp were then heated to the desired temperature, held there for 3 drawn, and emptied quickly into a pail of cold water close b nearly every case the temperature of the thermo-junction rose, by from 2° to 5°, on drawing and quenching. Though this ri represent a slight lagging of the temperature of the specimen that of the thermo-junction, we believe that it represents chi accelerated combustion of the charcoal present within the tube by the inrush of air, for on opening the furnace in like manne out withdrawing the specimen a like rise of temperature occur

In a few cases we used a vertical cylindrical muffle, 5 in. in eter inside and 28 in. high, heated by the resistance of a wire wound about it, setting the specimen in a chloride bath in crucible 4.25 in. in diameter and 5 in. high which stood muffle. Here the thermo-junction lay beside the specimen. E arrangement was soon abandoned because it gave slightly le cordant results. The advantage of having the thermo-junction the salt bath beside the specimen seems to have been outweig the steeper thermal gradient within a vertical than within a h tal furnace. In a vertical furnace the air cooled by contact w cooler top is in a position to fall like a douche on the surface bath within the crucible. Moreover, if the thermo-junction lies the chloride bath with the specimen in a vertical furnace, it is to make sure that, at the very moment of drawing and quen the specimen is truly at the temperature last recorded, than thermo-junction lies immediately above the bath in a hor furnace.

The galvanometer and the thermo-junction were calibrated Bureau of Standards at Washington, and repeatedly re-calibra the melting points of antimony and copper.

§ 14. *Does a 30-Min. Holding Suffice?*—In the case of st IIII, V, VIII, and II, after examining the specimens so as t the highest temperature after quenching from which the pro-eu element could still be detected, another specimen of that st next held at this same temperature for 60 instead of 30 min., i to learn whether prolonging the exposure to this temperature cause this element to disappear, or, in other words, to learn w the persistence of the element at this temperature at the end min. was due to insufficient opportunity for diffusion, or whe really meant that the temperature was below Ae3.

In three cases this prolonging of the holding had no effect though in two cases it seemed to lessen very slightly the quan

residual ferrite, the temperature-equivalent of this diminution was so slight that any merit of this prolongation in this respect seemed to us balanced by the greater danger of inaccuracy through uncontrolled temperature oscillations.

*Precipitation Experiments.*—These were carried out in the general way described for the reabsorption experiments. The bath of chlorides having been melted, the specimen was immersed in it, crucible and specimen were set within the horizontal nichrome furnace shown in Fig. 4, raised to 1,000°, held there for 10 min., cooled within the furnace to the quenching temperature, held there for 30 min., and then drawn and quenched in cold water, as already described.

*Repetition Heatings.*—In eight cases the reabsorption tests were confirmed by repetition, heating additional specimens independently either to the highest temperature at which unreabsorbed ferrite had been detected, or to the lowest at which none had been, or to both, and quenching as before. The results were concordant in every case.

Repetitions of the precipitation experiments, though, as might have been expected from the greater lag in cooling, they were less concordant than those of the reabsorption tests, yet in no case contradicted the initial results. They were of value in interpreting them.

§ 15. *Surface Decarburization* took place, probably during the initial coarsening. It is shown by the greater quantity of ferrite at the outside than in the interior of the specimens. Compare the outside of specimen IV, 93, Fig. *H*, Row 14, Plate 5, with its interior, Fig. *G*, Row 14, Plate 5. It is because of this tendency to decarburization that the specimens were made so thick. After the heat treatment and before etching, these wads were ground down ⅛ in. or sawed open in order to give access to their undecarburized interior.

But though in the various steels, when the quenching temperature had risen just high enough to leave no visible unreabsorbed ferrite in the interior of the specimen, some ferrite remained in the superficial layer, proving that there was superficial decarburization, yet none was found there in the specimen quenched from 10° higher. This disappearance of the skin ferrite at a temperature so slightly above the disappearance of the internal ferrite argues for the consistency of our results, though of course it throws no light on the accuracy of the method itself.

16. *Etching.*—The purpose of the etching, to detect the pro-eutectoid ferrite, or in the case of steel No. VIII the pro-eutectoid cementite, could be served best by darkening the martensite or troostite which formed the rest of the field, leaving the pro-eutectoid element

light. To that end the etching was done with nitric acid, u solution of 2 per cent. of acid of 1.4 sp. gr. in water, and an i sion of from 5 to 40 sec. or till the martensite or troostite turne yellow to black. This treatment affected the pro-eutectoid el but slightly. In order to clean the specimen and enable the acid to act evenly, a preliminary etching of about 30 sec. was with a solution of 5 per cent. of picric acid in alcohol. This p nary etching or cleaning was, on the whole, better done wit solution than with the others tried, such as alcohol alone, and or hydrochloric acid much diluted with alcohol. On leavin etching bath the specimen was washed with running water, an out drying at once immersed in the aqueous nitric acid.

Instead of nitric acid, alcoholic hydrochloric acid was trie with this the etching is much slower, needing from 5 to 15 mi the contrast between the pro-eutectoid element and the mart etc., is on the whole less clear.

§ 17. *Troostitizing.*—In many cases the pro-eutectoid e sought was made the more prominent by converting the rest field wholly into troostite by a reheating to 300° or 400° befor ing. As may be seen in Figs. *B*, *C*, and *D* of Row 4, Plate Fig. *A* of Rows 10, 11, and 12, Plate 4, this troostite darkens so on etching that the bright ferrite stands out in very strong co Were we to repeat this work we should thus troostitize specimens.

Thus the etching was directed strictly to the point imme before us, creating a very strong contrast between the unreab ferrite and all the remainder of the field, and sacrificing delib all indications of the structure of this remainder.

§ 18. *Troostitic Areas.*—Very often the interior of a sp quenched within the transformation range etched much dark the outside before troostitizing, as shown in Fig. *F*, Row 14 5. Here and in other cases the darkest region lay part way b the skin and the axis, and was neither circular nor concentr the specimen itself. These facts explain themselves as the re of two causes. In that the interior cools more slowly than t side, it tends to transform further during the quenching th outside. Were this the sole cause, then the darkest etching be at the axis, because it is unlikely that any part has trans beyond the osmondite stage of darkest etching. That this etching is not at the axis indicates that the varying pressure the specimen has had a part in determining its position, on th ciple that pressure retards the transformation. That it is n

centric with the specimen indicates that the conditions of quenching have led to an unbalanced escape of heat from the surface of the specimen, as indeed must occur if, in entering the bath, its axis is not strictly vertical, or indeed if there is any eccentricity in the path of the escaping bubbles of steam.

This darker etching of the interior was noticed by Heyn.[7]

§ 19. *Quenching Ferrite.*—In certain cases a very fine network occurred within the quenched specimens, as in Fig. *E*, Row 14, Plate 5, representing Steel I (carbon 0.40), heated to 810°, or to the top of the 10° step in which the reabsorption of ferrite becomes complete, held 60 min., and quenched. The other cases in which this network was noticed were as follows: Steel A, carbon 0.105 per cent., quenched from 890° and 880°; Steel IV, carbon 0.214 per cent., quenched from 840°; and Steel I, carbon 0.40 per cent., quenched from 810°; all after 30 min. holding. This fine network is very distinct from the nearly equiaxed spots of ferrite which gradually decrease as the quenching temperature rises towards *HLM*, and increase as it is lowered from *HLM*. This may be "quenching ferrite," *i. e.*, that precipitated during the quenching, though the time available is rather short for such a coalescence.

## Appendix 2.

§ 20. *The Course of Reabsorption with Steadily Rising Temperature.*—The micrographs of Row 13, Plate 5, represent a series of points in the length of one and the same piece of steel, I, of 0.40 per cent. of carbon, after a differential treatment which completed the reabsorption of pro-eutectoid ferrite in one end of the piece without allowing it to begin in the other end. In order to bring this about the piece was overheated to 1,350° and cooled slowly so as to assemble the ferrite together into coarse compact masses easy to recognize. Fig. *A* shows the structure of the coarsened bar at this stage. Successive stages in the reabsorption of the ferrite thus massed were established by heating the bar differentially so as to create a flat thermal gradient along its length, running through the transformation range. This was done by immersing one end in a bath of molten potassium sulphate and allowing the other to project into the air, under a suitable cover. These stages were then fixed by quenching the piece in cold water, and they were then revealed by etching the bar and photographing a series of points along its length, with the results shown in Row 13.

[7] *Verhandlungen des Vereins zur Beförderung des Gewerbfleisses*, p. 380 and Fig. 152 on Plate 17 (1904).

Fig. *B* shows, in its left-hand part, a polygonal grain which not passed above Ac1, and therefore is in the condition of ferrite pearlite, of which the former appears as the light network and spi the latter as the dark ground mass. The light-colored polygon the right of this dark one had passed above Ac1, and therefore a moment of quenching consisted of the unreabsorbed ferrite, sh still in white spines and network, and the austenite into which pearlite had changed, here appearing as a martensite ground which, though slightly less light than the ferrite, is yet far lig than the pearlite in the left-hand grain.

The successive figures *C* to *E* show stages in the further reab tion of the ferrite. First the interior or cleavage ferrite disapp then the network becomes discontinuous, and in Fig. *E* disapp In some parts the network which the ferrite had formed can be tr beyond where the ferrite itself vanishes, but this darkening seem be only a shadow effect, because under proper illumination the bo of the grooves which form the grain boundaries appears of the pearly white as the interior of the grains.

The progressive increase from Fig. *F* to Fig. *H* of the light tensite and decrease of the dark troostite represent the increasin of the transformation from the stage of martensite to that of troo an increase which the rise of the quenching temperature natu induces.[8] The white spottings at the right-hand part of Fig. Row 13 indicate a possible cause of error. These occur above w the absorption of the ferrite network has become complete, and presumably well above Ae3. If such spots should be noticed in ing down, they might well be taken for an early precipitation o rite, and thus the line A3 might be set not only too high but pe very much too high.

A careful study of the white constituent which forms the gr mass of the polygons in Figs. *C*, *D*, and *E* leaves no doubt tha martensite.

One asks why this cooler and hence less lagging part of the men should be caught in the stage of martensite though the part presented by Fig. *F* has transformed beyond into that of t ite, common report being that of the two it is quenching from A3 that yields martensite and quenching from the lower part transformation range that yields troostite. Two reasons why ma ite should be found in the cooler but not in the hotter parts su themselves: 1st, at the moment of immersion the austenite i

[8] See H. M. Howe, Why Does Lag Increase with the Temperature from which Q ing Starts? *Bulletin* No. 75, Mar., 1913, p. 479.

parts just above A1 was a higher carbon austenite than that in the hotter parts, and it is most familiar that an increase of carbon content increases lag. 2d, in quenching it was the hot end of the piece that entered the water first, and hence in its rapid descent through the water it was less surrounded by steam than the cooler parts above it. A repetition of this experiment gave like results.

The cleavage ferrite is often much more abundant near the boundaries of a given polygon than in the central part of it, suggesting that this cleavage ferrite has been trapped here in the course of its migration to the grain boundaries.

The shorter opportunity for balling up by surface tension explains why the ferrite network remains continuous to a further stage in its reabsorption here in rising temperature, than when it is reabsorbed at stationary temperature as in Row 6, Plate 2.

## Appendix 3.

§ 21. *Phosphorus Raises A3.*—One of us had inferred that phosphorus raises the temperature of A3, from the behavior of phosphoric steel, and more particularly from Stead's important discovery that though, as long as the solidificational segregation of phosphorus persists the slag inclusions always occur within pro-eutectoid ferrite islands, they do not if that segregation is effaced by diffusion.[9]

In view of later observations it is more accurate to say that, after the phosphorus has been diffused, each slag inclosure has at most a narrow reef of ferrite about it, instead of being inclosed in a large ferrite island. (J. E. Stead, Private communication, Mar. 4, 1913.)

This inference is confirmed by the micrographs of Plate 6. Here the presence of 0.417 per cent. of phosphorus has raised A3 from about 820°, its inferred position for this carbon content, 0.34 per cent., in the absence of phosphorus, to above 950° if the initial ingot structure is not removed, and to above 870° if it is removed. The specimens here represented were treated for reabsorption of ferrite (Rows 15 and 16), and for precipitation (Row 17), substantially like the steels represented on Plates 2, 3, and 4, and as set forth in Appendix 1, except that the specimens cut from the ingot were not coarsened before the reabsorption experiments, and that before the precipitation tests the rolled bar was coarsened.

Some of the light particles in *F*, Row 15, representing the unforged ingot, are evidently ferrite which has persisted unreabsorbed even at 950°.

[9] Howe, discussion of J. E. Stead's paper, *Proceedings of the International Association for Testing Materials*, VI Congress, III, to appear.

When the initial ingot structure, *A* of Row 15, is refined b ing and rolling to that shown in *A* of Rows 16 and 17, the el of A3 caused by phosphorus, though less, is yet very marked. even at 870°, or 50° above the normal position of A3 for this content, great masses of unreabsorbed ferrite may remain, as in *E* and *F* of Row 16. After the temperature has been raise enough to complete the reabsorption of ferrite and is again lo precipitation is very marked at 830° (*D*, Row 17). The lag great, the quantity of ferrite precipitated in a 30-min. stay being incomparably less than that unreabsorbed in a like stay (*F*, Row 16).

The black-centered islets in *D* and *E* of Row 15, the media sures in the network of *F*, Row 16, the rectangular ingot st in *A*, *B*, and *C*, Row 15, and the dark bands about the ferr work of *C* and *F*, Row 16, are of interest, as are also the sugg of ingot structures in the dark and light bandings of *D*, E and even in the fine-grained *A*, Rows 16 and 17, and the ence in parts of Fig. *A*, Row 15, of four pretty distinct co parts, the dark phosphoric nuclei, the light gray bands round the darker gray beyond, and outside of all the dark pearlitic

Though the rolled bar when re-coarsened (*B*, Rows 16 looks at first much like an unphosphoric steel of this carbon thus coarsened (*A*, Row 5, Plate 2), yet it has some pecul Note the dark parting between some of the polygon boundar the internal ferrite, recalling *C* and *F* of Row 16. It may b than a fancy that the network is more continuous, smoot more rounded than in unphosphoric steel, and the Widman structure, though recognizable, yet less marked than would pected in view of the very high temperature of coarsening, a t ture so high that part of the specimen melted. Evidence network in this coarsened piece is really unusually smooth a tinuous is given by the certainly unusual continuity and sm of the network after much reabsorption, as in *C* and *F*, Row

The summary of this paper will be found on pp. 1083 and

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, October, 1913, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1913. Any discussion offered thereafter should preferably be in the form of a new paper for publication in Vol. XLVII. (with suitable cross references in both volumes).

# Thermal and Microscopical Examination of Professor Howe's Standard Commercial Steels.*

BY G. K. BURGESS, J. J. CROWE, AND H. S. RAWDON, U. S. BUREAU OF STANDARDS, WASHINGTON, D. C.

(New York Meeting, October, 1913.)

## I. Cooling and Heating Curves.

BY G. K. BURGESS AND J. J. CROWE.

§ 22. The results published in Professor Howe's paper[10] of our determinations on the Ac3 and Ar3 points for a series of commercial carbon steels[11] containing manganese in varying proportion, represent the average temperatures found by taking several (4 or 5) heating and cooling curves.

The accompanying curves, Figs. 6 and 7, are, for each steel, the first heating and cooling curves as obtained by the inverse-rate method of Osmond, using a cylindrical chronograph sensitive to 0.1 sec. for recording time. Simultaneous observations were also taken by the differential method with a platinum neutral and plotted by Rosenhain's derived differential method and gave substantially the same results, the curves having sensibly the same contours in both cases.[12]

The observations were taken *in vacuo* on cylinders $\frac{5}{8}$ by 1 in. in length bored to center with a $\frac{1}{8}$-in. hole to receive porcelain insulated thermo-couple, and inclosed in an out- and in-glazed Berlin porcelain tube heated electrically by platinum foil wound on the tube. The vacuum was usually better than 0.1 mm. The heating and cooling of the furnace was automatically controlled by a salt-water rheostat with shaped electrodes, giving a uniform rate of heating and cooling.

* MS. received May 14, 1913.

[10] H. M. Howe, Life History of Network and Ferrite Grains in Carbon Steel. *Proceedings of the American Society for Testing Materials*, vol. xi., pp. 262 to 386 (1911). See pp. 330 to 332.

[11] The specimens sent to the Bureau of Standards were untreated sections cut from the rods of the various steels, in the condition in which they were received from the makers. H. and L.

[12] For a comparison of cooling curve methods see G. K. Burgess, *Bulletin No. 5, U. S. Bureau of Standards*, pp. 199 to 225 (1908).

Alternating current was used from a motor-generator driv storage battery, insuring the greatest steadiness in the heati rent. Contrary to the usual practice, the furnace had almost lating covering, so that it responded promptly to impressed qu of heat. This arrangement also facilitates running the furnace from high to low temperatures.

TABLE II.—*Determination by Burgess and Crowe of the Transf Points of the Steels of Howe and Levy, by Means of Hea and Cooling Curves.*[c]

| | | Low-Manganese Steels. | | | | | | High-M St |
|---|---|---|---|---|---|---|---|---|
| No. | | VI. | IV. | I. | VII. | VIII. | IX. | III. |
| | 1[a] | 743±10 | 754±5 | 754±5 | 761±5 | 746±10 | 751±15 | 738±1 |
| Ac1 | 2 | 755±5 | 762±5 | 757±5 | 768±5 | 760±10 | 759±20 | 745±1 |
| | 3 | 762±5 | 777±10 | 770±10 | ? | 774±10 | 772±20 | 761±1 |
| | 1 | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| Ac2 | 2 | 784±10 | 786±10 | ..... | ..... | ..... | ..... | 776±1 |
| | 3 | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| | 1 | ? | ? | ? | ? | ? | ? | ? |
| Ac3 | 2 | 901±5 | 852±20 | ? | 778±5[b] | 780±15[b] | 772 ?[b] | 836±1 |
| | 3 | 916±5 | 865±15 | 828 10 | 790±5 | 786±15 | 800 ? | 854±2 |
| | 1 | 889±10 | 820±10 | 754±5 | ..... | ..... | ..... | 786±1 |
| Ar3 | 2 | 870±5 | 796±15 | 712±5 | ? | 770±15[b] | ? | 763±1 |
| | 3 | ..... | ..... | ? | ..... | ..... | ..... | ..... |
| | 1 | 778±5 | ? | ..... | ..... | ..... | ..... | ? |
| Ar2 | 2 | 765±5 | ..... | ..... | ..... | ..... | ..... | ..... |
| | 3 | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| | 1 | 694? | 682±15 | 695±2 | 707±15 | 708±10 | 712±5 | 630±3 |
| Ar1 | 2 | 671±5 | 678±15 | 688±2 | 700±15 | 704±10 | 708±5 | 620±5 |
| | 3 | 668±20 | 666±20 | 666±5 | 662±20 | 666±10 | 678±5 | 600±1 |
| C | ..... | 0.03 | 0.21 | 0.40 | 0.92 | 1.14 | 1.45 | 0.14 |
| Si | ..... | ..... | 0.039 | 0.103 | 0.123 | 0.124 | 0.14 | 0.040 |
| Mn | ..... | 0.26 | 0.05 | 0.16 | 0.14 | 0.24 | 0.16 | 1.16 |
| P | ..... | 0.005 | 0.013 | 0.014 | 0.009 | 0.016 | 0.009 | 0.020 |
| S | ..... | 0.024 | 0.010 | 0.012 | 0.011 | 0.009 | 0.006 | 0.018 |

FOOT NOTE *a*.—1 = Beginning, 2 = maximum, 3 = end.

*b* These very faintly marked points may be A2 points.

*c* These determinations differ in only a few points from those which we gave Howe for his paper The Life History of Network and Ferrite Grains in Car *Proceedings of the American Society for Testing Materials*, vol. xi., p. 262 (1911).

In the figures, the ordinates are e.m.f.'s of the platinum- thermo-couple used, and also the corresponding temperatur grade of the sample; the abscissæ for each specimen are seconds for the specimen to cool 2° C. Each dot represent ual observation. A vertical line indicates a uniform rate of

or heating. The furnace was checked out with a platinum blank, also shown. For each specimen the times of heating and cooling for the plotted temperature range are indicated. The direction of the arrows shows whether it is a cooling or heating curve.

These curves appear to be fairly representative of what may be expected for commercial carbon steels containing manganese. The impurities appear to mask somewhat the sharpness of the A3 points and, when present, of the A2 points. The effect of increasing the manganese, as has been shown elsewhere, is to lower the A1 points and further separate Ar1 and Ac1.

The average thermal behavior, together with their analyses as given by Professor Howe, is shown in the accompanying table for each of these steels.

## II. Microscopical Examination for Decarburization Upon Heating.

BY H. S. RAWDON.

§ 23. *Sample No. 1.*—Carbon 0.40 per cent.

Thermal treatment: Heated to 978° C., cooled in furnace. Reheated, heated to 1,005° C.

Sections examined:

(*a*) End of cylinder used for cooling curve.
(*b*) Section 5 mm. below end.
(*c*) Section from original unheated rod.

Results: Estimated carbon by amount of precipitated pearlite, assuming pearlite contains 0.85 per cent. of carbon.

(*a*) C, — 0.20 to 0.30 per cent. Average C, 0.23 per cent.
(*b*) C, — 0.22 to 0.48 per cent. Average C, 0.30 per cent.
(*c*) C, — 0.34 to 0.38 per cent. Average C, 0.36 per cent.

Average C, 0.27 per cent. for (*a*), (*b*), and (*c*).

A *very thin* skin around the longitudinal hole in the cylinder for the thermo-couple was entirely decarburized.[13] No *systematic* variation in carbon content could be noted.

*Sample No. 2.*—Carbon. 0.46 per cent.

Thermal treatment: Heated to 972° C., cooled in furnace. Reheated, heated to 974° C.

Sections examined:

(*a*) End of cylinder used for cooling curve.
(*b*) 4 mm. below (*a*).
(*c*) 4 mm. below (*b*).
(*d*) From original unheated rod.

[13] This decarburization must have occurred in the heatings *in vacuo* at the Bureau of Standards, as the specimens sent there had undergone no heat treatment.—H. & L.

Results:

(*a*) C, 0.18 to 0.50 per cent. Average C, 0.32 per cent.
(*b*) C, 0.75 per cent.
(*c*) C, practically same as (*b*).
(*d*) C, practically same as (*b*).

(In (*c*) and (*d*) pearlite area was not measured, but sectio compared with (*b*)).

Under high magnification the pearlite was not of the lame but suggested sorbite. Carbon seems much higher than analysis calls for; may be due to the action of high mangane

*Sample No. 3.*—Carbon 0.16 per cent.

Thermal treatment: Heated to 978° C., cooled in furnace.

Sections examined:

(*a*) End of cylinder used for cooling-curve determinatio
(*b*) End of cylinder used for cooling-curve determinat deep grinding.

Results:

(*a*) C, 0.17 per cent.
(*b*) C, 0.13 to 0.19 per cent. Average C, 0.16 per cent.

Agrees with the chemical analysis (C, 0.16 per cent.).

*Sample No. 4.*—Carbon 0.21 per cent.

Thermal treatment: Heated to 978° C., cooled in furna peated, heated to 976° C.

Sections examined:

(*a*) End of cylinder used for cooling-curve determinatio
(*b*) Other end of cylinder used for cooling-curve deter after deep grinding.
(*c*) Original rod before heating.

Results:

(*a*) C, 0.10 to 0.16 per cent. Average C, 0.13 per cent.
(*b*) C, 1.13 to 0.22 per cent. Average C, 0.17 p Average C, 0.15 per cent. for (*a*) and (*b*).

Around central longitudinal hole and at edge of section a pearlite is somewhat less for slight depth.

*Sample No. 6.*—Carbon 0.03 per cent.

Thermal treatment: Heated to 1,024° C., cooled in furna peated, heated to 1,024° C.

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[14] Howe's paper, in which the early experiments on these steels were describ that manganese has the effect which Mr. Rawdon here infers. *Proceed American Society for Testing Materials*, vol. xi., pp. 365, 367, Proposition 15 (19

Sections examined:

(*a*) Upper end of cylinder used for cooling curve.
(*b*) Other end of cylinder used for cooling curve.
(*c*) Section 5 mm. from end of cylinder.
(*d*) Section from original unheated bar.

Results:

(*a*) End of cylinder nearly free from pearlite.
(*b*) Same as (*a*).
(*c*) Nearly a complete ring 2 mm. deep contained no pearlite. Microphotograph 6 *A* is a view of part of this area.
(*d*) Shows some pearlite; free ring as (*c*). Microphotograph 6 *B*, Plate I., is a view of part of this area.

)therwise the interiors of (*c*) and (*d*) are normal and showed about 3 per cent. of C. (did not use micrometer eye-piece).

*ıple No. 7.*—Carbon 0.92 per cent.

'hermal treatment: Heated to 972° C. for cooling-curve deter-ation, cooled in furnace. Repeated, heated to 972° C.

ections examined:

(*a*) End of cylinder used for cooling-curve determination.
(*b*) Other end of cylinder after deep grinding.

esults:

(*a*) Showed as a compact mass of pearlite, no ferrite.
(*b*) Same as (*a*).

ched with sodium picrate, showed faint traces of free cementite.

*ple No. 8.* Carbon 1.16 per cent.

Thermal treatment: Heated to 974° C., cooled in furnace. Repeated, heated to 994° C. Repeated, heated to 986° C.

Sections examined:

(*a*) Upper end of cylinder used for cooling-curve.
(*b*) Other end of cylinder used for deep grinding.
(*c*) Original unheated bar.

Results:

(*a*) The surface pits very badly on polishing; almost impossible to prevent it. After picric acid etching, the section showed as a mass of pearlite with much free cementite forming the outlines of the grains. After sodium picrate etching the outlines of the cementite show plainly with the many polishing pits. The amount of cementite appears to be slightly less than the carbon content demands. See Microphotograph 8 *A*.

(*b*) Similar in every repect to (*a*).

(*c*) Similar to (*a*) except the free cementite is not all coll the outlines of the grains. See Microphotograp Polishing pits are not produced to any extent.

*Sample No. 9.*—Carbon 1.45 per cent.

Thermal treatment: Heated to 1,056° C., cooled in furnac peated, heated to 1,055° C., cooled in furnace.

Sections examined:

(*a*) Upper end of cylinder used for cooling curve.

(*b*) Other end of cylinder after deep grinding.

(*c*) Section 8 mm. from end of cylinder.

(*d*) Section of unheated bar.

Results:

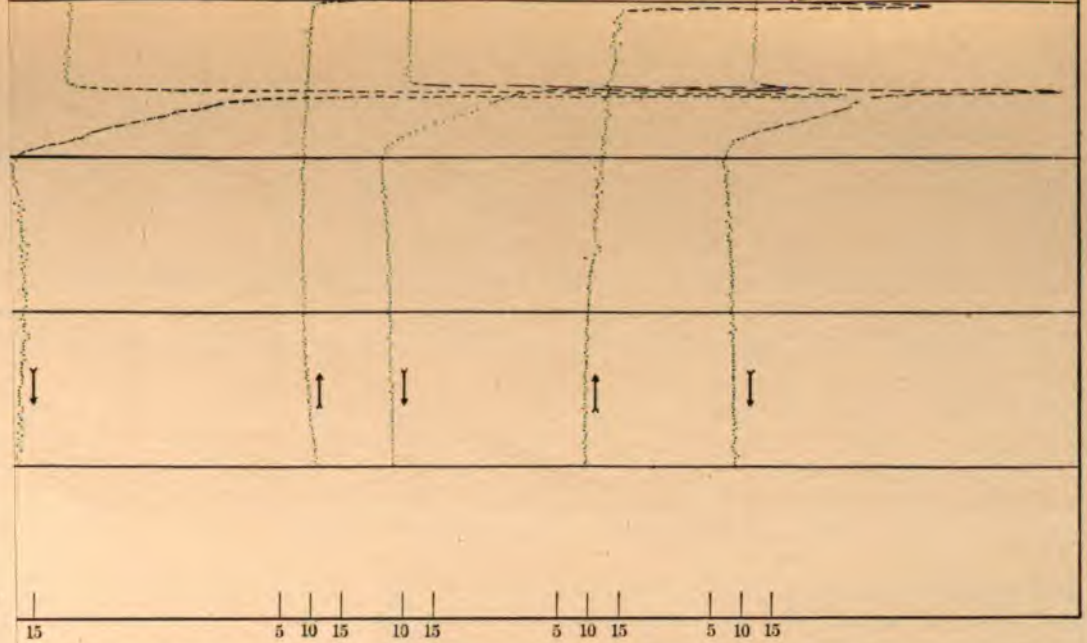
(*a*) The surface pits very badly upon polishing. So cementite is seen after picric acid etching together with mass of pearlite. After sodium picrate the free ce shows more plainly together with the many polishing pit tion is quite varied; some spots show about the correct of cementite occurring as outlines for the grains; in oth the free cementite is entirely wanting; still others show i lated masses; on the whole, the amount appears consi less than should be expected. Microphotograph 9*A* sho diminution of cementite, though it is not typical of th section.

(*b*) Similar to (*a*) except more cementite occurs as out the grains.

(*c*) The section is nearly uniform, though the amour mentite which occurs as outlines for the grains appears less than would be expected.

(*d*) The section, after sodium picrate etching, showed mentite largely distributed throughout the interior of the Polishing pits were almost entirely absent. Micropho 9*B*.

On the whole, the amount of cementite appears somew than the carbon content calls for; this is especially tru end of the cylinder.




AND COOLING CURVES OF BURGESS AND CROWE, ON THE STEELS OF HOWE AND LEVY.

he arrows pointing up indicate heatings; those pointing down, coolings.

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USSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the
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Street, New York, N. Y., for presentation by the Secretary or other representative of its
Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1913. Any
offered thereafter should preferably be in the form of a new paper for publication in Vol.
vith suitable cross references in both volumes).

# ussion of the Existing Data as to the Position of Ae3.*

BY H. M. HOWE,† NEW YORK, N. Y.

(New York Meeting, October, 1913.)

## PART I. INTRODUCTORY.

. INTRODUCTION.—This paper discusses the chief existing data
ne temperature, in iron-carbon alloys, of Ae3, the upper limit
transformation range when in equilibrium, as distinguished
c3, the temperature at which the end of the transformation is
in heating up, and from Ar3, that at which its beginning is
n cooling down. The former is in fact carried above Ae3 and
er below Ae3 by lag. I touch incidentally on the influence of
nese on Ae3 (§ 29, p. 1102).

sections, tables, and illustrations, are numbered consecutively
ose in the accompanying papers by Howe and Levy on Deter-
on of the Position of Ae3 in Carbon-Iron Alloys, and by Bur-
rowe, and Rawdon on Thermal and Microscopical Examination
fessor Howe's Standard Commercial Steels, to which the pres-
er forms a sequel.

. The *contradictoriness of our present evidence* is shown by
ng it together in Fig. 14. For instance, both Goerens and
(Series 2) and Meuthen (Series 12) used practically stationary
ature, and thus reduced the influence of lag very greatly; yet
r 0.32 carbon steel according to the former investigators is
ut according to Meuthen it is at or below 770°, a discrepancy
° or more. In view of our ability to measure these high tem-
es within a few degrees, the existence of such a discrepancy is
elcome as it seems needless. I attack the thankless task of
g some order out of this chaos.

'art II. (§§ 29 to 39) I consider the chief causes of error. In
II. (§§ 40 to 48) I apply certain corrections to some of the
g data, review those data, weight them, and calculate a prob-

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ontribution from the Metallurgical Laboratories of Columbia University. MS.
May 14, 1913.

fessor of Metallurgy, Columbia University.

able position of Ae3, which is shown in Fig. 14 as the broa
line *AAA*. But this calculated line is not entitled to great
because of the weakness of the evidence. Much better c
needed, reached with pure materials and with the many c
error reduced to a minimum. In particular, great chemica
geneousness should be assured by microscopic examination;
decarburization should be prevented; and lag should be redu
minimum by using stationary-temperature methods, such as th
graphic and dilatational, and preferably by using two such
on the same specimens. The thermal methods, useful as they
reconnaissances, should be used only as a check. This is
worthy of a national laboratory, especially in view of the in
importance of Ae3.

§ 26. *The Industrial Importance of Ae3.*—Industrially thi
of the most important lines in the carbon-iron diagram, and
the most important. When medium-carbon and high-carbo
eutectoid steels, such as are fitted for forgings, the better c
shafting, and like engineering purposes, remain undisturbed
important length of time at temperatures anywhere between 7
the upper part of the transformation range, their ferrite coale
harmfully large particles. This is true, though in a less de
the alloy steels which are likely to compete for such purpo
these carbon steels, such as low-nickel steel and low-mangane
To undo this damage and give the metal even approximately
properties of which it is capable, a first step is to cause the r
tion of that ferrite, and the effacement of the local segrega
the heterogeneousness which its coalescence constitutes. T
sorption and this effacement are brought about by heating
and holding there long enough to allow diffusion to become th
and the ferrite to be reabsorbed, or in short by "grain r
This refining or diffusion process should be followed by as
cooling through the transformation range as is practicable, b
slow cooling through that range would enable the ferrite
coalesce into masses which are harmfully large, though o
much smaller and less harmful than those which form wh
cooling is from a much higher temperature still.[15]

[15] When forging is done properly, and in particular is continued till the t
has sunk at least to the lower part of the transformation range, the degree of
of the ferrite in the remaining cooling is relatively slight. Even that slight
may be removed by the process of grain refining. But if the cooling down from
refining is slow, the whole process is likely to do more harm than good, becs
ferrite coalescence would arise in the slow cooling of a large forging from Ae3
to remain at the end of the forging operation. Unless the cooling can be made

grain refining of the core of case-hardened objects requires a heating to Ae3.

se engineering objects are now so important, and are likely to e so much more important, that trustworthy knowledge of the ature of this therapeutic line Ae3 is much needed.

ervers have not hitherto sought Ae3, but have been content to ine points easier to find, such as the beginning of Ar3, which ie an indeterminate distance below Ae3, or even the maximum , which lies an indeterminate distance lower yet.

. *Industrial A3 vs. A3 of Cooling-Curve Method.*—Osmond d what we should naturally expect, that the lag increases with pidity of cooling. Thus it occurs that, in the determination of the usual method of taking cooling curves and heating curves ch cooling or heating is relatively rapid, there is greater lag the heating of steel for industrial heat-treatment, for here the ature usually rises slowly, especially as it approaches the peak ature aimed at, and in general it remains stationary at that emperature. Hence in these industrial heatings the actual Ac3, nperature at which the reabsorption of the pro-eutectoid ferrite ally complete, lies nearer to the equilibrium temperature of A3, han the A3 as determined by cooling and heating curves does. , it is probable that in the industrial heating of large objects is negligible, the reabsorption of ferrite practically completing t each spot as the temperature of that spot reaches Ae3, though rse the heating of the interior may lag many degrees behind the outside. Hence at first sight we infer that the tempera- which heating should be directed should be exactly Ae3, above ginning of Ar3 as found by cooling curves and below the be- g of Ac3 as found by heating curves.

. *Should Industrial Grain Refinings Aim at Ae3 or at Some Other ature?*—This inference may have to be modified because of a al consideration. Are the best mechanical properties given ctly completing the reabsorption of the ferrite, or by leaving eabsorption slightly incomplete? We must remember that in industrial heatings we aim at a specific degree of tempera- hat we really hit is a range, in about the middle of which that lies. We do not bring even the immediate vicinity of our -junction, calorimetric ball, or whatever, to the specific degree

to 700°, grain refining should be replaced by a moderate reheating sufficient to he forging stresses and to give the needed ductility, usually to between 500° and n the case of large objects the cooling from this should of course be slow, lest new arise.

at which we aim, but into a range including that degree, a more or less narrow according to our skill and care. But th perature of that vicinity is not the temperature reached by th parts of the objects we are treating. These other parts may be or they may be colder than that vicinity; so that we bring thes parts not even into the relatively narrow range to which we that vicinity, but only into a still wider range of temperature, which range lies the specific degree of temperature at which Not only parts but important parts of our objects may rise wel that specific degree, or may not rise to within a considerabl of it.

Therefore we ask whether it is better to err on the high or low side; and how the benefit from completing the last of th sorption of the ferrite in the cooler parts of our object compar the harm done by simultaneously carrying the hotter parts object appreciably above A3, and thus inducing a certain coa of the austenite grains, and of the resultant ferrite network for re-cooling.

Indeed, there is evidence to suggest that even before the te ture reaches A3, *i. e.*, even before the last of the ferrite is reab there is a material coarsening of the grain of the austenite w mixed with that ferrite, a coarsening which is likely to be har

This question whether it is exactly A3 or some temperature higher or slightly lower that we should aim at industrially n answered by direct experiments, of which some are in hand.

## Part II. The Chief Causes of Error.

§ 29. *The Influence of Manganese.*—I have pointed out[16] tha ganese has a general sluggardizing or retarding influence chemical and structural changes in steel, retarding (1) the for and (2) the break up of the ferrite network; (3) the coagulation rite into recognizable masses; (4) the growth of the ferrite thus lessening the size which they reach; (5) the growth lamellæ of pearlite, thus lessening their size, probably both and through lowering the temperature of Ar1 at which they ar (6) and the growth of the austenite grains, thus lessening t which they reach: and (7) lessening the coarseness of the ferr work based on that grain size. Further, (8) that we may Osmond's diagram, Fig. 8, a like retarding effect on the part of for though the lowering of the Ac points indeed indicates that

[16] *Proceedings of the American Society for Testing Materials*, vol. xi., p. 365 (19

s the Ae points, Ae1, Ae2, and Ae3, yet the continuous widen-
f the gap between the Ac and the Ar points as the nickel con-
ncreases explains itself as a result of this retarding action, for
an action should have just this effect, as regards the Ar points
rcing the effect of the lowering of the Ae points, but as regards
c points opposing that effect.

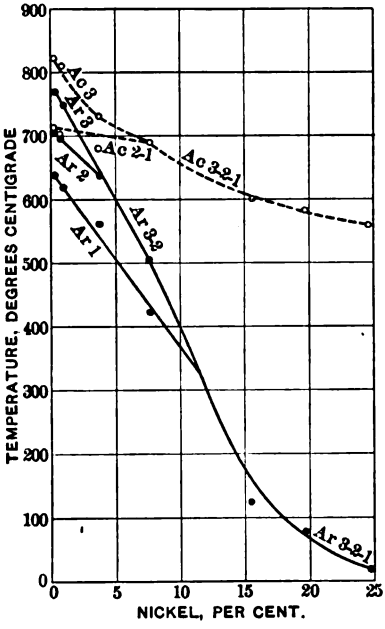


FIG. 8.—OSMOND'S IRON-NICKEL DIAGRAM.
(After Institution of Civil Engineers, Proceedings, Vol. 138, 1898–99).

ther evidence that nickel has such a retarding influence is given
causing some structural effects like those which I have just
ed to the retarding action of manganese; for instance, retarding
owth of the austenite grains, and hence of the ferrite network
on those grains, so that nickel steel coarsens and embrittles on
heating less readily than carbon steel of like carbon content. To
e sluggardizing effect we may refer the fact that with 25 per
of nickel, or with 11 per cent. of manganese plus 1 per cent. of
n, the transformations are so far retarded that the austenite
persists even after common slow cooling, whereas with an inter-
te quantity of either element the transformation in a common
cooling goes only as far as the state of martensite.

TABLE III.—*Influence of Manganese on the Temperature of the A1 and A3, of the Hypo-Eutectoid Transformation Rang*

| No. | Authority. | Composition C. Per Cent. | Composition Mn. Per Cent. | Ar1. Beginning. Observed. | Ar1. Beginning. Actual Lowering. | Ar1. Beginning. Lowering Calculated for 1⅛ Mn. | Ar1. Maximum. Observed. | Ar1. Maximum. Actual Lowering. | Ar1. Maximum. Lowering Calculated for 1⅛ Mn. | Ac1. Beginning. Observed. | Ac1. Beginning. Actual Lowering. | Ac1. Beginning. Lowering Calculated for 1⅛ Mn. | Ac1. M Observed. |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | ..... | 0.57 | 0.23 | ..... | ..... | ..... | **661** | ..... | ..... | ..... | ..... | ..... | ..... |
| 2 | Osmond | 0.29 | 0.27 | ..... | ..... | ..... | **660** | ..... | ..... | ..... | ..... | ..... | ..... |
| 3 | (uncor- | 0.32 | 0.50 | ..... | ..... | ..... | 640 | 20 | 80 | ..... | ..... | ..... | ..... |
| 4 | rected)..... | 0.42 | 1.00 | ..... | ..... | ..... | 625 | 35 | 47 | ..... | ..... | ..... | ..... |
| 5 | ..... | 0.46 | 1.08 | ..... | ..... | ..... | 620 | 40 | 45 | ..... | ..... | ..... | ..... |
| 6 | ..... | 0.36 | 0.42 | **680** | ..... | ..... | **675** | ..... | ..... | **720** | ..... | ..... | **735** |
| 7 | ..... | 0.48 | 0.45 | **670** | ..... | ..... | **665** | ..... | ..... | **715** | ..... | ..... | **735** |
| 8 | ..... | 0.36 | 0.52 | 660 | 15 | 170 | 655 | 15 | 170 | 720 | ..... | ..... | 735 |
| 9 | Saladin..... | 0.22 | 0.53 | 655 | 20 | 200 | 645 | 25 | 250 | 710 | ..... | ..... | 735 |
| 10 | | 0.46 | 0.63 | 680 | ..... | ..... | 665 | 5 | 25 | 702 | ..... | ..... | 720 |
| 11 | ..... | 0.66 | 0.71 | 680 | ..... | ..... | 665 | 5 | 18 | 720 | ..... | ..... | 735 |
| 12 | ..... | 0.31 | 1.66 | 630 | 45 | 37 | 610 | 60 | 45 | 730 | ..... | ..... | 745 |
| 13 | ..... | 0.90 | 4.16 | 620 | 55 | 15 | 585 | 85 | 25 | 730 | ..... | ..... | 740 |
| 14 | Carpenter, | 0.47 | trace | **770** | ..... | ..... | **690** | ..... | ..... | ..... | ..... | ..... | ..... |
| 15 | Hadfield & Longmuir. | 0.47 | 0.95 | 706 | 64 | 67 | 665 | 25 | 26 | ..... | ..... | ..... | ..... |
| 16 | ..... | 0.38 | 0.51 | **675** | ..... | ..... | **683** | ..... | ..... | **712** | ..... | ..... | **725** |
| 17 | ..... | 0.44 | 1.23 | 655 | 20 | 28 | 666 | 17 | 24 | 730 | ..... | ..... | 745 |
| 18 | Sargent..... | 0.46 | 1.23 | 660 | 15 | 21 | 666 | 17 | 24 | 735 | ..... | ..... | 742 |
| 19 | | 0.56 | 1.28 | 652 | 23 | 30 | 655 | 28 | 36 | 725 | ..... | ..... | 740 |
| 20 | ..... | 0.58 | 1.28 | 652 | 23 | 30 | 657 | 26 | 34 | 720 | ..... | ..... | 735 |
| 21 | ..... | 0.62 | 1.66 | 642 | 33 | 29 | 655 | 28 | 24 | 720 | ..... | ..... | 737 |
| 22 | ..... | 0.21 | 0.05 | **682** | ..... | ..... | **678** | ..... | ..... | **754** | ..... | ..... | **762** |
| 23 | Burgess | 0.14 | 1.16 | 630 | 52 | 47 | 620 | 58 | 52 | 738 | 16 | 14 | 745 |
| 24 | and Crowe | 0.40 | 0.16 | **695** | ..... | ..... | **688** | ..... | ..... | **754** | ..... | ..... | **757** |
| 25 | ..... | 0.46 | 1.21 | 648 | 47 | 45 | 646 | 42 | 40 | 730 | 24 | 23 | 736 |
| 26 | ..... | 0.21 | 0.05 | ..... | ..... | ..... | **695** | ..... | ..... | ..... | ..... | ..... | **730** |
| 27 | Howe and | 0.14 | 1.16 | ..... | ..... | ..... | 645 | 50 | 45 | ..... | ..... | ..... | 730 |
| 28 | Levy..... | 0.40 | 0.16 | ..... | ..... | ..... | **705** | ..... | ..... | ..... | ..... | ..... | **735** |
| 29 | ..... | 0.46 | 1.21 | ..... | ..... | ..... | 664 | 41 | 40 | ..... | ..... | ..... | 728 |
| 30 | ..... | 0.86 | 0.05 | ..... | ..... | ..... | ..... | ..... | ..... | **730** | ..... | ..... | ..... |
| 31 | Stead..... | 0.88 | 0.56 | ..... | ..... | ..... | ..... | ..... | ..... | 728 | 2 | 4 | ..... |
| 32 | ..... | 0.89 | 0.94 | ..... | ..... | ..... | ..... | ..... | ..... | 720 | 10 | 11 | ..... |
| 33 | ..... | 1.00 | 2.04 | ..... | ..... | ..... | ..... | ..... | ..... | 715 | 15 | 7 | ..... |

| | | | | Ar3. Beginning. | | | Ar3. Maximum. | | | Ac3. End. | | | Ac3. M |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 34 | ..... | 0.57 | 0.23 | **750** | ..... | ..... | **695** | ..... | ..... | ..... | ..... | ..... | ..... |
| 35 | Osmond | 0.29 | 0.27 | **780** | ..... | ..... | **717** | ..... | ..... | ..... | ..... | ..... | ..... |
| 36 | (uncor- | 0.32 | 0.50 | 740 | 37 | 148 | 700 | 15 | 60 | ..... | ..... | ..... | ..... |
| 37 | rected)..... | 0.42 | 1.00 | 725 | 41 | 55 | 665? | 42 | 56 | ..... | ..... | ..... | ..... |
| 38 | ..... | 0.46 | 1.08 | 725? | 37 | 45 | 658? | 15 | 54 | ..... | ..... | ..... | ..... |
| 39 | ..... | 0.03 | 0.26 | **889** | ..... | ..... | **870** | ..... | ..... | **916** | ..... | ..... | **908** |
| 40 | Burgess | 0.21 | 0.05 | **820** | ..... | ..... | **796** | ..... | ..... | **865** | ..... | ..... | **852** |
| 41 | and Crowe | 0.40 | 0.16 | **754** | ..... | ..... | **712** | ..... | ..... | **828** | ..... | ..... | ..... |
| 42 | | 0.14 | 1.16 | 786 | 62 | 62 | 763 | 60 | 60 | 854 | 30 | 30 | 836 |
| 43 | ..... | 0.46 | 1.21 | 724? | 10 | 9 | ..... | ..... | ..... | 784 | 30 | 29 | ..... |

NOTES TO TABLE III.—**1** to **5** and **34** to **38**. Osmond, *Transformation d carbone*, pp. 41, 48, and 93 (Paris, 1888). Rolled or forged rods, apparently in diameter or width, and 60 cm. long, were heated in a porcelain tube in a g with incomplete exclusion of air. The rate of cooling was between 1 and 2 sec. p

**6** to **13**. Saladin, *The Iron and Steel Magazine*, vol. vii., p. 237 (1904). For 9, and 11, the curves were "obtained in 10 minutes" or at the rate of about ¼ o degree. For Nos. 8 and 10 the cooling from 1,050° to 450° lasted one hour, or of 6 sec. per degree. The heating from 0° to 1,050° lasted one hour, or at about 3.5 sec. per degree. For Nos. 12 and 13 the rate of cooling is not gi specimens appear to have been small cylinders.

**14** to **15**. Carpenter, Hadfield, and Longmuir. Seventh Report Alloy Committee, *Proceedings of the Institution of Mechanical Engineers*, 1905, pp. 9 Cylinders ⅝ in. long and ⅝ in. in diameter, machined from cast material, were h

l tube furnace in air. Cooling from 900° to 50° required about 2.25 hours, or at of 9.5 sec. per degree.

21. G. W. Sargent, Metallurgist of the Crucible Steel Company of America, ommunication, May 1, 1911. Cylinders 0.75 in. long and 0.75 in. in diameter d. The rates of heating and cooling are not given.

25 and 39 to 43. Burgess and Crowe. See p. 1093, of this *Bulletin*. Rolled 0.5 to 1 in. long and ⅝ in. in diameter were used. The rate of heating and cool- bout 5 to 7 sec. per degree.

29. Howe and Levy. Rolled cylinders about 1 in. long and ⅝ in. in diameter d. The rates of heating and cooling varied from 3 to 12 sec. per degree. Their s and those of Burgess and Crowe were cut from the same bars.

33. Stead, *Proceedings of the North-East Coast Institution of Engineers and Ship-* vol. xxix., 1912–1913, p. 19 of reprint.

ng thus found that nickel both lowers Ae1 and Ae3 and also the transformations and some at least of the structural changes; anganese certainly retards these structural changes and also thermal Ar1 as shown in 1888 by Osmond; and that manganese ckel behave alike in a general way, we ask whether this resem- does not go farther, the lowering effect of manganese applying y to Ar1 but also to both Ae1 and Ae3, and its retarding effect to a widening of the gap between the Ac and the Ar points at caused by the retarding effect of nickel. Let us consider dence.

THERMAL A1.—Ar1. The evidence collected in Table III. bears mond's early discovery that manganese lowers Ar1, but shows does not always lower Ac1. To show this I have set in broad ype in that table the data for steels relatively poor in manga- hich may be used as a standard with which to compare the those beneath them richer in manganese, the average of *1* as a standard to try 3 to 5 inclusive, the average of *6* and *7* 13, *14* for 15, etc. This shows a lowering of both beginning ximum of Ar1 in every case except lines 10 and 11, exceptions may indeed be due to the difficulty of interpreting accurately ns on so small a scale as that of these most interesting ones of 's.

But in the majority of cases there is either no lowering at all beginning or the maximum of Ac1, or at most a lowering s within the wide limits of experimental error. Indeed on an both the beginning and maximum of Ac1 appear to be some- aised by manganese. These majority data have such advan- there is in being autographic; but on the other hand they are experimenters only, and against them must be weighed the ensitive determinations of Burgess and Crowe, which show a te lowering of both beginning and maximum of Ac1, lines 22 and the most ingenious determinations of Stead, which show a

constant though slight lowering of Ac1–2–3, in the extreme
the rate of only 7.5° per 1 per cent. of manganese.

The widening of the gap caused by this excess of the lo
effect of manganese on Ar1 over its effect on Ac1 is shown
cally in Fig. 9.

The inference is strong that manganese increases lag, fr
contrast between the almost invariable marked lowering of A
the rarer lowering of Ac1.

(2) THERMAL A3.—In the 0.14 and 0.46 carbon steels of
and Crowe, Tables III. and IV., about 1 per cent. of mar
lowers Ac3 about 30°, and Ar3 about 60° in one case, but
other, by a smaller quantity, which cannot be estimated conf

In Stead's eutectoid steel manganese lowers Ac1–2–3 slig
the rate of 7.5° per 1 per cent. of manganese in the extreme
already indicated under Ac1.

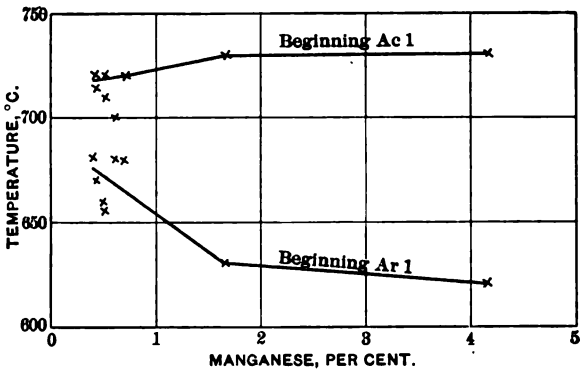


FIG. 9.—MANGANESE WIDENS THE GAP BETWEEN Ac1 AND Ar1, CALCULATED FROM SALADIN'S DATA.

(3) MICROGRAPHIC A3.—Though the manganese in the fou
manganese steels of Howe and Levy runs up to 1.21 per ce
the lowering of their micrographic Ac3 in no case exceeds 13
deed in two cases, carbon 0.14 and 0.46 per cent., it is about
than the lowering of the Burgess and Crowe thermal Ac3 i
same specimens. This paradoxical result may indeed be due
in detecting the end of the reabsorption of the ferrite, or it ma
that, in these sluggish high-manganese steels, a small microsc
recognizable segregated residue of ferrite may persist for so
after the retardation in the heating curve caused by the ferri
sorption has become so slight as to be undetectable thermally.

These results are recapitulated in Table IV.

V.—*Lowering Effect of Manganese on the Transformation Points Ac3 and Ar3.*

| | Mn. Per Cent. | Micrographic (Howe and Levy). Ac3. Lowered by | Thermal (Burgess and Crowe). Ac3 (End) Lowered by | Ar3 Beginning. Lowered by | Gap Between Thermal Ac3 and Ar3. |
|---|---|---|---|---|---|
| 3 | 0.26 | ..... | Nil | Nil | ..... |
| 1 | 0.60 | Nil | ..... | ..... | ..... |
| 4 | 1.16 | < 10° | 30° | 63° | Widened by 33° (about 47 per cent). |
| 2 | 0.41 | < 10° | ..... | ..... | ..... |
| 3 | 1.21 | < 13° | 30°? | Very little? | Narrowed by an indeterminate amount. |

this up, as regards both A1 and A3 manganese lowers the point very materially.

points are not clearly lowered by manganese in the results Sargent, and Howe and Levy, but they are in Stead's in the sensitive but scanty determinations of Burgess and

hat manganese probably lowers both Ae1 and Ae3 slightly. place this is antecedently probable from the general close between the effects of manganese and those of nickel, and fact that nickel evidently lowers the Ae points. In the ce, the effects of manganese are explained far more readily othesis that it lowers Ae1 and Ae3 slightly than by the That it increases lag may be admitted. But unless it low- Ae points, its increasing lag would lead to its raising the invariably. In the relatively rapid heatings of Saladin t the great increase of lag which manganese causes should marked raising effect on Ac1, but in fact the raising effect much as 1.66 and 4.16 per cent. of manganese is only a very s (lines 12 and 13). The invariable slightness of the effect ese on the Ac points, taken with the varying sign of such here is, indicates a double effect of manganese on these that it increases lag it tends to raise these Ac points, and owers the Ae points it tends to lower the Ac points as well. former effect is the greater of the two the net effect is to c point; when it is the less of the two the net effect is to Ac point. The amount of its lowering of the Ae points constant for each specimen, but the amount of its raising gh lag should vary with the conditions of heating.

*nganese Lessen the Eutectoid Carbon Content?* The general

belief that it does may be due to its retarding the coalescence pro-eutectoid ferrite, with the result that, in steels which a slightly hypo-eutectoid, the pro-eutectoid ferrite present may detection, and thus give the impression that the steel is eutect

If manganese did lessen the eutectoid carbon content, it wou ably thereby steepen the line A3, so that, for given high ma content, the narrowing of the gap between A3 and A1 which ing carbon content causes should be more rapid than in low-ma steel, as sketched in Fig. 10. Here the broken lines show tively how the expected effect of manganese would affect th formation range, if in addition to lowering the observed po the thermal Ar points from their position *GOS*, *MO*, and *P* lessened the eutectoid carbon content. Here the distance (

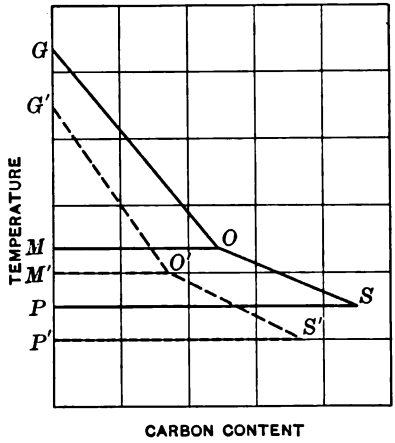


FIG. 10.—TO LESSEN THE EUTECTOID CARBON CONTENT STEEPENS A3, *GO* and *OS*.

*MM'* and *PP'* would represent the lowering of the transf points of carbonless iron by manganese, but the steepening *G'O'* and of *OS* to *O'S'* would represent the lessening of t toid carbon content.

I have long been on the watch for data throwing light on ening. The only ones I have found, those of Burgess an collected in Table III:, do not indicate such a lessening, tho are too scanty to carry much weight. Thus, though the A are lowered by manganese in the 0.14 carbon steel about as in the 0.46 carbon steel, the A3 points are, if anything, low in the higher than in the lower carbon steel, pointing rath increase than to a decrease of the eutectoid carbon content.

*The Influence of Phosphorus.*—§ 21 shows that, at least in ed castings of about 0.34 per cent. of carbon, phosphorus raises perature of Ae3 as found micrographically. How this effect phorus is related to the carbon content will be discussed in § 43. r consultation with Dr. Stead, I offer the following interpreta- the structural peculiarities of these high phosphorus specimens in Plate 6, Rows 15, 16, and 17, and described in § 21.

persistence of the dendritic markings represents the extreme ss with which phosphorus diffuses.

dark median layers represent phosphoric concentrations, per- e phosphorus eutectic in irresoluble masses, persisting because same slowness of diffusion.

the position of the phosphoric matter within the ferrite two suggest themselves. First, in the slow cooling down through stenite stage, from the solidus towards the transformation the phosphoric matter differs enough from the remainder of tenite to be treated like a foreign body. The growing grains enite eject it to their boundaries as the eye ejects a grain of r as every organism strives to eject foreign matter, by surface and whatever other means it has. In the passage down h the transformation range that same austenite in turn ejects rite to which it now gives birth, and it must needs pile this ection on the earlier, which, like slag particles, no more at- errite than the underlying archaic rocks attract the palæozoic. mply bear to them their inevitable stratigraphic relation.[17]

second reason is seen best in the case of the dendrites in Fig. 15, Plate 6. The phosphoric concentrate looks as if it were xis of the solidification dendrites, imbedded in the later depos- rite, and this in turn surrounded by the dark pearlite, the last o solidify. But we readily see the fallacy in this interpreta- Clearly the phosphoric concentrate must have been the last to , so that its position is not in the axis of the dendrites but in ng between them. The true dendrites, *A* in Fig. 11, are the which is now dark and pearlitic; the light ferrite forms the *B*, between these dendrites; and the phosphoric concentrate , *c*, of these fillings. This is the reverse of the initial relation. y the low-carbon parts formed the dendrite axes, *A*, and the rbon parts the filling, *B*, between.

reversal is clearly brought about in cooling through the trans-

the Author, *Proceedings of the American Society for Testing Materials*, vol. xi., p. ), and discussion of J. E. Stead's paper, *Proceedings of the International Association g Materials*, VI. Congress, III., to appear.

formation range by a very simple mechanism. Because pho raises A3, and because the phosphorus content shades off fro wards *A*, in cooling the beginning of the precipitation of fer occur much earlier at *ccc* than at *A*, and will occur later and we pass from *ccc* towards *A*. We have but to assume that t earliest precipitate of all will adhere to the phosphoric con *ccc* alongside which it is generated, for this would lead to a p sive withdrawing of the austenite towards *A* as it generates fe *ccc*. As each successive concentric layer farther from *ccc* generating ferrite, that generation is continuing in the la which earlier started this generation; and through this gener ferrite which coheres with the earlier formed ferrite against w is born, the residual austenite is gradually elbowed away fr Thus the carbon movement which occurred in solidificatio

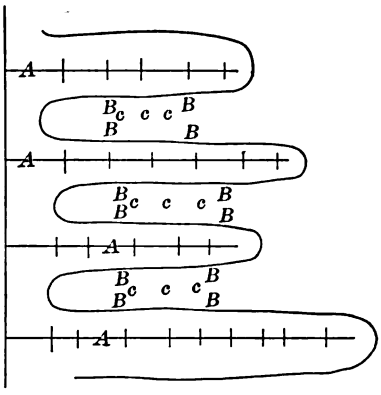


FIG. 11.—TRANSFORMATION THROWS THE CARBON INTO THE AXIAL REGION OF THE DENDRITES.

versed, and the carbon which the differentiation of solidificati concentrated in the fillings, *B*, is by the differentiation of tran tion concentrated about the lines *AAAA*, which had been th drite axes at the time of solidification.

In a word, the ferrite surrounds the phosphoric concentrates because as a later deposit it must needs lie on the earlier, and because the enrichment in phosphorus of the surrounding au leads to both an earlier and a more abundant precipitation of along the shore of those concentrates, and thus to displacen the residual mother austenite away from them.

Having admitted this for the dendritic structure of Fig. 15, Plate 6, we see that it suffices to account for the median p of the dark phosphoric masses in the network of Figs. *E* an Row 16, and for the encircling of the dark phosphoric dot ferrite in Figs. *C* to *F* of Row 15.

| | Cause of Error. | Hypo-eutectoid Steel, A3. | | | | | Hyper-eutectoid Steel Amc. | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | Method. | | | | | Method. | | | |
| | | Dilatational. | Thermal. | | Micrographic. | | Thermal. | | Micrographic. | |
| 1 | 2 | 3 Cooling down. | 4 Cooling Curves. | 5 Heating Curves. | 6 Precipitation. | 7 Reabsorption. | 8 Cooling Curves. | 9 Heating Curves. | 10 Precipitation. | 11 Reabsorption. |
| | | Influence of Cause of Error Tends to Make Observed Temperature. | | | | | | | | |
| 1 | Lag | Slightly too low; avoidable. | Too low. | Too high. | Too low; avoidable. | Slightly too high; avoidable. | Too low. | Too high. | Too low; avoidable. | Slightly too high; avoidable. |
| 2 | Surface decarburization | ..... | Too high; avoidable. | Too high; avoidable. | Too high; avoidable. | Too high; avoidable. | Too high; avoidable. | Too low; avoidable. | Too low; avoidable. | Too low; avoidable. |
| 3 | Persistence of solidificational segregation (of phosphorus and carbon) | ..... | Too high. | Too high. | Too high. | Too high. | | | | |
| 4 | Persistence of transformational segregation | ..... | Too high. | Too high. | Too high. | Too high. | Too high. | Too high. | Too high. | Too high. |
| 5 | Precipitation of "quenching" ferrite or cementite | ..... | ..... | ..... | Too high; avoidable. | Too high; avoidable. | ..... | ..... | Too low. | Too low; avoidable. |
| 6 | Martensite mistaken for ferrite | ..... | ..... | ..... | Too high; avoidable. | Too high; avoidable. | | | | |
| 7 | Failure to detect first (or last) traces of pro-eutectoid element | ..... | ..... | ..... | Too low (slight). | Too low (slight). | ..... | ..... | Too low. | Too low. |
| 8 | Failure to identify first (or last) traces of retardation. | | Indeterminate. | | | | Indeterminate. | | | |

§ 31. *Other Causes of Error.*—These are set forth in
Let us pass them in review.

*Lag, Undercooling, or Hysteresis.*—The failure of pure water
till after the temperature has fallen materially below 0°,
recalescence in steel to set in till the temperature has fal
degrees, perhaps even 100°, below the temperature at wh
due, Ae1, explains itself as a sort of molecular inertia, or r
to break away from the initial grouping and adopt that wh
becomes due. This may be called lag proper.

It is asserted that the melting of a pure substance is not
lag of this kind; and this may be true of the transformatio
occur in a pure solid substance in heating up. That is to
for the freezing point, and for the transformation points wh
below the freezing point in the solid metal, lag proper
only in cooling and never in heating up. This phil
portant question need not concern us here. Suff
that if the solid is composite its actual melting
temperature at which a given transformation
tectable velocity in heating up, may be raised ma
temperature at which melting or transformation is
reflection shows. Mix spherical quartz pebbles with sp
corundum and others of hard burnt lime, in the fusible pro
6 $SiO_2$, $Al_2O_3$, 3CaO, the melting point of which is reported
Set this mixture in a furnace which is at 1,600°. When th
reach 1,300° no fusion will occur, because nowhere can an
in that fusible proportion actually exist. Silica is in con
lime at certain points and with alumina at others, but at n
it in contact with both. No fusion will occur till the te
reaches the melting point of the most fusible silicate of l
alumina, the most fusible mixture of substances actually in
because though the silicate of lime and alumina is more fu
any of its components that silicate cannot form except wh
components are in actual contact. Spheres of these three
not touching each other at all could be held forever at 1,30
melting, if we ignore sublimation.

Heat up pearlite, and its constituent lamellæ of ferrite a
tite may indeed begin to unite to form austenite at their a
faces of contact at the instant of reaching Ae1; but the p
that ferrite and that cementite which lie back of that conta
unite till, by the dissolving action of the resultant contact
they are brought together. As the temperature rises abov
eutectoid austenite formed at the initial junction is able t

cementite which adjoins it on one side and some of the
adjoins it on the other; and these two diffusing out
tenite enable its alongshore layers to dissolve more ferrite
te, and thus the whole of the pearlite at last transforms
te. But as long as the temperature is only strictly Ae1,
ontact mass of austenite can dissolve neither ferrite nor
om the masses between which it forms the contact; only
ove Ae1 can that dissolving begin, and only as it rises
bove Ae1 can that dissolving be rapid.
ity of ferrite and cementite which can unite at exactly
ently extremely small; indeed we could maintain that
xtend only the width of one molecule from each side of
line. The heat evolved by this is of course too small to
the temperature rises, and as through this process of dif-
ontact austenite dissolves ferrite and cementite from either
ultant heat absorption accelerates till it becomes detect-
But it seems clear that, other things being equal, the
emperature is rising the farther will it have risen at the
his dissolving process has reached such rapidity as to
ectable heat absorption, *i. e.*, the higher will be the ob-
ning of the transformation.
of diffusional lag evidently applies to the melting of com-
s including eutectics, and to the transformations in such
olid masses including eutectoids in heating up. Its cir-
are different from those of the lag in the freezing of a
s liquid, or the transformation of a nominally chemically
s substance like austenite in cooling. But it seems to be
e same general influences, and in particular to increase
idity of heating as lag proper increases with the rapidity
As we shall see in the next section, the grosser form of
usness caused by solidificational segregation raises the
and the beginning of Ar3, and presumably to a degree
ases with the rapidity with which the temperature is
ling.
the heating lag should be much less than the cooling lag.
re are no nuclei of ferrite present to induce the precipita-
te on cooling to the true solubility line Ae3, that precipi-
bly can never begin till the supersolubility or lability line
eached, so that a certain degree of lag in cooling is prob-
ble. But it is not clear that any measurable lag need
eating up, reabsorption completing itself at Ae3, if the
low enough.

How wide this metastable range is between the solubility cu
Ae3 and the supersolubility curve above which precipitation ca
occur on cooling, our present data are too contradictory to sh
The wide gap between the end of Ac3 and the beginning of Ar
the data of Table IX., after p. 1136, indeed point to a pretty w
metastable range. On the other hand the fact that the very ca
Ar3 data of Heyn, Series 2 of Fig. 14, are either above or but slig
below the Ac3 data of Burgess and Crowe, Series 4, goes to s
that this range is narrow, and so does the fact that in the statio
temperature micrographic determinations of Goerens and Meyer,
in those of Howe and Levy, Ac3 and Ar3 occurred within the s
rather narrow temperature gap. It is to be remembered that, th
because of lag, the gap between the observed Ac3 and Ar3
exceed the width of this range by an indeterminate amount, y
can never be narrower than this range.

Because lag increases with the rate of heating or cooling,
thermal method is at a disadvantage compared with the statio
temperature methods, such as the micrographic, and such as the
tational may be made, because in these methods the holding at
tionary temperature may be made long enough to lessen the ef
of lag very greatly, possibly to a quantity which is negligible fo
present purposes. It may be possible to carry out the thermal me
with coolings and heatings so extremely slow as to reduce lag gre
indeed to a negligible quantity, but in the experiments here in
tion this has not been done. The difficulties, especially in the
of a very slow yet regular rise of temperature, are serious.

The actual lowering of the beginning and maximum of Ar
hastening the cooling as found by Osmond is as follows:

TABLE VI.—*Influence of Rate of Cooling on Ar1.*

| Cooled. | Semi-Hard Steel, 0.57 Per Cent. C. | | | Hard Steel, 1.25 Per Cent. C | | |
|---|---|---|---|---|---|---|
| | Time of Cooling Between 685°–658°. | Recalescence Ar 1. Beg. ° C. | Max. | Time of Cooling Between 705°–678°. | Recale Ar 3 Beg. ° | |
| 1. Gently in tube | 116 seconds | 653 | 656 | 49 seconds | 671 | |
| 2. As usual in tube | 24.5 seconds | 648 | 655 | 20 seconds | 670 | |
| 3. Rapidly in air | Not measured | 637 | 640 | Not measured | 642 | |
| 4. Very rapidly, quenched in water | Not measured | Absent | | Not measured | Ab | |
| Lowering from lines 1 to 3 | ..... | 16° | 16° | ..... | 19° | |

Incidental variations in the rate of heating or cooling of the
used in this investigation caused a change of about this same or

n the temperature of Ar1 maximum, but no recognizable
ne temperature of Ac1 maximum, as is shown in Table

II.—*Influence of Rate of Heating or Cooling on Thermal Ac1 and Ar1 (Howe and Levy).*

| rbon. Cent. | Manganese. Per Cent. | Rate of Heating. Seconds per 1° C. | Ac1. Maximum °C. | Rate of Cooling. Seconds per 1° C. | Ar1. Maximum °C. |
|---|---|---|---|---|---|
| 0.40 | 0.16 | 4 | 736 | ..... | ..... |
| ..... | ..... | 8 | 735 | ..... | ..... |
| ..... | ..... | 12 | 731–736 | ..... | ..... |
| 0.46 | 1.21 | 4 | 729 | 6 | 661–665 |
| ..... | ..... | 8 | 730 | 6½ | 661–666 |
| ..... | ..... | 14 | 731 | 16 | 666–671 |
| 0.92 | 0.12 | 5 | 739 | ..... | ..... |
| ..... | ..... | 8 | 734 | ..... | ..... |
| ..... | ..... | 11 | 738 | ..... | ..... |
| 1.14 | 0.24 | 3 | 738 | ..... | ..... |
| ..... | ..... | 3½ | 737 | ..... | ..... |
| ..... | ..... | 13 | 739 | ..... | ..... |

Burgess's more sensitive observations lowered the end of
n retarding the rate of heating from 3.9 to 4.3 seconds
and by 20° on retarding it to 9 seconds per degree, as is
ble VIII., p. 1133.

lag of A3 may be very considerable is shown by the ob-
Burgess and Crowe as pointed out in § 10. With their
asurements there was a gap of from 38° to 74° between
c3 and the beginning of Ar3 for given specimens.

o line 1 of Table V., both for the hypo-eutectoid and for
oid steel, lag is likely to lower the observed position of
l in cooling curves, and to raise Ac3 as observed in heat-
nd such effect as it has in stationary temperature methods
this same kind.

*rors from Local Lowering of the Carbon-Content*, (2), (3),
general surface decarburization, and uneffaced segrega-
r solidificational or transformational, tend to raise the
nperature of A3 of hypo-eutectoid steels above Ae3.
this cause is especially to be dreaded in the thermal
it may be of importance in the other methods, such as
nal and the electrical resistance and even in the micro-
nod. A glance at the conditions of the thermal method
lear.

erature of A3 rises rapidly as the carbon content falls.
ere is superficial decarburization, the A3 of the outside

of the specimen lies correspondingly higher than the A3 v
sought, the A3 corresponding to the chemical compositio
specimen undecarburized. In taking a cooling curve wi
superficially decarburized specimen, if for simplicity we igno
break in the curve will begin as soon as the cooling reaches
of the decarburized outer layer, and before it reaches the A
sponding to the undecarburized steel. We are likely to
higher first break as the Ar3 of the specimen, because it is th
ning of the break that we look for. The more sensitive the
the higher will it detect such a slight break.

What is thus true when a local impoverishment in carbon
by surface decarburization is true when it is caused by seg
whether solidificational or transformational. We usually
segregation as the enrichment of one part, *e. g.*, in carbon
enrichment of one part must needs cause the impoverishme
other. Clearly the upper limit of A3, the highest temperatu
our methods of observation detect, is likely to be the A3 of
poorest in carbon, because A3 rises as the carbon content fal

The dilatational method is affected in like manner by er
these sources.

The micrographic method has an advantage over others
may be applied to the undecarburized interior of the specim
is large enough to have such an undecarburized interior, an
it enables us to detect and reject specimens in which any con
degree of solidificational segregation persists. For that m
fitness of the specimens used in other methods should always
micrographically.

§ 33. *How Rapid is Surface Decarburization?*—It is very
atmospheric air, rather rapid in an atmosphere chiefly of
oxide and nitrogen, and not without effect even in a good
Thus heating in an air vacuum "usually better than 0.1 mm
Bureau of Standards for the purpose of taking heating and
curves, caused noticeable surface decarburization in the steel
0.21, 0.40, and 0.46 per cent. of carbon of Howe and Levy (S
VI., and II.), though not of the steels of 0.16, 0.92, 1.14, and
cent. of carbon. (See § 23.) In the case of the 0.40
carbon steel there was a "very thin skin entirely decarburiz
I found but slight suggestions of tarnish in this very surfa
§ 23.)

§ 34. *The Persistence of Solidificational Segregation* has rec
little attention. The parts which it impoverishes in carbo
like correspondingly low-carbon steel, whether in the the

or the micrographic method, and thus tend to give too
ns for A3. Heyn[18] showed macroscopically how extraor-
sistent this structure is, finding that an 8-day annealing at
ailed to wipe it out. N. T. Belaiew[19] has lately confirmed
ncy in a most striking manner, in particular showing that
endritic structure of a casting was affected little if at all,
om his illustrations, by a series of heat treatments which
ojourn of 24 hours at 1,160°.

he most striking case of the persistence of solidificational
is Stead's.[20] A bar of 1.2 per cent. carbon steel was im-
3-ton mass of blast-furnace slag, with which it cooled, of
mely slowly. Though the highest temperature must have
ort of the liquidus, because the bar retained not only its
so the maker's stamp marks undulled, yet a drop, consist-
of eutectic and as a whole containing more than 3 per
bon, liquated out and hung down. The liquating shows
perature must have risen above the solidus, because below
and equalization occur, and only above it can melting

of the metal should become fluid enough to migrate in a
s a whole retained this degree of solidity shows that, in
and rolling operations in which the original bar was pre-
ge degree of local enrichment in carbon must have per-
showing the slowness with which diffusion effaces segre-
r those particles which melted must have been so far
carbon as to be above their own liquidus; and their being
their liquidus means that that liquidus had been lowered
ess of carbon content, so as to bring it below the existing
, a temperature far below the liquidus of the bar as a
shown by the degree of solidity which it retained. Hence
great gap between the local liquidus of these residual
carbon and the liquidus of the bar as a whole, and hence
persistence of uneffaced initial local enrichment in carbon.
ness of diffusion is shown here in an additional way.
stration indicates an extremely sharp carbon gradient at
of the liquated drop and the body of the bar. The sharp-
gradient indicates the feebleness with which diffusion
egation even at this very high temperature close beneath

sche *Untersuchungen, Verhandlungen des Vereins zur Beförderung des Gewerb-*
1904).
*Métallurgie*, vol. ix., pp. 669, 671 (Sept., 1912).
cal Use of the Iron-Carbon Equilibrium Diagram, *Proceedings of the North-*
*tution of Engineers and Shipbuilders*, vol. xxix. (Jan. 24, Feb. 14, 1913).

the eutectic melting point, 1,130°. For it is clear that i
tremely slow cooling the sojourn at temperatures but sligh
1,130° must have been very long; and unless diffusion
slow it would have flattened the carbon gradient at this ju
well as elsewhere.

That such segregation, though it is no doubt made les
by diffusion in long heating, may yet persist to a mark
in gray cast-iron is shown by Figs. *C* and *D*, Row 14,
These represent the sodium picrate etching of a dendrite
sisted after five heatings to 800° each followed by a slo
in a button made by melting Professor C. F. Burgess's e
iron with sugar charcoal in a magnesia crucible brasqued
charcoal. It contained combined carbon, 1.598; graphi
total carbon, 2.954; silicon, 0.040; manganese, none; ph
0.050; sulphur, 0.035. After this sodium picrate etching t
tite and graphite present would be dark. The great contras
the lightness of the dendrite trunks and branches and the d
the fillings between them shows that very marked solid
segregation of carbon persisted even after these heatings.

I must admit that the persistence in the cases whic
quoted may have been due to the presence of phosphoru
in only one of them is the phosphorus known, and in this
enough, 0.05 per cent., to exaggerate the persistence mater
the Stead case the phosphorus was probably low, because r
of steel of 1.20 per cent. carbon stamped with the mak
usually contain not more than 0.03 per cent. of phosphoru

We reasonably expect the effect of solidificational segrega
most serious in unforged castings, because the drawing o
metal in forging or rolling lessens proportionally the dista
covered by diffusion, and probably for other reasons, such as
tural instability which the mechanical distortion causes. Th
two series, those of Heyn and of Goerens and Meyer, 2 and
14, are unforged castings. The high position of Series 4 is
to its being an Ac series.

It is true that a comparison of the heating and cooling
certain unforged castings of nickel steel with the curves of
material after forging gave Carpenter, Hadfield, and Long
indication of such segregation, at least none that can be r

[21] See the Author's Fig. 9, *Bulletin* No. 71, Nov., 1912, opposite p. 1220.
one of the dendrites in this same specimen.

[22] Seventh Report, Alloys Research Committee, *Proceedings of the Institutio*
*ical Engineers*, 1905, p. 927.

of their published plottings. Whether this is referable
nce of nickel in restraining segregation, to an effective
nealing, or to what other cause I do not know. In spite
nce I believe that uneffaced solidificational segregation is
se the observed position of A3 materially, in view of the
at the first deposited layers in iron-carbon alloys are
orer in carbon than the average of the specimen; (2) that
cates that this local impoverishment may be effaced only
; (3) that the shortage of ferrite found by Heyn and by
Levy itself points towards the peristence of such impover-
) that if such impoverishment persisted it would inevitably
al elevation of A3.

*e Persistence of Transformational Segregation.*—The data
nd Meyer[23] and of Jung[24] have been interpreted as prov-
s segregation is effaced rapidly; but however well they
hat the carbon gradient is rapidly flattened very mate-
are hardly competent to show how rapidly that flattening
leted.[25] The persistent reports that after grain refining
of the old ferrite grain boundaries may be disclosed by
g and cooling slowly, argue that a considerable degree of
onal segregation may long persist, the impoverishment of
se grain boundaries, which the grain refining aimed to
ining to a degree sufficient to induce a more abundant
of ferrite there in the subsequent slow cooling. If, as is
s persistence of the old grain boundaries is greater in
han in purer steels, that points to its being due to per-

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vol. vii., p. 310 (1910).

*le Zeitschrift für Metallographie,* vol. i., p. 215 (1911).

d Meyer found that, when hypo-eutectoid steel was cooled to 840°, within
ion range for their steel, held there, and then quenched, the quantity
nizable increased on prolonging the holding from 5 to 15 min., but not
ing. But very careful and extended planimetric measurements would be
that this further holding did not actually increase the quantity of ferrite
leed Howe and Levy found a very great increase in detectable ferrite on
r holding at 750° from 36 to 108 min., so that 36 min. is clearly far from
complete the structural equilibrium at 750° (*Internationale Zeitschrift für*
ol. iii., p. 4 (1912)).

hat some pro-eutectoid cementite in his steel of 1.33 per cent. of carbon
cooling to 950°, so that the line *SE* cuts this carbon content above 950°.
l that, when this same steel was heated up to 975° and held there, the pro-
ite initially present failed to redissolve completely in 3 min., but dissolved
nrecognizable in 4.5 min. This certainly implies a pretty rapid flattening
adient at 975°; yet it does not show that the completion of this flattening
t this temperature. To be more specific, it implies that, with this high
the transformational segregation residual at the end of 4.5 min. is not
the observed position of *SE* by 25°.

§ 31. *Other Causes of Error.*—These are set forth in T
Let us pass them in review.

*Lag, Undercooling, or Hysteresis.*—The failure of pure water t
till after the temperature has fallen materially below 0°, or
recalescence in steel to set in till the temperature has falle
degrees, perhaps even 100°, below the temperature at whi
due, Ae1, explains itself as a sort of molecular inertia, or rel
to break away from the initial grouping and adopt that whi
becomes due. This may be called lag proper.

It is asserted that the melting of a pure substance is not su
lag of this kind; and this may be true of the transformation
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for the freezing point, and for the transformation points whic
below the freezing point in the solid metal, lag proper ma
only in cooling and never in heating up. This phil
portant question need not concern us here. Suff
that if the solid is composite its actual melting
temperature at which a given transformation
tectable velocity in heating up, may be raised ma
temperature at which melting or transformation is a
reflection shows. Mix spherical quartz pebbles with sp
corundum and others of hard burnt lime, in the fusible pro
6 $SiO_2$, $Al_2O_3$, 3CaO, the melting point of which is reported as
Set this mixture in a furnace which is at 1,600°. When the
reach 1,300° no fusion will occur, because nowhere can any
in that fusible proportion actually exist. Silica is in cont
lime at certain points and with alumina at others, but at no
it in contact with both. No fusion will occur till the tem
reaches the melting point of the most fusible silicate of lin
alumina, the most fusible mixture of substances actually in
because though the silicate of lime and alumina is more fusi
any of its components that silicate cannot form except whe
components are in actual contact. Spheres of these three
not touching each other at all could be held forever at 1,300°
melting, if we ignore sublimation.

Heat up pearlite, and its constituent lamellæ of ferrite and
tite may indeed begin to unite to form austenite at their act
faces of contact at the instant of reaching Ae1; but the pa
that ferrite and that cementite which lie back of that contact
unite till, by the dissolving action of the resultant contact a
they are brought together. As the temperature rises above
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cementite which adjoins it on one side and some of the
n adjoins it on the other; and these two diffusing out
stenite enable its alongshore layers to dissolve more ferrite
te, and thus the whole of the pearlite at last transforms
te. But as long as the temperature is only strictly Ae1,
contact mass of austenite can dissolve neither ferrite nor
om the masses between which it forms the contact; only
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the temperature rises, and as through this process of dif-
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sultant heat absorption accelerates till it becomes detect-
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ectable heat absorption, *i. e.*, the higher will be the ob-
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re are no nuclei of ferrite present to induce the precipita-
te on cooling to the true solubility line Ae3, that precipi-
bly can never begin till the supersolubility or lability line
eached, so that a certain degree of lag in cooling is prob-
ble. But it is not clear that any measurable lag need
eating up, reabsorption completing itself at Ae3, if the
low enough.

How wide this metastable range is between the solubilit Ae3 and the supersolubility curve above which precipitation occur on cooling, our present data are too contradictory t The wide gap between the end of Ac3 and the beginning of the data of Table IX., after p. 1136, indeed point to a pre metastable range. On the other hand the fact that the very Ar3 data of Heyn, Series 2 of Fig. 14, are either above or but below the Ac3 data of Burgess and Crowe, Series 4, goes that this range is narrow, and so does the fact that in the st temperature micrographic determinations of Goerens and Me in those of Howe and Levy, Ac3 and Ar3 occurred within t rather narrow temperature gap. It is to be remembered that because of lag, the gap between the observed Ac3 and A exceed the width of this range by an indeterminate amour can never be narrower than this range.

Because lag increases with the rate of heating or cool thermal method is at a disadvantage compared with the st temperature methods, such as the micrographic, and such as tational may be made, because in these methods the holdin tionary temperature may be made long enough to lessen th of lag very greatly, possibly to a quantity which is negligibl present purposes. It may be possible to carry out the thermal with coolings and heatings so extremely slow as to reduce lag indeed to a negligible quantity, but in the experiments here tion this has not been done. The difficulties, especially in of a very slow yet regular rise of temperature, are serious.

The actual lowering of the beginning and maximum of hastening the cooling as found by Osmond is as follows:

TABLE VI.—*Influence of Rate of Cooling on Ar*1.

| Cooled. | Semi-Hard Steel, 0.57 Per Cent. C. | | | Hard Ste 1.25 Per Ce | |
|---|---|---|---|---|---|
| | Time of Cooling Between 685°–658°. | Recalescence Ar 1. Beg. ° C. | Max. | Time of Cooling Between 705°–678°. | R Be |
| 1. Gently in tube | 116 seconds | 653 | 656 | 49 seconds | 6 |
| 2. As usual in tube | 24.5 seconds | 648 | 655 | 20 seconds | 6 |
| 3. Rapidly in air | Not measured | 637 | 640 | Not measured | 6 |
| 4. Very rapidly, quenched in water | Not measured | Absent | | Not measured | |
| Lowering from lines 1 to 3 | ..... | 16° | 16° | ..... | 1 |

Incidental variations in the rate of heating or cooling of t used in this investigation caused a change of about this same

n the temperature of Ar1 maximum, but no recognizable
ne temperature of Ac1 maximum, as is shown in Table

II.—*Influence of Rate of Heating or Cooling on Thermal Ac1 and Ar1 (Howe and Levy).*

| arbon. r Cent. | Manganese. Per Cent. | Rate of Heating. Seconds per 1° C. | Ac1. Maximum °C. | Rate of Cooling. Seconds per 1° C. | Ar1. Maximum °C. |
|---|---|---|---|---|---|
| 0.40 | 0.16 | 4 | 736 | ..... | ..... |
| ..... | ..... | 8 | 735 | ..... | ..... |
| ..... | ..... | 12 | 731–736 | ..... | ..... |
| 0.46 | 1.21 | 4 | 729 | 6 | 661–665 |
| ..... | ..... | 8 | 730 | 6½ | 661–666 |
| ..... | ..... | 14 | 731 | 16 | 666–671 |
| 0.92 | 0.12 | 5 | 739 | ..... | ..... |
| ..... | ..... | 8 | 734 | ..... | ..... |
| ..... | ..... | 11 | 738 | ..... | ..... |
| 1.14 | 0.24 | 3 | 738 | ..... | ..... |
| ..... | ..... | 3½ | 737 | ..... | ..... |
| ..... | ..... | 13 | 739 | ..... | ..... |

Burgess's more sensitive observations lowered the end of
on retarding the rate of heating from 3.9 to 4.3 seconds
and by 20° on retarding it to 9 seconds per degree, as is
able VIII., p. 1133.

lag of A3 may be very considerable is shown by the ob-
f Burgess and Crowe as pointed out in § 10. With their
asurements there was a gap of from 38° to 74° between
Ac3 and the beginning of Ar3 for given specimens.

o line 1 of Table V., both for the hypo-eutectoid and for
oid steel, lag is likely to lower the observed position of
l in cooling curves, and to raise Ac3 as observed in heat-
nd such effect as it has in stationary temperature methods
this same kind.

*rors from Local Lowering of the Carbon-Content*, (2), (3),
general surface decarburization, and uneffaced segrega-
r solidificational or transformational, tend to raise the
mperature of A3 of hypo-eutectoid steels above Ae3.
this cause is especially to be dreaded in the thermal
it may be of importance in the other methods, such as
nal and the electrical resistance and even in the micro-
hod. A glance at the conditions of the thermal method
lear.

erature of A3 rises rapidly as the carbon content falls.
ere is superficial decarburization, the A3 of the outside

of the specimen lies correspondingly higher than the A3 sought, the A3 corresponding to the chemical compositio specimen undecarburized. In taking a cooling curve w superficially decarburized specimen, if for simplicity we igno break in the curve will begin as soon as the cooling reaches of the decarburized outer layer, and before it reaches the A sponding to the undecarburized steel. We are likely to higher first break as the Ar3 of the specimen, because it is t ning of the break that we look for. The more sensitive th the higher will it detect such a slight break.

What is thus true when a local impoverishment in carbon by surface decarburization is true when it is caused by seg whether solidificational or transformational. We usually segregation as the enrichment of one part, *e. g.*, in carbon enrichment of one part must needs cause the impoverishme other. Clearly the upper limit of A3, the highest temperatu our methods of observation detect, is likely to be the A3 of poorest in carbon, because A3 rises as the carbon content fa

The dilatational method is affected in like manner by e these sources.

The micrographic method has an advantage over others may be applied to the undecarburized interior of the specim is large enough to have such an undecarburized interior, a it enables us to detect and reject specimens in which any co degree of solidificational segregation persists. For that n fitness of the specimens used in other methods should always micrographically.

§ 33. *How Rapid is Surface Decarburization?*—It is very atmospheric air, rather rapid in an atmosphere chiefly of oxide and nitrogen, and not without effect even in a good Thus heating in an air vacuum "usually better than 0.1 mm Bureau of Standards for the purpose of taking heating an curves, caused noticeable surface decarburization in the stee 0.21, 0.40, and 0.46 per cent. of carbon of Howe and Levy (S VI., and II.), though not of the steels of 0.16, 0.92, 1.14, and cent. of carbon. (See § 23.) In the case of the 0.40 carbon steel there was a "very thin skin entirely decarburiz I found but slight suggestions of tarnish in this very surfa § 23.)

§ 34. *The Persistence of Solidificational Segregation* has rec little attention. The parts which it impoverishes in carbo like correspondingly low-carbon steel, whether in the the

in the temperature of Ar1 maximum, but no recognizable the temperature of Ac1 maximum, as is shown in Table

VII.—*Influence of Rate of Heating or Cooling on Thermal Ac1 and Ar1 (Howe and Levy).*

| Carbon. Per Cent. | Manganese. Per Cent. | Rate of Heating. Seconds per 1° C. | Ac1. Maximum °C. | Rate of Cooling. Seconds per 1° C. | Ar1. Maximum °C. |
|---|---|---|---|---|---|
| 0.40 | 0.16 | 4 | 736 | ..... | ..... |
| ..... | ..... | 8 | 735 | ..... | ..... |
| ..... | ..... | 12 | 731–736 | ..... | ..... |
| 0.46 | 1.21 | 4 | 729 | 6 | 661–665 |
| ..... | ..... | 8 | 730 | 6½ | 661–666 |
| ..... | ..... | 14 | 731 | 16 | 666–671 |
| 0.92 | 0.12 | 5 | 739 | ..... | ..... |
| ..... | ..... | 8 | 734 | ..... | ..... |
| ..... | ..... | 11 | 738 | ..... | ..... |
| 1.14 | 0.24 | 3 | 738 | ..... | ..... |
| ..... | ..... | 3½ | 737 | ..... | ..... |
| ..... | ..... | 13 | 739 | ..... | ..... |

Burgess's more sensitive observations lowered the end of on retarding the rate of heating from 3.9 to 4.3 seconds and by 20° on retarding it to 9 seconds per degree, as is Table VIII., p. 1133.

lag of A3 may be very considerable is shown by the ob- of Burgess and Crowe as pointed out in § 10. With their easurements there was a gap of from 38° to 74° between Ac3 and the beginning of Ar3 for given specimens.

to line 1 of Table V., both for the hypo-eutectoid and for toid steel, lag is likely to lower the observed position of ed in cooling curves, and to raise Ac3 as observed in heat- and such effect as it has in stationary temperature methods of this same kind.

*Errors from Local Lowering of the Carbon-Content*, (2), (3), n general surface decarburization, and uneffaced segrega- ner solidificational or transformational, tend to raise the emperature of A3 of hypo-eutectoid steels above Ae3. this cause is especially to be dreaded in the thermal t it may be of importance in the other methods, such as onal and the electrical resistance and even in the micro- thod. A glance at the conditions of the thermal method clear.

perature of A3 rises rapidly as the carbon content falls. here is superficial decarburization, the A3 of the outside

the eutectic melting point, 1,130°. For it is clear that tremely slow cooling the sojourn at temperatures but slig 1,130° must have been very long; and unless diffusion slow it would have flattened the carbon gradient at this j well as elsewhere.

That such segregation, though it is no doubt made les by diffusion in long heating, may yet persist to a mark in gray cast-iron is shown by Figs. *C* and *D*, Row 14, These represent the sodium picrate etching of a dendrite sisted after five heatings to 800° each followed by a slo in a button made by melting Professor C. F. Burgess's e iron with sugar charcoal in a magnesia crucible brasqued charcoal. It contained combined carbon, 1.598; graphi total carbon, 2.954; silicon, 0.040; manganese, none; pl 0.050; sulphur, 0.035. After this sodium picrate etching t tite and graphite present would be dark. The great contra the lightness of the dendrite trunks and branches and the d the fillings between them shows that very marked solid segregation of carbon persisted even after these heatings.

I must admit that the persistence in the cases whic quoted may have been due to the presence of phosphoru in only one of them is the phosphorus known, and in this enough, 0.05 per cent., to exaggerate the persistence mate the Stead case the phosphorus was probably low, because r of steel of 1.20 per cent. carbon stamped with the mak usually contain not more than 0.03 per cent. of phosphoru

We reasonably expect the effect of solidificational segrega most serious in unforged castings, because the drawing metal in forging or rolling lessens proportionally the dista covered by diffusion, and probably for other reasons, such as tural instability which the mechanical distortion causes. T two series, those of Heyn and of Goerens and Meyer, 2 and 14, are unforged castings. The high position of Series 4 i to its being an Ac series.

It is true that a comparison of the heating and cooling certain unforged castings of nickel steel with the curves of material after forging gave Carpenter, Hadfield, and Long indication of such segregation, at least none that can be

[21] See the Author's Fig. 9, *Bulletin* No. 71, Nov., 1912, opposite p. 1220. one of the dendrites in this same specimen.

[22] Seventh Report, Alloys Research Committee, *Proceedings of the Institutio ical Engineers*, 1905, p. 927.

of their published plottings. Whether this is referable ence of nickel in restraining segregation, to an effective nealing, or to what other cause I do not know. In spite ence I believe that uneffaced solidificational segregation is se the observed position of A3 materially, in view of the hat the first deposited layers in iron-carbon alloys are orer in carbon than the average of the specimen; (2) that icates that this local impoverishment may be effaced only ; (3) that the shortage of ferrite found by Heyn and by Levy itself points towards the peristence of such impover- ) that if such impoverishment persisted it would inevitably cal elevation of A3.

*he Persistence of Transformational Segregation.*—The data and Meyer[23] and of Jung[24] have been interpreted as prov- is segregation is effaced rapidly; but however well they that the carbon gradient is rapidly flattened very mate- are hardly competent to show how rapidly that flattening leted.[25] The persistent reports that after grain refining of the old ferrite grain boundaries may be disclosed by g and cooling slowly, argue that a considerable degree of ional segregation may long persist, the impoverishment of se grain boundaries, which the grain refining aimed to ining to a degree sufficient to induce a more abundant of ferrite there in the subsequent slow cooling. If, as is is persistence of the old grain boundaries is greater in than in purer steels, that points to its being due to per-

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, vol. vii., p. 310 (1910).

*le Zeitschrift für Metallographie*, vol. i., p. 215 (1911).

d Meyer found that, when hypo-eutectoid steel was cooled to 840°, within tion range for their steel, held there, and then quenched, the quantity nizable increased on prolonging the holding from 5 to 15 min., but not ing. But very careful and extended planimetric measurements would be that this further holding did not actually increase the quantity of ferrite deed Howe and Levy found a very great increase in detectable ferrite on r holding at 750° from 36 to 108 min., so that 36 min. is clearly far from complete the structural equilibrium at 750° (*Internationale Zeitschrift für* vol. iii., p. 4 (1912)).

that some pro-eutectoid cementite in his steel of 1.33 per cent. of carbon cooling to 950°, so that the line *SE* cuts this carbon content above 950°. d that, when this same steel was heated up to 975° and held there, the pro- tite initially present failed to redissolve completely in 3 min., but dissolved nrecognizable in 4.5 min. This certainly implies a pretty rapid flattening radient at 975°; yet it does not show that the completion of this flattening t this temperature. To be more specific, it implies that, with this high the transformational segregation residual at the end of 4.5 min. is not the observed position of *SE* by 25°.

sistence of transformational segregation, because it is kno other sources that the diffusion of phosphorus is slow. A great persistence of solidificational segregation makes the pe of transformational segregation antecedently probable.

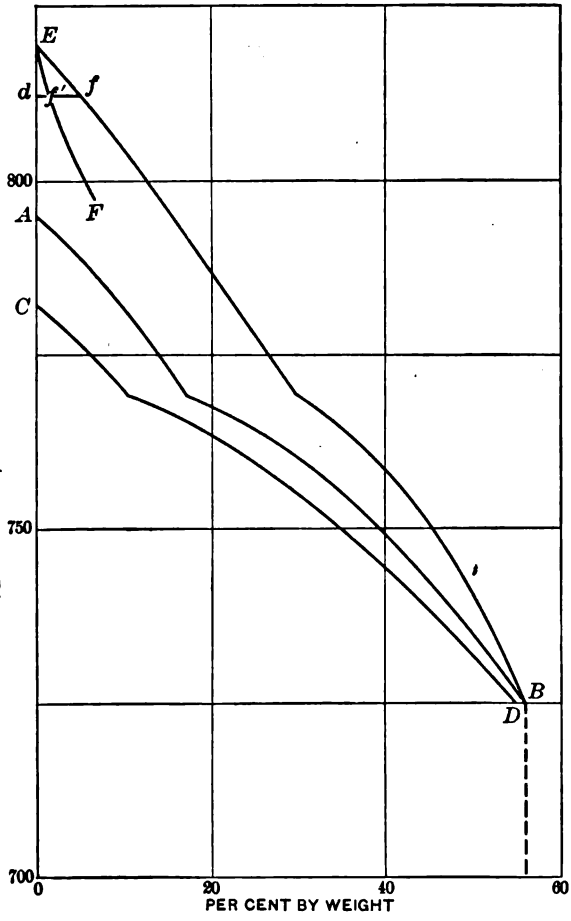
The dark lines in Fig. D, Row 8, Plate 3 (Steel E), argu persistence of this transformational segregation. This s after coarsening, was heated apparently to above the end held there for 30 min., and quenched in water. The dark dently represent the old coarse grain boundaries, such as sh in specimens quenched from below the end of Ac3. Th dark can be interpreted as meaning that the transformation ceeded further here than elsewhere in the direction of tro the ground that the further the transformation goes the dark etching, and this further transformation may be interpreted ing that the carbon content here is lower than elsewhere ground that the lower the carbon content the faster is the t ation; and this in fine as meaning that the transformational ishment in carbon here has not been completely effaced. B other hand the darkening is capable of being interpreted as that there is still in these old boundaries a quantity of unre ferrite which, though too small to be detected microscopical sufficient to intensify the etching by difference of potential.

The darkening clearly represents the persistence of some formational segregation, and not the grain boundaries of the formed in the present heating, because these grains must very fine. Since the time when the austenite was coarsene give rise to the coarse boundaries the loci of which now sh the metal has cooled below the transformation range, thus d that austenite, and has been heated up through that ran giving rise to a new and finer set of grains, the boundaries are not here shown. The dark lines are the persisting seg caused by a grain system which itself has ceased to exist.

Moreover, the shortage, inferred in § 37, of recognizable the micrographs of Howe and Levy argues for considerable tion in their specimens. The fact that their specimens a bars indicates that there has been good opportunity here f facement of solidificational segregation. This raises a p that the segregation which those specimens are inferred t transformational at least in part, and that it has persisted their 30 or even 60 min. holding at or above A3.

§ 36. *Specific Influence of the Persistence of Segregation whet ficational or Transformational.*—I will now explain a spec

egregation must needs have, an effect which is prominent of Heyn and of Howe and Levy. To fix our ideas let us such segregation actually persists, and that it has left the in Fig. 13 impoverished in carbon, and the spot *H* carbon, with a carbon gradient of unknown shape from et this figure refer to a steel of 0.40 per cent. of carbon,




ALCULATED QUANTITY OF PRO-EUTECTOID FERRITE PRECIPITATED:
*AB* by a homogeneous steel of 0.40 per cent. of carbon.
*EF* by such a steel if segregated.

onsider the precipitation in falling temperature. That ue of precipitation is true also of reabsorption, *mutatis*

*AB* in Fig. 12 represents the percentage of pro-eutectoid should be present in such a steel at various temperatures

in the transformation range, on the assumption that the line Fig. 14 is the true Ae3.

The precipitation of ferrite at given temperature at each segregated steel such as represented in Fig. 13 correspond the average carbon content of the whole specimen, but to th content of that spot. Because the spots *LLL* have bee erished in carbon, the precipitation of ferrite in cooling wi not at the Ae3 of the specimen as a whole, but at the highe these particular spots. But as we naturally take the tempe which this first precipitation is observed to be Ae3 for th content of the specimen as a whole, we incorrectly put A3 0.40 carbon steel at *E*, Fig. 12, above *A* where it truly belo our whole series of specimens with which we are determi locus of A3 is segregated in like manner, we shall set t throughout correspondingly too high. Calculating from t the quantity of ferrite due at different points within the tra tion range of our 0.40 carbon steel, we shall get a false c instead of the true curve *AB*.

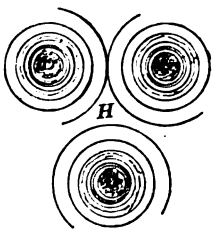



FIG. 13.—PRECIPITATION OF FERRITE ABOUT SPOTS IMPOVERISHE IN CARBON SEGREGATION.

But the quantity of ferrite actually precipitated will follow like *EF*. Thus on cooling to *d* the whole of the specim precipitate a quantity of ferrite *df* if *E* were truly the Ae3 per cent. of carbon, and if the false high position found for t of Ae3 were the true locus. But in fact it is only the very sp that will precipitate ferrite in that proportion, *df*. At temp the shore layers about *LLL* will precipitate less and less as tance from *LLL* increases, till at a very short distance fro the carbon content is so much greater than that of *LLL* tha rite at all will precipitate at temperature *d*. Thus there marked shortage of precipitated ferrite in the specimen as below that calculated from the observed position of the c The ferrite actually precipitated will be *df'* instead of *df*.

this specific cause of error goes the gap between *EF* and decrease progressively, but *EF* should never cross *AB*, nothing here to make the quantity of precipitated ferrite at due according to the true Ae3.

*hortage of Recognizable Ferrite.*—The reasoning in § 36 ena- nterpret the shortage of recognizable ferrite in specimens within the transformation range, observed by Heyn and by Levy, a shortage which means that, as the temperature elow that at which the first precipitation of ferrite is ob- increase in the quantity of ferrite recognizable, instead of curve of the family of *AB* and *CD*, follows one like *EF*. we may take this to mean that there is marked heteroge- n both sets of specimens, and the existence of this hetero- s would tend to raise the observed locus of Ae3 above the n of Ae3.

probably flattens down pretty rapidly the extreme sharp- carbon gradient which the expulsion of ferrite into the aus- boundaries causes in cooling through the transformation the further flattening of that gradient is a self-retarding iffusion completes itself only asymptotically. In fact a ep enough to raise the observed position of A3 measura- ably very persistent, because Howe and Levy found that of unreabsorbed ferrite decreased only extremely slightly ing the holding period at their quenching temperatures 60 min. (§ 14.)

reting this shortage of ferrite we should remember how hese proportions often are. On inspecting pearlite one infer that the proportions of ferrite to cementite in it equal instead of being 6 : 1. Mr. Levy and I have pointed eatly the apparent proportion of pro-eutectic cementite the true proportion, especially on low magnification.[26] t matter the quantity of recognizable ferrite even in our rsened specimens, Figs. *A* of Rows 4 to 9, Plates 2 and 3, spection much less than that which is due theoretically.

*he Precipitation of "Quenching Ferrite" or Cementite, i. e.*, of ectoid element during the quenching proper, tends to raise A3 above Ae3. If, equilibrium being reached at Ae3, free pro-eutectoid element present before quenching, yet incipiency of the cooling cannot be extremely rapid in of the specimen, it may result that some ferrite or cemen-

History of Pro-Eutectic Cementite, *Proceedings of the International Association rials*, VI. Congress, II., 1912, and the discussion.

tite precipitates. Inferring that this was present at the qu
temperature itself we should infer that this temperature wa
Ae3. To raise the quenching temperature somewhat highe
be to induce a more rapid cooling from Ae3 down, and thus
to prevent detectable precipitation of pro-eutectoid eleme
should thus come to take such higher temperature for Ae3.
this cause of error tends to raise the observed position of A3
Ae3. But it is unlikely to cause any important error excep
case of very low carbon steel, with less than say 0.10 per cen
bon, in view of the extreme sluggishness with which Mr. Le
found that ferrite coalesces into ready visibility in steel of
cent. of carbon.[27] The coalescence of pro-eutectoid cemen
visibility seems to be still slower.

§ 39. *Failure to Detect the First (or Last) Traces of the Pro-
Element.*—We might well fail to recognize the earliest pre
ferrite because of the tardiness with which it coalesces into
able masses.

This same slowness of coalescence may in another way pre
recognizing the beginning of precipitation and the end of
tion in specimens quenched even after a long holding at
temperature, as in the experiments of Howe and Levy. Fo
such nominally constant temperatures there are slight u
oscillations of temperature, and moreover the solution pressu
ferrite is different towards different austenite masses presen
these things should lead to a constant slow passage of fer
solution and its reprecipitation in the form of an emulsion,
cause this freshly precipitated ferrite coalesces so slowly in
nizable masses, there may be at all times a certain quantity
ognizable ferrite, a quantity which would be the smaller the
oscillations of temperature. The temperature at which w
recognize the first or last traces of ferrite would be that at w
quantity of ferrite actually present was in excess of that wh
long remains emulsified, that excess having coalesced into
able masses, a temperature evidently below the true Ae3.

The effect of this failure would be to make the observed
of ferrite lie in a curve like *CD*, Fig. 12, below *AB*, but app
it gradually as the temperature falls, because the larger the
of ferrite present the more thoroughly should it coalesce, s
as the quantity of unconsumed carbon left in the ash of coal

[27] Belated Coalescence vs. Balling Up as the Cause of the Degradation of th
Structure of Hypo-Eutectoid Steel, *Internationale Zeitschrift für Metallographie*, v
(1912).

en to specimen as the quantity of ash decreases, or as a
urns into butter faster than a poor one.
ration of the ideas set forth in Figs. 12 and 13 inclines
that this cause is not likely to lower the observed position
served micrographically, to materially below Ae3. I in-
that if the precipitation (to fix our idea) of ferrite follows
*EF*, the rate at which the precipitated ferrite increases
low that a considerable gap may exist between the higher
at which ferrite begins locally to precipitate, and the
rature at which its precipitation reaches such a quantity
f it becomes recognizable; so that failure to detect the
lly precipitated might lower the observed position of A3
below *E* of Fig. 12. But in doing this it is merely cor-
her error, that due to local impoverishment in carbon; it
ing the observed temperature from the falsely high point
he true point *A*. The important thing to recognize is that,
, the shape of *AB* shows that the increase in the quantity
e to precipitate is so extremely rapid as the temperature
this true point *A*, that before the temperature sinks mate-
*A* the precipitation of ferrite should be so abundant that
xpect it to escape detection.

## III. Discussion of the Chief Data as to A3.

*General.*—The relations between the chief data are shown
l way by plotting them in Fig. 14, after p. 1136, but
ter by assembling them in Table IX., in which they
d from 1 (Osmond's thermal end of Ac3) to 13 (Burgess
thermal Ar3 for high-manganese steels), with the inter-
s in their general temperature order, the order which fits
r comparison, in spite of the fact that in some cases it is
sed by their crossing each other.

11, shows the micrographic results of Howe and Levy
mal upper limit of A3 of Burgess and Crowe, for the
nese steels, Series 13, together with the corresponding
lower-manganese steels of the same observers reproduced
son from Fig. 1.

rrections.—In drawing curves of Fig. 14, I have taken
ties which seem necessary in the search for truth. First,
ted certain thermal beginnings of Ar3 and endings of
ground that neither where placed by their observers nor
these cooling or heating curves is there any inflection so
o carry appreciable weight when in conflict with strong

evidence. The points thus rejected are given in Fig. 14, but inclo in circles to indicate that they are neglected.

Second, I have lowered summarily some of these beginnings endings from the position assigned by their observers, thus lessen the discrepancies. In Series 1 and 5 there is reason to fear import surface decarburization, and in Series 3 and 7 there is reason to that the combined effects of such decarburization as there was, of uneffaced solidificational segregation due to the unforged cast s of the specimens, are considerable. Here we may assume reasona that the last faint tailing out of Ac3 and the first creeping in of represent these parts, thus impoverished in carbon. For the purp of our present comparison we may neglect these extreme ends, take that part of the heating or cooling curve in which the inflec is strong enough to indicate that it represents a considerable frac of the whole specimen. In this way allowance may be made for face decarburization where it is not extreme, but hardly for soli cational segregation in unforged castings. The first deposited la have about half the carbon content of the specimen as a whole, such diffusion as occurs in the cooling of a small casting from liquidus down is likely to leave a very large fraction of the w materially impoverished in carbon, and hence with a materi higher A3 than that of the specimen as a whole.

Series 11 hardly needs such a correction, both because of the sl of its curves and because segregation and decarburization were p ably very slight in it. It would be hard to apply such a correc to the micrographic results, Series 2 and 6, without very careful p metric determination of the rate of increase of the ferrite pres and indeed in Series 6 the effects of decarburization are remove ignoring the outside of the specimens. To the dilatational result such correction can be made, because one only of the observed cu is accessible. But corrections have been applied to certain of data in Series 1, 3, 5, and 7 as explained under Fig. 14.

§ 42. *What are the Discrepancies?*—Lag must necessarily the Ac3 results above and lower the Ar3 results below Ae3. lag the only cause of discrepancy, then the temperature-order shoul

1st and highest, the thermal Ac3 results in the order of the r ity of heating; Series 1 and 4;[28]

2d, the electric resistance Ac3, Series 8, in reaching which

[28] Lest the reader be confused by assertions lately made that there is no lag in trans ations in rising temperature, let me point out that in these assertions the word "tran ation" is used in the sense of "fusion," and not in the usual sense of "a transform within the solidified metal," as in the present case.

, though indeed rising, rose so slowly that this series
e close to the stationary-temperature ones;
ationary temperature results, Series 2, 6, and 12;
dilatational Ar3 results, Series 9, in reaching which the
probably fell decidedly more slowly than in the thermal

hermal Ar3 results, inversely in the order of rapidity of
slowest highest and the fastest lowest, Series 7, 11, 3,

up, 1, 4, 8, 2, 6, 12, 9, 7, 11, 3 (10), 5.
easonable allowance for capriciousness of lag, the condi-
ch are so little known, the discrepancies between Series
, 8, 9, 10, and 11 are not greater, indeed they are less,
l be expected from observational errors. Thus the sta-
perature, series 6, lies as it should below the thermal Ac3
4, which two moreover lie in their expected order; and
the electric resistance and dilatational series 8 and 9, and
rn lie above the thermal Ar3 series 10 and 11. That
ich ought to be above the thermal Ar3 series 5, is 20°
nd 13° below another of its points is hardly surprising in
serious opportunity for decarburization in series 5, of the
of its results, and of probable error in correcting Series 5
the temperatures then used for calibration. The one
ries 8 which deserves weight is so ill defined in the origi-
at the surprise is rather that it fits in so well with the
than that it happens to lie a few degrees on the wrong
thermal Series 7. That some points of this latter series
stead of below the probably slower cooled Series 9, the
the wrong side reaching in only one case as much as 16°,
rising in view (1) of the cast state of Series 7, (2) of the
sitiveness of the observational method of Series 7, both
ould tend to raise the observed beginning of Ar3, (3) of
lity that a calibrational error has lowered the observed
es of Series 9.[29]

d Grenet, *Bulletin de la Société d'Encouragement pour l'Industrie Nationale*,
, calibrated their couple for the boiling point of sulphur, 448°, and for the
f gold, 1,060°, and assumed that between these temperatures the deflections
meter were proportional to the temperature. This is not strictly true for
An examination of the calibration of five Pt, Pt–Ir–couples in use in the
laboratories of Columbia University shows that this assumption that the
ht between 445° and 1,063° would introduce an error of about 10° in two
15° in two, and none in the fifth. The temperatures which they assign to
points are very close to those now given by the United States Bureau of
.7° and 1,063 ± 3°.

Thus nine out of the twelve series are on the whole surprisin concordant, in view of the many sources of error. There remain th important Series, 2, 3, and 12, which are less closely in harm with the others. Indeed 2 and 12 for part of their length di widely from the rest.

Series 2, which, because it is a stationary temperature series, ou to lie near to Ae3, below the Ac3 and above the Ar3 series, is most of its length far above all other series. Series 3, which, beca it is a thermal Ar3 series with rather rapid cooling, ought to lie below the thermal Ac3 series, No. 4, and the stationary tempera series 6 and 12, lies much farther above some of them even as corrected than should be explicable by any errors in tempera measurement. Series 12, which, because it is a stationary temp ture method, should lie above all the thermal Ar3 series, 3, 5, 7, 11, lies for much of its length below 3, 5, and 7. That Series 1 for most of its length above Series 10 deserves little weight, bec Series 10 represents maxima of Ar3 which are an indeterminate tance below the beginning of Ar3.

§ 43. *The Discrepancies in Detail.*—Let us address ourselve these three discrepancies.

*Series 2.*—Its unduly high position is at once explained by the that its specimens are both in the cast state and rich in phosph a combination of conditions which, as shown in § 21, raises greatly. Attention is called to the fact that this raising effe phosphorus is greatest midway of the hypo-eutectoid range, a rently shading off to a small quantity at the ends of this range. agrees with Osmond's observation that in a very low-carbon (carbon 0.05 per cent.) the presence of even 0.38 per cent. of p phorus did not raise the maximum of Ar3 at all. I indeed find i curves that this phosphorus content raised the beginning of Ar about 10°, but this is only a small fraction of the raising effect o smaller quantity of phosphorus in Series 2, midway of the hypo-e toid range. The greatest depression of Series 12 below the ma the other Series also lies midway of this same range, decreasing idly towards either end, a depression which may be due in pa the pretty high manganese content of the specimens. But th probably only a coincidence, for no such relation between the dep ing effect of manganese and the carbon content can be traced an the high-manganese steels represented in Fig. 3.

§ 44. *Series 3*, which because it is an Ar3 series with rather fall of temperature should lie below the stationary temperature S 6, in fact lies nearly 20° above it. Because its rate of coolin

n that of the Ar3 series 7 and 11, it should lie below them, it lies from 25° to 55° above the former and from 45° to the latter, confining ourselves for the moment to the left k or point *O*. Its being above Series 6 and 11 may be its uneffaced solidificational segregation, and its being s 7 to its being raised more than Series 7 by that segre- ıch data as are given suggest that in Series 3 the sojourn vas not long enough to efface transformational segregation, may have helped to raise the temperatures of Series 3. ion already introduced probably suffices to cover surface tion, but not segregation.

*eries 12*:—The low position of Series 12 I cannot explain. xperiments, which were *in vacuo* throughout, the specimen to the predetermined temperature, held there for between min., and then dropped into an ice calorimeter, by means e heat evolved in the further slow cooling to 0° was deter- ıanks to the vacuum this cooling in the calorimeter was h to yield lamellar and even granular pearlite. Plotting s abscissa, and the temperature of entry into the calorim- nate, gives concordant curves in most of which Ac1 and Ac3 are indicated clearly by sharp breaks, the total heats alling in a straight line, and those between A3 and A1 in ıight line.[a] The point at which these lines if produced sect is indicated for each specimen by a four-sided star in ile the lowest observed temperature thus found to be above highest thus found to be below A3 are represented by the vertical lines drawn there for the several specimens. ne tries to explain the low position of Ar3 thus found by arge manganese content of the specimens. But we have that the effect of manganese in lowering the transforma- is chiefly through increasing lag, and but little through e Ae points. Hence in Series 12, with its long stay at emperature and its consequent lessening of lag, manga- have had the relatively small effect which it has in Series tually its effect is much greater, as is shown by consider- ns 3 of Series 12, and B and 2 of Series 13:

| C. | Si. | Mn. | P. | S. | A3. | Ac3. Of Burgess and Crowe. | Ar3. |
|---|---|---|---|---|---|---|---|
| 0.32 | 0.075 | 0.45 | 0.054 | 0.057 | 750–770 | ..... | |
| 0.32 | 0.122 | 0.41 | 0.004 | 0.022 | 810–820 | 845 | 777 from their line. |
| 0.46 | 0.094 | 1.215 | 0.017 | 0.015 | 765–775 | 785 | 725 from their points. |

ıen A2 is separated from A3, there is a break in this second line.

3 of Series 12 and B of Series 13 are practically identical in co sition, yet the A3 for 3 of Series 12 is not only some 50° below tl B of Series 13 but slightly below the bottom of the gap betwee Ac3 and Ar3 for this same specimen as found by Burgess and C though Ae3 should lie above the middle of this gap, so that it below the probable temperature of Ae3 which the data of Bu and Crowe imply. Indeed its 0.45 manganese seems to have l lowering effect much greater than that of the thrice as great m nese content of 2 of Series 13, to judge from the probable positi Ae3 in this latter specimen as indicated by the micrographic d Howe and Levy and the thermal data of Burgess and Crowe.

The manganese content of 3 of Series 12 cannot explain the wide gap, 75°, between its A3 and the Ar3 of Series 3, or the gap between it and the Ar3s of Series 5 and 7, without assigni manganese an effect in lowering Ae3 far beyond what the available data imply.

A systematic temperature error in Series 12 does not explai discrepancy, because for the low-carbon specimen of this serie position of A3 agrees with that of other observers. Moreover th the temperatures of Meuthen's A1s suggest that his temperatu general are slightly too low, yet it is by an amount conside smaller than that by which his A3s lie below those of Se and 6.

§ 46. *A Weighted Average* is calculated for what it is wort signing to the various series the weights indicated in column Table IX. These weights are based on the principles

(1) of assigning a maximum weight of 12;

(2) of deducting 2 for the cast as distinguished from the f state; and

(3) of deducting 2 for data based on heating or cooling curv stead of stationary temperature.

Beyond this I deduct arbitrarily as follows:

1 from Series 5, because the fixed temperatures on which it brations were based are not those of to-day. Though I hav rected these temperatures in accordance with Osmond's own i tions, yet even as thus corrected a certain residual error is st to be feared.[30]

3 from Series 6, because of probable error due to inability tect the last traces of unabsorbed ferrite, and to uneffaced transf tional segregation, the opportunity for which is greater here th

[30] *Journal of the Iron and Steel Institute*, vol. lxxi., 1906, No. iii., p. 451, foot-n

es. A smaller deduction might be justified in view of hese two errors are of opposite sign.

es 9, because with one exception the original curves are .

the small number of specimens represented at the left point *O*, I deduct 1 from Series 2, 4, 5, 11, and 12, and es 9.

impurity of the specimens I deduct 6 from Series 2, 1 5, 6 and 11, and 2 from Series 12.

temperature error indicated by the low position of A1, es 12.

ight Series 1 because it is confined to so very short a on content, and also because of Osmond's calibration t noted; nor Series 8 (*a*) for the former of these same cause the number of points is so small, and (*c*) because I y the points with any confidence in the original curves; because it represents maxima of Ar3, which lie such an distance below Ar3, itself an indeterminate distance

*and OS.*—Series 9 runs only to carbon 0.24, so that, were veight as far as this carbon content and then omit it, ere cause a purely fictitious break. A like difficulty rd to Series 4, 5, and 11; how may we meet it? Let as an example.

ely probable that Ae1-2-3 lies very near 725° and be- d 0.90 per cent. of carbon. But the direction of the all the curves of Fig. 14, from 2 to 8 inclusive, would 725° far at the left of carbon 0.85, and perhaps even arbon 0.70. Because it is in the highest degree improb- -2-3 lies so far to the left, it is probable that Ae3 in rns to the right, and all the curves of Fig. 14 which uestion indicate such a turning, 3, 6, and 7 indicating between 765° and 790°, where Roberts-Austen's line um of Ar3 also shows such a break, while Series 5 ightly higher, at about 810°. This turning may well ; but the present crude state of our knowledge is fitted esenting it as a break between two straight lines, as by most writers. The temperature of this break may be put at about 770°, or at about the maximum of A2, netic changes at which would naturally correspond to The part of Ae3 above this we may call *GOe*, and the is *OSe*.

Producing Series 9 to 770°, I use its weight in calculating th
*GOe* but not *OSe*. So *mutatis mutandis* with Series 4, 5, and 11
deed the data for *OSe* are so much scantier than those for *GO*
in my opinion carry so much less weight than the belief that *S*
725° and carbon about 0.90, that we may provisionally adop
position, and draw *OSe* as a straight line from it to where *G*
found by our weighting cuts the abscissa of 770°.

§ 48. *The Result.*—The broad black line *AAA* in Fig. 14
the position of Ae3 thus calculated.

This line should be received with great caution and only as
porary expedient, because of the necessary arbitrariness in calcu
it, and because the thermal data are such a poor foundation. I
an average calculated even from an infinite number of absolute
curate thermal determinations would be untrustworthy without
more than our present knowledge of the influence of the rate of
ing and cooling on the lag. Some may question the wisd
presenting this calculated line, holding it better to trust that w
have said of the contradictoriness of our data, of the causes of
and of the industrial importance of Ae3, would induce some na
laboratory to determine this line with these causes of error re
to a minimum. The close agreement between *AAA* and th
*HLM* of Howe and Levy is evidently in large part fortuitou
was not foreseen when the weightings were assigned to the s
series of data.

§ 49. *The Calculated Line Tested Against Late Determin
of A3 in Very Pure Iron.*—Table VIII. condenses the positi
A3 for extremely pure iron found in three late investigation
the line *AAA* calculated for Ae3, shown in Fig. 14, is corre
naturally expect its prolongation to cut the vertical axis at a
which agrees with these late data for A3 of carbonless iron. B
and the line *HLM* bear this test better than Heyn's line, whi
now has, I believe, seemed the most trustworthy. *AAA* cu
axis at about 919°, and *HLM* at 912°, temperatures compatible
all the data in Table VIII., because they are below all the Ac3
and above all the Ar3 beginnings.

Heyn's line even as corrected cuts the axis at about 931°,
therefore has the disadvantage of being above two of the Ac3
which it could not be if those ends are correctly observed. A
because Heyn's line is an Ar3 beginning it should be tested a
the Ar3 beginnings of Table VIII.; but its intersection with the
from 27° to 61° above those beginnings, a fact which tends t
port my suspicion that his results are raised above the A

| | | C. | Si. | Mn. | P. | S. | | Rate of heating per min. | °C. | °C. | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | G. K. Burgess | Very pure electrolytic iron from Prof. C. F. Burgess. | | | | | | 3.9 | 955 | ..... | 51 | |
| 2 | G. K. Burgess | | | | | | | 4.3 | 950 | ..... | 46 | |
| 3 | G. K. Burgess | | | | | | | 9 | 935 | ..... | 31 | |
| 4 | G. K. Burgess | | | | | | | 3.2–4.4 | ..... | 904 | ..... | Mean of 4 series. |
| 5 | Guillet and Portevin | Unweighable | Nil | Nil | 0.025 | Nil | As deposited | ..... | 937 | 902 | 35 | As = 0.011. |
| 6 | Guillet and Portevin | | | | | | After heating to 900° for 2 hours | ..... | 932 | 902 | 30 | |
| 7 | Carpenter | 0.008 | 0.014 | 0.009 | 0.002 | Trace | Specimen 1: 4th heating | 2 to 9 | 950 | 882 | 68 | |
| 8 | Carpenter | | | | | | Specimen 1: 5th heating | 2 to 9 | 954 | 886 | 68 | |
| 9 | Carpenter | | | | | | Specimen 1: 6th heating | 2 to 9 | 962 | 882 | 80 | |
| 10 | Carpenter | | | | | | Specimen 2: 4th heating | 2 to 9 | 927 | 874 | 53 | |
| 11 | Carpenter | | | | | | Specimen 2: 5th heating | 2 to 9 | 936 | 874 | 62 | |
| 12 | Carpenter | | | | | | Specimen 2: 6th heating | 2 to 9 | 924 | 870 | 54 | |

NOTE TO TABLE VIII.—**1** to **4**. G. K. Burgess, private communication, August 10, 1911.

**5** to **6**. Guillet and Portevin, *Comptes Rendus*, 1913, vol. clvi., p. 702. A Saladin-Le Chatelier apparatus was used.

**7** to **12**. Carpenter, Advance Proof, *Journal Iron and Steel Institute*, May, 1913. The rate of heating varied from 2 sec. per degree at the lowest temperatures to 9 sec. per degree at the highest temperatures, and the rate of cooling from 3 sec at the highest temperatures to 8 sec. at the lowest temperatures.

The specimens were cathode sheets of extremely pure electrolytic iron, heating and cooling curves of which were taken repeatedly *in vacuo*. The results reached in the first three heatings of each specimen are not included here, because they were much affected by the hydrogen occluded in the iron as deposited.

uneffaced solidificational segregation of carbon. But if the s
inflection which Heyn himself puts in the upper part of his l
preserved, this objection to it falls to the ground.

The Ac3 determinations of Table VIII. agree in[31] a genera
with the late results of Broniewski, who found a break in the el
resistance curve of "pure iron" at 950°. His dilatation resul
less easily interpreted, showing as they do a maximum at about
then a sharp contraction to about 975°, and then a re-expansi
about the previous rate.

§ 50. *Summary.*—1. Industrial A3 is probably lower tha
end of Ac3, but above the beginning of Ar3 of the thermal me
(§ 27, p. 1101.)

2. Experiments are needed to show whether the best
refining temperature is Ae3 or some temperature a little abov
little below it. (§ 28, p. 1101.)

3. Manganese lowers the thermal Ar1 and Ar3, at a rate
for Ar1 is in most cases between 24 and 50° per 1 per ce
manganese. (§ 29, p. 1102.)

4. Its effect on Ac1 and Ac3 varies in sign from case to ca
casionally lowering them but often raising them slightly.
pp. 1105 to 1107.)

5. From 4 it is inferred that manganese probably lowers Ae
Ae3, but by a degree so small as often to be masked by its in
ing the lag. (§ 29, p. 1107.)

6. The present evidence is insufficient to show whether m
nese lessens the eutectoid carbon content. (§ 29, p. 1107.)

7. An explanation of the effects of phosphorus on the str
is attempted. (§ 30, p. 1109.)

8. Lag tends to bring the observed Ar points below an
observed Ac points above the Ae points. (§ 31, p. 1112.)

9. Local lowering of the carbon content through surfac
carburization and uneffaced segregation, whether solidificatio
transformational, may raise the observed Ac3 and Ar3 points.
p. 1115.)

10. Evidence is given tending to show that the raising ef
this uneffaced segregation may be material. (§§ 34, 35, pp. 11
1119.)

11. The observed shortage of recognizable ferrite in spec
quenched within the transformation range is interpreted as in
ing that the observed A3 is above Ae3 in these cases. (§§
pp. 1120 and 1123.)

[31] *Comptes Rendus*, 1913, vol. 156, p. 699.

precipitation of ferrite or cementite during the quench-
is unlikely to cause important error in determining the
e of A3 micrographically. (§ 38, p. 1123.)
ure to recognize the first or last traces of precipitated or
ed ferrite may bring the position of Ac3 and Ar3 observed
cally below Ae3. (§39, p. 1124.)
chief data touching A3 are discussed. All but three
ound reasonably consistent. (§§ 40 to 45, pp. 1125 to 1132.)
veighted average of the more important determinations
ormulæ:
17° — 306 × C for *GO*, and
20° — 105.5 × C for *OS*.
percentage of carbon, and
mperature in degrees Centigrade, for Ae3.
agrees fairly with the late determinations of A3 in nearly
pure iron. It implies the following points:

| | Carbon. | Temperature. |
|---|---|---|
| G..... | 0 | 917° |
| O..... | 0.483 | 769° |
| S..... | 0.9 | 725° |

IG. 14.—As explained in § 41, certain points in the line A3 of certain
e been omitted, and certain other have been lowered, as indicated in the
e:

| Observer. | Points Omitted. Carbon Content. | Points Lowered. Carbon Content. | Points Raised. Carbon Content. |
|---|---|---|---|
| mond, Ac3 ..... | 0.16 | ..... | |
| eyn, Ar3..... | 0.75 | 0.08, 0.20, 0.39, 0.50 | |
| mond, Ar3..... | ..... | ..... | 0.08 |
| rpenter and Keeling, Ar3., | 0.12 | 0.01, 0.02, 0.05, 0.16, 0.24 0.38, 0.47, 0.53 | |
| berts-Austen, Ar3..... | 0.22 | ..... | |

hough Carpenter and Keeling do not give any temperature as the beginning
eir steels of 0.61 and 0.81 per cent. of carbon, I have ventured to select a
cooling curve of each of these steels at which the inflection seems to me so
orm valid evidence that Ae3 lies at least as high as this. It is some justifi-
ese points, though selected from the cooling curves without knowing how
t in with the points selected for their other steels, actually form a fairly
ve with them.
l data of Charpy and Grenet are not included in this figure, because their
sults appear to be much better evidence.
In this series. as indicated in § 45, p. 1129, what was actually determined
xact temperature of A3 but a gap of 10° or 20° within which A3 lies, though
the intersection of the curves obtained points to a definite temperature within
A3. To meet this situation Fig. 14 gives for each of the steels of this series
covering this gap, and wherever possible indicates in addition by a special
t position of A3 as inferred from this intersection.

*Legend for Fig.* 14 *and Table Showing Actual Points Plotted in*

◇ SERIES 1. OSMOND, thermal end of Ac3 (corrected, from curves).

| | | |
|---|---|---|
| Carbon, | 0.08 | (0.16) |
| Temperature, | 918 | (945) |

⊡ SERIES 2. GOERENS AND MEYER, micrographic (Ar3).

| | | | | | | |
|---|---|---|---|---|---|---|
| Carbon, | 0.16 | 0.30 | 0.47 | 0.54 | 0.70 | 0.78 |
| Temperature, | 905 | 895 | 885<br>875 | 855 | 785<br>775 | 795<br>785<br>765 |

□ SERIES 3. HEYN, thermal beginning of Ar3 (from curves).

| | | | | | | | |
|---|---|---|---|---|---|---|---|
| Carbon, | 0.08 | 0.17 | 0.20 | 0.39 | 0.50 | (0.75) | 0.95 |
| Temperature, | 905 | 890 | 870 | 800 | 785 | (7.55) | 707 |

+ SERIES 4. BURGESS AND CROWE, thermal end of Ac3.

| | | | |
|---|---|---|---|
| Carbon, | 0.03 | 0.215 | 0.40 |
| Temperature, | 916 | 865 | 828 |

◇ SERIES 5. OSMOND, thermal beginning of Ar3 (corrected, from curves and t

| | | | | |
|---|---|---|---|---|
| Carbon, | 0.08 | 0.16 | 0.29 | 0.57 |
| Temperature, | 890 | 878 | 810 | 780 |

*HLM* SERIES 6. HOWE AND LEVY, micrographic (Ac3).

| | | | |
|---|---|---|---|
| Carbon, | 0 | 0.495 | 0.9 |
| Temperature, | 912 | 768 | 725 |

△ SERIES 7. CARPENTER AND KEELING, thermal beginning of Ar3 (from cu

| | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Carbon, | 0.01 | 0.02 | 0.05 | (0.12) | 0.16 | 0.24 | 0.38 | 0.47 | 0.53 | 0.61 | 0.81 |
| Temperature, | 895 | 892 | 900 | (885) | 850 | 815 | 765 | 757 | 755 | 740 | 72 |

B ꓭ SERIES 8. BOUDOUARD, electric resistance.

| | | | | | |
|---|---|---|---|---|---|
| Carbon, Heating (B) | 0.205 | 0.493 | Cooling (ꓭ) | 0.205 | 0.493 |
| Temperature, | 835 | 825?<br>792 | | 835 | 730 |

○ SERIES 9. CHARPY AND GRENET, dilatational, cooling (Ar3).

| | | | | | |
|---|---|---|---|---|---|
| Carbon, | 0.03 | 0.07 | 0.15 | 0.20 | 0.25 |
| Temperature, | 912 | 885 | 860 | 833 | 808 |

▽ SERIES 10. ROBERTS-AUSTEN, thermal, maximum of Ar3 (from diagram).

| | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Carbon, | 0.08 | 0.09 | 0.10 | 0.11 | 0.16 | 0.165 | (0.22) | 0.30 | 0.34 | 0.54 | 0.6 |
| Temperature, | 882 | 860 | 877 | 860 | 835 | 790 | (768) | 790 | 768 | 730 | 7 |

× SERIES 11. BURGESS AND CROWE, thermal beginning of Ar3.

| | | | | |
|---|---|---|---|---|
| Carbon, | 0.03 | 0.215 | 0.40 | 0.92 |
| Temperature, | 889 | 820 | 754 | 707 |

SERIES 12. MEUTHEN, calorimetric, heating (Ac3).

| | | | | | | | |
|---|---|---|---|---|---|---|---|
| Carbon, | 0.06 | 0.13 | 0.32 | 0.54 | 0.63 | 0.80 | 0.90 |
| Temperature gap, | 900<br>880 | 890<br>870 | 770<br>750 | 740<br>730 | 740<br>730 | 720<br>700 | 720<br>700 |

Temperature of intersection, 767 738 738

*AAA* AVERAGE LINE.

| | | | |
|---|---|---|---|
| Carbon, | 0 | 0.483 | 0.9 |
| Temperature, | 917 | 769 | 725 |

○ In four cases certain of these symbols are enclosed in large circles to indica the observations represented are ignored in calculating the average line A

E

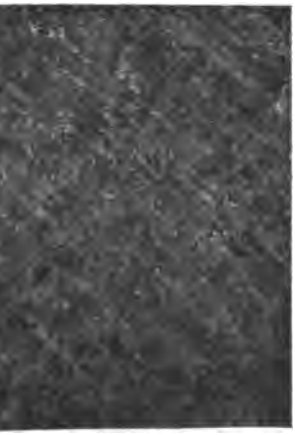
F

1000° —> 830°.

15

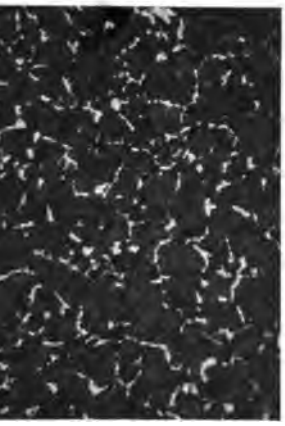
RECIPITATION.

quenched from 830° after
1,000° and then holding
830°.



E

A



A

D

810°

0

775°

0

D

> <

About 10 mm.

>

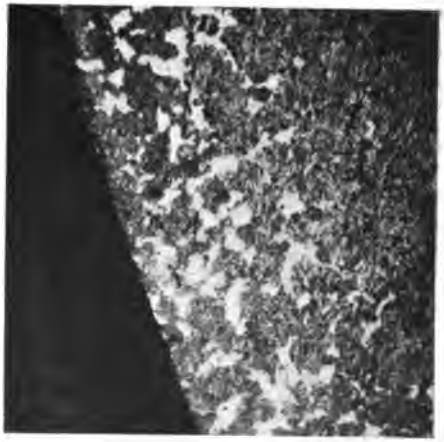
G

H

Row 13

× 45

× 45



× 45

× 45

Centre

Edge

eated at 850° for 30 min., and quenched, shows much decarburization at edges.

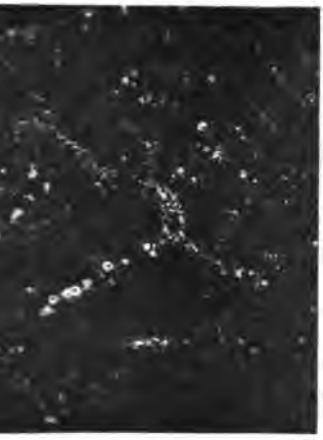
G

H

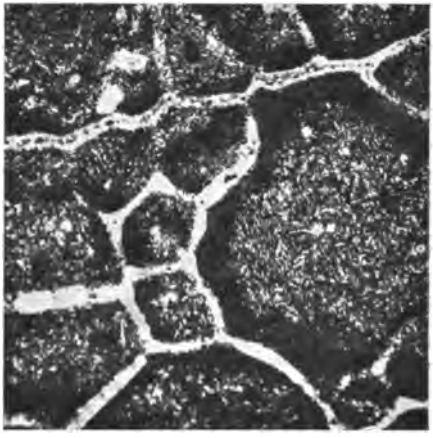
E

F

950°



870°. Interior. troostitized



E

F

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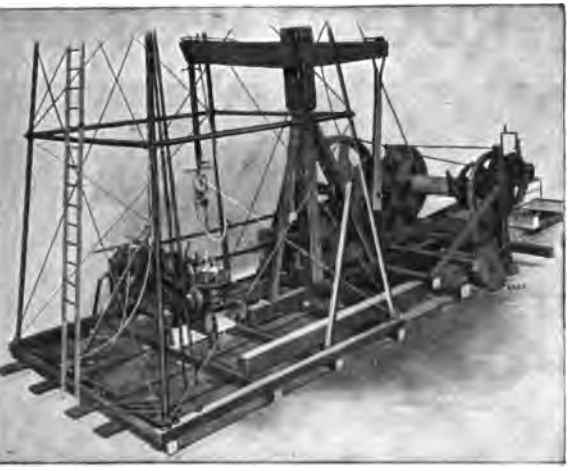

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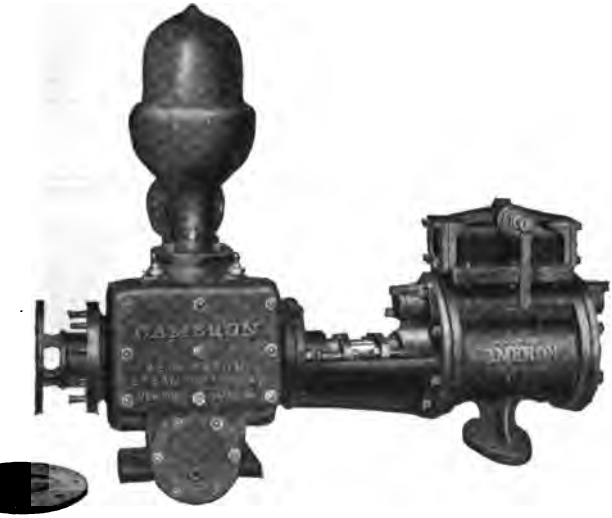

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| .093 | 2.362 | 8 |
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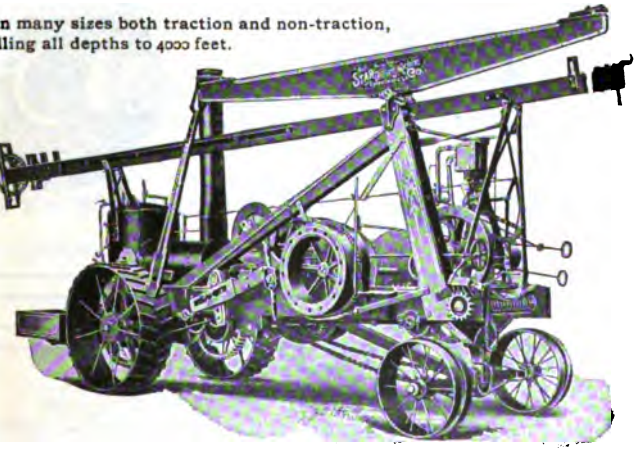

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**PROPOSALS FOR MEMBERSHIP.** A blank proposal for bership is included in Year Book and Bulletin. In the event of not being able to secure one, applications for membership can be made out on any blank pi paper, provided they include the class to which the candidate is pro whether member, associate or junior member; the signature of three bers or associates (junior members must also have the endorsem two of their instructors); a brief history of the candidate, includin and place of birth, his education and technical record, and finally a statement by the applicant that he desires to become a member.

ERICAN INSTITUTE OF MINING ENGINEERS
29 West 39th Street, New York, N. Y.

# PROPOSAL FOR MEMBERSHIP

_____
(Name in Full)

*ion*_____

_____

*proposed by the undersigned, as a*_____

_____

*nerican Institute of Mining Engineers.*

_____
_____
_____
Signatures of three Members or Associates.

*irth*_____ *Year of birth*_____

*, general and technical, when, where and how acquired, with degrees, if any.*

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*Record of experience. Briefly, the past and present em*
*with names of employers, companies and associates. (Pro*
*names of companies, etc., should be written without abbr*

| *Dates* | |
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*Present position*_____

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*Dated*_____ *191*

## EXTRACTS FROM THE CONSTITUTION.

### ARTICLE II.—MEMBERS.

SEC. 1. The membership of the Institute shall comprise four classes, namely: 1. M
orary Members; 3. Associates; 4. Junior Members. * * *

SEC. 2. The following classes of persons shall be eligible for membership in the Insti
Members, all professional mining engineers, geologists, metallurgists, or chemists,
actively engaged in mining and metallurgical engineering, geology, or chemistry; a
persons desirous of being connected with the Institute who in the opinion of the Bo
are suitable.

As Junior Members, all students in good standing in engineering schools who have
degrees and who are nominated by at least two of their instructors. * * *

Every candidate for election as a Member, Associate, or Junior Member must be prop
by at least three Members or Associates, must be approved by the Committee on Mem
scribed in the By-Laws, and must be elected by the Board of Directors.

JULY 1913

# Bulletin of the ıerican Institute of ining Engineers.

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JOURNALS AND OTHERS DESIRING TO REPUBLISH ARTICLES CONTAINED IN THIS BULLETIN SHOULD APPLY FOR PERMISSION TO THE EDITOR, AT 29 WEST 39TH STREET, NEW YORK, N. Y.

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No. 79 JULY 1913

PUBLISHED MONTHLY
BY THE AMERICAN INSTITUTE OF MINING ENGINEERS
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Subscription (including postage), $10 per annum; to members of the Institute, public libraries, educational institutions and technical societies, $5 per annum. Single copies (including postage), $1 each; to members of the Institute, public libraries, etc., 50 cents each.

Entered as second class matter, October 16, 1911, at the post office at Philadelphia, Pa., under the Act of March 3, 1879.

**Advantages to Members of Our Advertising Section.**—It is our intention to make the advertising section of the *Bulletin* as comprehensive as possible, so that members can turn to it for complete information as to where to buy the best mining and metallurgical supplies at the lowest prices. No firms or supplies of dubious reputation are advertised in the *Bulletin*. If members, in buying any material, will refer to the advertisement of that material in the *Bulletin*, or if they do not find it advertised in the *Bulletin*, will, when ordering, ask the manufacturers why the goods were not advertised in the Institute publication, it will be of material benefit to all of us, because the chief reason why advertisers are unable to trace direct results is because purchasers do not mention the *Bulletin* when writing. Finally, if members, when in need of anything whatsoever, will write to the Institute headquarters, an effort will be made to send them a complete list of manufacturers able to supply the desired materials of good quality. The furnishing of these lists will be a benefit to our Advertising Department, as well as to the members. We need scarcely call attention to the advantages obtained by the members from the money which advertising brings in, because all this money is returned directly to the members in one form or another.

---

**Butte Meeting.**—The Secretary's office has already received advices from about 200 members who expect to attend the Butte meeting, many of them being accompanied by members of their family or other guests. The activities of the Committee on Precious and Base Metals have brought out a great many papers of extraordinary value. It has been reported by members of the Committee on Papers and Publications, that two or three of the monographs presented are the most important publications on their particular subjects that have ever been written. The assurance of attendance of a representative body of members; the publishing and distributing of all papers in advance, accompanied by a vigorous campaign to have them actively discussed; the opening to members of the Montana mines and metallurgical works to an extent never before realized, should assure every one that a visit to Montana from August 16 to 21, inclusive, will be well repaid. Members are urged to carry their Membership Identification Cards to Butte with them, as these will be of service in connection with the visits.

**Nominations for Officers.**—The Committee appointed
of Directors to nominate officers for the Institute for the yea
that every member participate in the work by offering sug
proposing names for the various offices, thereby obtaining t
that can be recommended.

The officers to be elected at the annual meeting in Februa

One officer, known as Director and President.
Two officers, known as Director and Vice-President.
Five officers, known as Director.

The attention of members is called to Articles V and VII
tution, and to By-Law XIII, which reads as follows:

"The Geographical districts to be considered by the Committee on N
be as follows, until otherwise ordered by the Board.
District No. 1. New England, New York, and New Jersey, excepting
and district, which is provided for in the Constitution. [N. B.—New Y
trict is designated District 0.]
District No. 2. Pennsylvania.
District No. 3. Ohio, Indiana, Illinois, Iowa, and Missouri.
District No. 4. Minnesota, Wisconsin, and Michigan.
District No. 5. Montana, North and South Dakota, Wyoming, Ne
Washington, Oregon, Idaho, and Alaska.
District No. 6. California and Nevada.
District No. 7. Utah, Colorado, Arizona, and New Mexico.
District No. 8. Louisiana and Texas.
District No. 9. Other Southern States and District of Columbia.
District No. 10. Mexico.
District No. 11. Canada."

An excerpt from Article VII of the Constitution reads as

"In making such selections [namely, the 8 Directors to be elected],
Committee shall, so far as practicable, distribute the representation on
graphically, so that seven members shall be residents of the district incl
City and the territory within a radius of fifty miles of the headquarters
and one member a resident of each of the geographical districts en
By-Laws."

The officers of the Institute whose terms expire are as foll
President, Charles F. Rand (not eligible for re-election).
Past President, Charles Kirchhoff.
Vice-Presidents, Karl Eilers, District 0; Waldemar Lindgr
Directors, James Douglas, District 0; James Gayley, Dist
R. Ledoux, District 0; Charles W. Merrill, District 6; an
Robinson, District 3.

It is hoped that the members of the Institute will help the
the utmost by making suggestions and proposals as req
which should be sent to the Chairman of the Nominatin
John Hays Hammond, 71 Broadway, New York, N. Y.

It is the intention of some of the members of the Nominati
to attend the meeting at Butte. This will enable visiting
express their views in person.

JOHN HAYS HA
*Chairman, Nominatin*

## READY FOR DISTRIBUTION.

### EMMONS VOLUME ON ORE-DEPOSITS.

*Ore-Deposits—a continuation of the "Posepny" Volume.*

Comprising papers descriptive of ore-deposits and discussions of their origin, edited, with an introduction, by Dr. S. F. Emmons. The volume contains also a Biographical Notice of Dr. Emmons by his associate and friend, Dr. George F. Becker, and a comprehensive Bibliographical Index of the Science of Ore-Deposits, prepared by Prof. John D. Irving, of the Sheffield Scientific School of Yale University. Dr. Emmons had finished his editorial work and written his Introduction before his lamented death in 1910; and the Volume contains his last words upon the subject to which he had given the work of his life, and on which he was justly regarded as the foremost authority.

*Contents.*

Genesis of Certain Ore-Deposits. By S. F. EMMONS.
Structural Relations of Ore-Deposits. By S. F. EMMONS.
Geological Distribution of the Useful Metals in the United States. By S. F. EMMONS. Discussion, by JOHN A. CHURCH, ARTHUR WINSLOW, S. F. EMMONS, and WILLIAM HAMILTON MERRITT.
Torsional Theory of Joints. By GEORGE F. BECKER. Discussion, by H. M. HOWE, R. W. RAYMOND, C. R. BOYD, and GEORGE F. BECKER.
Allotropism of Gold. By HENRY LOUIS.
Superficial Alteration of Ore-Deposits. By R. A. F. PENROSE, JR.
Some Mines of Rosita and Silver Cliff, Colorado. By S. F. EMMONS.
Genesis of Certain Auriferous Lodes. By JOHN R. DON. Discussion, by JOSEPH LE CONTE, S. F. EMMONS, G. F. BECKER, ARTHUR WINSLOW, W. P. BLAKE, and J. R. DON.
Influence of Country-Rock on Mineral Veins. By WALTER HARVEY WEED.
Igneous Rocks and Circulating Waters as Factors in Ore-Deposition. By J. F. KEMP.
Consideration of Igneous Rocks and Their Segregation or Differentiation as Related to the Occurrence of Ores. By J. E. SPURR. Discussion, by A. N. WINCHELL.
Chemistry of Ore-Deposition. By WALTER P. JENNEY. Discussion, by JOHN A. CHURCH.
Ore-Deposits near Igneous Contacts. By WALTER HARVEY WEED. Discussion, By W. L. AUSTIN.
Ore-Deposition and Vein-Enrichment by Ascending Hot Waters. By WALTER HARVEY WEED.
Basaltic Zones as Guides to Ore-Deposits in the Cripple Creek District, Colorado. By E. A. STEVENS.
Geological Features of the Gold-Production of North America. By W. LINDGREN. Discussion, by W. G. MILLER and W. L. AUSTIN.
Osmosis as a Factor in Ore-Formation. By HALBERT POWERS GILLETTE.
Ore-Deposits of Sudbury, Ontario. By CHARLES W. DICKSON.
Genesis of the Copper-Deposits of Clifton-Morenci, Arizona. By W. LINDGREN.
Copper-Deposits at San Jose, Tamaulipas, Mexico. By J. F. KEMP.
Magmatic Origin of Vein-Forming Waters in Southeastern Alaska. By A. C. SPENCER.
Genetic Relations of the Western Nevada Ores. By J. E. SPURR.
Are the Quartz Veins of Silver Peak, Nevada, the Result of Magmatic Segregation? By J. B. HASTINGS.
Occurrence of Stibnite at Steamboat Springs, Nevada. By W. LINDGREN.
Summary of Lake Superior Geology with Special Reference to Recent Studies of the Iron-Bearing Series. By C. K. LEITH.
Geological Relations of the Scandinavian Iron-Ores. By H. SJÖGREN.
Formation and Enrichment of Ore-Bearing Veins. (With Supplementary Paper.) By GEORGE J. BANCROFT.
Distribution of the Elements in Igneous Rocks. By H. S. WASHINGTON.
Agency of Manganese in the Superficial Alteration and Secondary Enrichment of Gold-Deposits of the United States. By W. H. EMMONS.
Cognate Papers.
Bibliography of the Science of Ore-Deposits. By J. D. IRVING, H. D. SMITH, and H. G. FERGUSON.

The volume contains 1002 pages. Price, bound in cloth, $5; in half morocco, $6. Both the Emmons and the Posepny Volumes on Ore-Deposits, bound in cloth, $8; in half morocco, $10.

## THE INSTITUTE FORUM.

*Discussion of Papers.*

Meetings of the Institute offer opportunity for social acquaintance and exchange of ideas, for the presentation of papers, and for discussions thereon; also incidentally for viewing under especially favorable conditions mining operations and industrial establishments, and for hospitalities which demonstrate the brotherhood due to kindred interests.

Placing sociability prominent is not to be assumed as relegating other features of meetings to subordinate positions; but during about 40 years of membership in the Institute, the value of acquaintances formed (often ripening into close friendship), and the interchange of data, have frequently demonstrated that the social feature of meetings is of great and lasting value to the membership.

Of a roll embracing 4,011 members, resident in all parts of our country. and scattered in various portions of the world, only a small fraction are able to attend meetings, even if the localities selected are alternately widely separated. Hence, the return which most members receive for their annual assessment of dues is limited to the publications issued, and the credit of being recognized as a member of a great and worthy organization. The greater the value of the contents of our publication, the fuller the return which goes to members.

While the Institute disavows responsibility for views or statements made in papers which appear in its *Transactions*, the fact that these contributions, after passing proper censorship, are accepted by the Institute and spread before the world may properly be received by the writers as endorsements of their (the writers') ability to present a specific technical or industrial proposition worthy of being carefully read.

But the mere presentation and publication of contributed papers falls short of fulfilling one important feature of our meetings. If the purpose is confined to having the writers' views, discoveries, theories, or methods published, this can be obtained through the technical or trade journals and magazines.

The true value of a paper presented at a meeting of the Institute may often be gauged by the discussion which it encourages, although there are contributions of decided value which do not especially invite discussion. However, most of the papers offered contain statements apt to encourage criticism or elaboration by others whose experience or knowledge, or both, assist in combating or endorsing statements made.

The writer of a paper may make statements, prophesy results, or draw conclusions, the correctness of which may escape the editorial review, but may not pass the censorship of 4,011 members, or even be unchallenged by the relatively small attendance at a meeting. On the other hand, discussion often accentuates the statements of a writer, endorses his conclusions, and adds new data germane to the subject treated of in his paper.

Extemporaneous oral discussion following the presentation of a paper enlivens a meeting, and often brings to his feet a participant whose modesty restrains him from taking part in written discussions prepared in advance or subsequent to a meeting.

The decadence of discussions may be traceable to an effort to crowd too many papers into a few sessions, and to the policy of some large corporations of discouraging their representatives from preparing papers or participating in their discussion, a policy apparently unfair to the employee and surely of small advantage to the employer, for experience has shown that some of the greatest advances in practice or progress are the result of

frank interchange of experiences or opinions. The value of statements is brought out by arguments pro and con, and expecting that papers presented will be discussed, writers are encouraged to use special care in their preparation, and a knowledge that discussions will be a prominent feature of meetings will attract increased attendance. JOHN BIRKINBINE.

---

## PERSONAL.

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members who registered at Institute headquarters during June:

**Ralph H. Sweetser**, Columbus, Ohio.
**Etienne A. Ritter**, Colorado Springs Colo.
**Dwight E. Woodbridge**, Duluth, Minn.
**Arthur Chippendale**, Mexico City, Mex.
**E. T. Miller**, Louisville, Ky.
**Jesse Scobey**, Denver, Colo.
**Charles Rhodes**, Auckland, N. Z.
**John W. Finch**, Denver, Colo.
**O. M. Bilharz**, Rivermines, Mo.
**Joseph A. Holmes**, Washington, D. C.
**Francis Church Lincoln**, Urbana, Ill.
**William E. Saunders**, Philadelphia, Pa.
**E. P. Ross**, Johnson City, Tenn.
**H. M. Boylston**, Cambridge, Mass.
**Edward Keller**, Perth Amboy, N. J.
**F. G. Cottrell**, San Francisco, Cal.

**John W. Finch** has received the Honorary Degree of Doctor of Science from Colgate University.

**Prof. Alfred C. Lane** received the Honorary Degree of Doctor of Science from Tufts College.

**Reginald E. Hore** has joined the Editorial Staff of the *Canadian Mining Journal* at Toronto, Canada.

**W. J. Hamilton** is General Manager of the Cerro de Pasco Copper Co.

**A. B. Willmott**, of Toronto, has become Consulting Engineer with the Lake Superior Corporation. He will also continue practice as a consulting mining engineer.

**A. M. Swartley, Dr. Ulysses S. Grant, Prof. H. M. Parks, Dr. A. N. Winchell**, and **Prof. Solon Shedd** are members of the technical staff of the newly-established Oregon Bureau of Mines.

**A. J. Dull** has been elected President of the Pulaski Iron Co.

**E. H. Coxe** is General Manager of the La Follette Coal and Iron Co.

**Corey C. Brayton** will be in Nome, Alaska, till late in the fall.

**G. A. Reinhardt** is metallurgist of the Youngstown Sheet & Tube Co.

**Dr. Joseph A. Holmes** delivered the Class Day address at the Michigan College of Mines.

**John M. Sherrerd** has become General Manager of Sales for the Titanium Alloy Manufacturing Co., Niagara Falls, N. Y.

**Albert Broden** has been decorated by the King of Sweden a Knight of the Order of Vasa, in recognition of his part in strengthening the friendly ties between the United States and the Kingdom of Sweden.

**J. B. Risque** has become General Manager of the Tennessee Copper Co.

**Richard Marsh** has become Superintendent of the Yankee Girl gold mine, at Ymir, B. C.

**Ivan DeLashmutt** is Manager for the Standard Silver-Lead Mining Co.

**F. H. Morley** was married on May 5 at Bournemouth, England, to Mrs. Davis. Mr. and Mrs. Morley will reside at Colorado Springs, Colo.

**Hallet R. Robbins** has become Consulting Engineer for the Chewelah Copper King Mining Co.

**Jose Ramon Villalon**, of Havana, has been appointed Secretary of Public Works in the Cabinet of the President of Cuba.

## ENGINEERS AVAILABLE.

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons.)

An experienced gold and silver metallurgist specializin is desirous of obtaining a position in America. Speaks M fluently.

A member, a technical graduate, with four years' experie work and general construction, two years' on mine examin logical reconnaissance, and two a half years' drilling explor work and petroleum, together with prospecting and geolog tion, desires a position as a mining engineer or mining geol

An experienced mine superintendent, manager, or field en for engagement. He has had 12 years' experience as supe manager of large mines in the United States and Mexico, in mine examination.

A member of the Institute with several years' experienc the Lake Superior copper district, is open for engagement.

A member, at present in the Southeast, desires a positi superintendent or assistant manager with a mining compar Washington or Alaska. Experienced assayer and engineer

An expert mining and mechanical engineer of 40 years open for engagement.

A technical graduate, 33 years of age, with 11 years' min as engineer, foreman, superintendent and manager of deve would like to make a change.

A technical graduate of five years' experience undergrou an engagement.

A member with technical education who has had experi chanical lines at iron and steel works and as salesman for electrical companies desires a position as representative West of a machinery or electrical supply house.

## BOARD OF DIRECTORS.

*Meeting, June* 26, 1913.—On the petition of 29 members near San Francisco, Cal., the San Francisco Local Section w

The territory of the St. Louis Local Section was establis the following States and parts thereof:

All the State of Missouri;
That part of the State of Illinois south of and including the city of Ur
That part of the State of Indiana south of and including the city of In
That part of the State of Kentucky west of and including the city of F
That part of the State of Tennessee west of and including the city of N
That part of the State of Arkansas north of and including the city of I

The By-Laws of the Colorado Local Section were approve

It was voted that those volumes of the *Transactions* pub any year shall be sent to members who pay dues for tha that members who join the Institute after the date upon w is issued shall receive only the volumes issued subsequentl

J. S. Unger, Chief of the Research Department of the Carn was added to the Iron and Steel Committee.

A Committee on Mining Frauds was established.

## IRON AND STEEL COMMITTEE.

CHARLES KIRCHHOFF, *Chairman.*

ALBERT SAUVEUR, *Vice-Chairman.*

A. A. STEVENSON, *Vice-Chairman.*

ERBERT M. BOYLSTON, *Secretary*, Abbot Bldg., Cambridge, Mass.

| | | |
|---|---|---|
| kinbine, | J. Esrey Johnson, Jr., | Felix A. Vogel, |
| H. Blauvelt, | William Kelly, | Leonard Waldo, |
| yley, | Richard Moldenke, | William R. Walker, |
| Hibbard, | Joseph W. Richards, | William R. Webster. |
| Howe, | E. Gybbon Spilsbury, | Frederick W. Wood. |
| . Hunt, | | |

evenson is appointed an additional Vice-Chairman of the He will serve as Acting Chairman during the absence abroad Sauveur.

neeting of the Institute which is to be held in New York City, 17, 1913, the following papers are now assured:

(Those marked * are in the Editor's hands.)

Howe, Equilibrium Temperature for Ae1 in Carbon Steel.
Howe, The Divorcing of the Eutectoid in Meteorites.
Howe and A. G. Levy, Determination of the Position of $Ae_3$ in Carbon-Iron vs.
Burgess, J. J. Crowe, and H. S. Rawdon, Thermal and Microscopical Exami-n of Professor Howe's Standard Commercial Steels.
Howe, Discussion of the Existing Data as to the Position of $Ae_3$.
Hall, Shock Tests of Cast Steel.
Miller, Jr., New Design of Regenerators for Open-Hearth Furnaces.
tz, The Scoria Process for the Manufacture of Fine-Ore Briquettes.
Hall, The Life of Crucible Steel Furnaces.
A. Vogel, Briquetting.
Johnson, Jr., The Influence on Quality of Cast Iron Exerted by Oxygen, ogen, and Some Other Elements.
Abbott, Influence of Alloying Elements in Carburization of Steel.
Emmons, Some Phase of the Practical Treatment of Tool Steel.
d Advantages of Briquettes in Blast Furnace Practice. (Vogel?).
m A. Forbes, Blast Furnace Gas Cleaning.
Clevenger and B. Ray, The Influence of Copper upon the Physical Proper-of Steel.
Shimer, Over-Oxidation of Steel.
Norris, The Resistance of Steels to Wear in Relation to their Hardness and ile Properties.

on to this there are good prospects of about ten other papers. rs and titles will be given in an early issue of the *Bulletin*.
e being made to secure full discussion of all the papers. To is again urged that authors submit their manuscripts at an n order that they may be printed and distributed to those offer discussions.

nely interesting meeting of the Iron and Steel Committee was Hotel Traymore, Atlantic City, N. J., on June 25, 1913. The entlemen were present: A. A. Stevenson, Herbert M. Boylston, Hunt, Richard Moldenke, William R. Webster, Charles F. Rand, ughton, Edgar Marburg, George C. Stone, E. F. Kenney, C. F. acob Cambier, E. O'C. Acker, A. D. de Pierrefeu, Frank N. . Unger, Guilliaem Aertsen, P. E. Carhart.

At this meeting the matter of subjects for papers and nam might write discussions was thoroughly considered, mar being received in these connections as well as on matters of The spirit of co-operation between the committee and its manifest and the meeting was so enjoyable that it is hoped t meetings from time to time.

Among the subjects considered at this meeting for papers t at the October meeting of the Institute or future meetings a nace Experience with Sintered, Briquetted, and Nodulized Gas Producers; Slagging Producers; Powdered Coal as a Fu of Waste Gases from Open-Hearth Furnace; Use of By-Pro

The relations of the various technical societies were also d cially the relations of the Institute with the American Soci Materials. In response to a suggestion it has been arra members of technical committees from this Society to atte meeting of the Institute. Other means of co-operation were are being considered. The President of the Institute was up with the Board of Directors the question of the appoint mittee on the relations between the various technical so promised to do so.

A program committee, consisting of Messrs. A. A. Stevens bard, J. Esrey Johnson, Jr., E. Gybbon Spilsbury, and H has been appointed to take charge of the program and en the October meeting. A preliminary meeting of this sub- held at Atlantic City, June 25, 1913, and another meeting o

As an evidence of the increase of interest in the Institu and steel men, it may be stated that about 200 men intere iron and steel have been taken into membership durin months.

HERBERT M. B

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## COMMITTEE ON PRECIOUS AND BASE M

CHARLES W. GOODALE, *Chairman.*

L. D. RICKETTS, *Vice-Chairman.*

DARSIE C. BARD, *Secretary.*

Montana State School of Mines, Butte, Mont.

| | | |
|---|---|---|
| Leonard S. Austin, | Samuel S. Fowler, | Willet G. |
| David W. Brunton, | Thomas J. Grier, | Albert F. |
| Theodore B. Comstock, | Hennen Jennings, | George C. |
| Stanley A. Easton, | Edmund B. Kirby, | Benjamin |
| | Charles W. Merrill. | |

The following papers have been already secured and se tary of the Institute:

METALLURGICAL SUBJECTS.

1. Great Falls Converter Practice. By A. E. Wheeler and Milo W.
2. Development of the Basic-lined Converter for Copper Mattes. By
3. Great Falls Flues and Dust Chamber System. By C. W. Goodal inger.
4. Determination of Gases in Smelter Flues; and Notes on the Dete Losses at the Washoe Reduction Works, Anaconda. By E. M.
5. Notes on the Great Falls Electrolytic Plant. By Willis T. Burn
6. The Development of Blast Furnace Construction at the Boston & By J. A. Church, Jr.
7. The Metallography of Refined Copper. By E. S. Bardwell.

termination of Arsenic and Antimony in Converter and Electrolytic Copper.
. E. Brownson.
Trioxide from Flue Dust. By J. O. Elton.
ng the Efficiency of MacDougall Roasters at the Great Falls Smelter of the onda Copper Mining Co. By Frank R. Corwin and Selden S. Rodgers.
pele Plant of the International Smelting & Refining Co. By H. N. Thomson.
ecent American Progress in the Assay of Copper Bullion. By Edward Keller.
l Effects of Blast-Furnace Jackets. By R. P. Roberts.

*Milling, Etc.*

eat Falls System of Concentration for Copper-Bearing Sulphide Ores from , Mont. By Albert E. Wiggin.
ration of Slimes at Anaconda, Mont. By Ralph Hayden.
olution of the Round Table for the Treatment of Metalliferous Slimes. By dore Simons.
ating Copper from Mine Waters in Butte. By J. C. Febles.
aching of Copper Tailings. By Frederick Laist.
aconda Classifier. By Robert Ammon.
tion of Hindered Settling to Hydraulic Classifiers. By E. E. Bardwell.

*Mining Subjects.*

electric Development in Montana. By Max Hebgen.
ng Methods at Butte. By B. H. Dunshee.
d Mine Accounting. By H. T. Van Ells.
umlummon Mine, Marysville, Mont. By C. W. Goodale. Discussion by Sizer and others,
nking Methods at Butte. By Norman Braly.

*Geological Subjects.*

of Butte. By R. H. Sales.
Associations at Butte, Mont. By D. C. Bard and M. H. Gidel.

on to these the following papers have been promised, and it is they will be ready in time for the meeting:

of Precious Metals in Converter Copper. By A. E. Wheeler.
ncentrating Tables. By Arthur Crowfoot.
Generated Compressed Air in Butte Mines. By Bruno Nordberg.
rn Cross Mine. By Paul Billingsley.
ry Smelting with Low-Grade Coal. By C. D. Demond.
g Mill. By Rush J. White.
nprovements in Cyanide Practice. By J. V. N. Dorr.
tricity in Butte Mines. By John Gillie.
ines at Butte. By James I. Bruce.
ing at Ruby, Mont. By Hennen Jennings.
Mine Equipment at Butte. By F. A. Linforth.
apping at Butte. By F. A. Linforth.
s of Mining Geology as Developed by Recent Litigation in the Cœur d'Alênes. Greene.
e District, Idaho. By Ralph Nichols.
Industry of Montana. By W. H. Andrews.
hate Deposits in Western America.
eposits of Montana.

D. C. BARD, *Secretary.*

## BOSTON LOCAL SECTION.

*Executive Committee.*

ROBERT H. RICHARDS, *Chairman.* ALBERT SAUVEUR, V
TIMOTHY W. SPRAGUE. HENRY A. WENT
AUGUSTUS H. EUSTIS, *Secretary,* 131 State St., Bost

The twelfth meeting of the Boston Section was held at t Club, 2 Commonwealth Ave., Boston, on Monday evening fifteen members and three guests being present. The meeti to order by Vice-Chairman Albert Sauveur, who introduce of the evening, **Utley Wedge,** of Ardmore, Pa. Mr. Wedge on the chloridizing roasting of copper ores, which he illus stereopticon, and which will be published in full by the Inst

After Mr. Wedge had closed his paper, which was receiv interest by all the members, the following discussion took pl

**Professor Hofman** said that iron was formerly thought t to the Henderson process, but that it is not absolutely essentia said that, as far as he knew, the presence of some sulphates lute necessity, and that lime ores can be worked satisfacto lime has an effect on the subsequent leaching. He added fuel consumption of the latest Wedge furnace is a result of tion of the combustion gases to the ore. Ten per cent. of fu with the muffle furnace. By direct firing he has been able fuel needed to from 2 to 4 per cent. In the latest types o gases from the upper hearth are not used. There is doubtl perature at which the loss from copper chloride volatiliza serious, but in practice on Rio Tinto ores troublesome losses rienced. The copper apparently does not become volatile a tures where we would expect that it would. There are dou which there would be serious danger from losses from this latest furnaces do not use the high temperatures that were former single-hearth Wedge furnaces.

**Dr. Peters** commented on the difficulties encountered by were trying to chloridize all metals, and Mr. Wedge explai was very expensive. He added: The recovery of the zinc is plished in various works. At least two works are practicing an absolute stop between the two parts of the furnace. A or trap, the gases are taken to the chimney; below it they a through the furnace. There are two kinds of traps in use; and one is a mechanical valve. No air should be admit valve. There is no appreciable difference in the amount o the reverberatory and in the muffle furnace. The usual grind the salt and ore together. In some plants an attem have the furnace mix the ingredients, but I prefer to grind the ore. Poor mixing produces an excess of salt in so which makes sticky patches in the mass when heated. Ma ride can be used, and less of it is needed than of sodium c cost of giving material a chloridizing roast in this latest fu 35 cents per ton, not including the cost of the salt.

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F. A. EUSTIS, *Secre*

## SPOKANE LOCAL SECTION.

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GLENVILLE A. COLLINS, *Vice-Chairman.*
FRANCIS A. THOMSON.
JAMES A. MCCARTHY.
LYNDON K. ARMSTRONG, *Secretary-Treasurer.*

enth meeting of the Spokane Local Section was held in Rossland, a side trip to Trail to visit the smelter and refinery of the Con- Mining & Smelting Co. of Canada, Limited. The meeting was ly with the Western Branch of the Canadian Mining Institute, those present being members of both societies.

of the members left Spokane on the morning of May 22, arriving ng at Rossland, where they were met by members of the Western . M. I., and, after dinner, a joint session of the two sections was nt 50 members and visitors being present.

**Purcell,** Chairman of the Western Branch, C, M. I., occupied the E. Jacobs, Secretary of the Western Branch, C. M. I., acted as **Prof. R. S. McCaffery,** Chairman Spokane Local Section, C., occupied a seat beside the Chairman.

e usual greetings and responses **E. Jacobs** read a paper on the Rossland camp, much of which was especially interesting to of the A. I. M. E. from across the international border in the had formerly been financially interested in the camp. Discus- oped the fact that the mines have produced to date $55,000,000; ough formerly considered copper mines, they are now producing relative value in gold; that deep levels show less copper value, ss is more than made up in greater gold contents and deep-level being prosecuted vigorously. **S. S. Fowler** added interest to by relating some personal experiences.

S. McCaffery then read a paper on The Use of Oxygen Helmets ires, by **E. P. Dudley,** of Kellogg, Idaho, being a statement of e in a recent fire in the Bunker Hill & Sullivan mine. The s followed by an exhibition of the Draeger pulmotor by Mr. d Prof. McCaffery.

wler read a paper prepared by **J. Buchanan,** Superintendent rail smelter, entitled, Copper Smelting at the Works of the ted M. & S. Co. of Canada, Ltd., which wasdiscussed by several and non-members. **Mr. Purcell** described the haulage and g system of the Centre Star mine, of which, with the War Eagle, erintendent.

owing morning about 25 persons paid a visit to the mines of the ted Co., entering by the Centre Star shaft and by shaft and tunnel at mine and the War Eagle. A 30-ft. wide stope was a feature nterest, the same ore shoot on levels above being about 700 ft. in On the level visited, however, the length of the shoot had not yet mined, but, so far as explored, the width varies from 10 or 12 Gold values predominate. Passing from the workings of the ar-War Eagle mines to the Le Roi mine, the party was led to a he 800-ft. level which is 40 ft. wide and carries good gold values. bell rang for the hoist the visitors had collectively learned at mportant fact, which was that the mines have not been worked ppear in a very healthy state of youthful development.

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## SPOKANE LOCAL SECTION.

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**Nominations for Officers.**—The Committee appointed b
of Directors to nominate officers for the Institute for the year
that every member participate in the work by offering sugg
proposing names for the various offices, thereby obtaining th
that can be recommended.

The officers to be elected at the annual meeting in February

One officer, known as Director and President.
Two officers, known as Director and Vice-President.
Five officers, known as Director.

The attention of members is called to Articles V and VII of
tution, and to By-Law XIII, which reads as follows:

"The Geographical districts to be considered by the Committee on Nom
be as follows, until otherwise ordered by the Board.
District No. 1. New England, New York, and New Jersey, excepting N
and district, which is provided for in the Constitution. [N. B.—New York
trict is designated District 0.]
District No. 2. Pennsylvania.
District No. 3. Ohio, Indiana, Illinois, Iowa, and Missouri.
District No. 4. Minnesota, Wisconsin, and Michigan.
District No. 5. Montana, North and South Dakota, Wyoming, Nebr
Washington, Oregon, Idaho, and Alaska.
District No. 6. California and Nevada.
District No. 7. Utah, Colorado, Arizona, and New Mexico.
District No. 8. Louisiana and Texas.
District No. 9. Other Southern States and District of Columbia.
District No. 10. Mexico.
District No. 11. Canada."

An excerpt from Article VII of the Constitution reads as fol

"In making such selections [namely, the 8 Directors to be elected], th
Committee shall, so far as practicable, distribute the representation on t
graphically, so that seven members shall be residents of the district includ
City and the territory within a radius of fifty miles of the headquarters o
and one member a resident of each of the geographical districts enum
By-Laws."

The officers of the Institute whose terms expire are as follo
President, Charles F. Rand (not eligible for re-election).
Past President, Charles Kirchhoff.
Vice-Presidents, Karl Eilers, District 0; Waldemar Lindgre
Directors, James Douglas, District 0; James Gayley, Distri
R. Ledoux, District 0; Charles W. Merrill, District 6; and
Robinson, District 3.

It is hoped that the members of the Institute will help the C
the utmost by making suggestions and proposals as requ
which should be sent to the Chairman of the Nominating
John Hays Hammond, 71 Broadway, New York, N. Y.

It is the intention of some of the members of the Nominatin
to attend the meeting at Butte. This will enable visiting
express their views in person.

JOHN HAYS HAM
*Chairman, Nominating*

## READY FOR DISTRIBUTION.

### EMMONS VOLUME ON ORE-DEPOSITS.

*Ore-Deposits—a continuation of the "Posepny" Volume.*

ing papers descriptive of ore-deposits and discussions of their ted, with an introduction, by Dr. S. F. Emmons. The volume lso a Biographical Notice of Dr. Emmons by his associate and . George F. Becker, and a comprehensive Bibliographical Index ence of Ore-Deposits, prepared by Prof. John D. Irving, of the Scientific School of Yale University. Dr. Emmons had finished al work and written his Introduction before his lamented death nd the Volume contains his last words upon the subject to which ven the work of his life, and on which he was justly regarded as ost authority.

*Contents.*

tain Ore-Deposits. By S. F. EMMONS.
ations of Ore-Deposits. By S. F. EMMONS.
tribution of the Useful Metals in the United States. By S. F. EMMONS. Discussion, by HURCH, ARTHUR WINSLOW, S. F. EMMONS, and WILLIAM HAMILTON MERRITT.
ory of Joints. By GEORGE F. BECKER. Discussion, by H. M. HOWE, R. W. RAYMOND, C. R. GEORGE F. BECKER.
Gold. By HENRY LOUIS.
teration of Ore-Deposits. By R. A. F. PENROSE, JR.
Rosita and Silver Cliff, Colorado. By S. F. EMMONS.
tain Auriferous Lodes. By JOHN R. DON. Discussion, by JOSEPH LE CONTE, S. F. EMMONS, ER, ARTHUR WINSLOW, W. P. BLAKE, and J. R. DON.
ountry-Rock on Mineral Veins. By WALTER HARVEY WEED.
and Circulating Waters as Factors in Ore-Deposition. By J. F. KEMP.
of Igneous Rocks and Their Segregation or Differentiation as Related to the Occurrence By J. E. SPURR. Discussion, by A. N. WINCHELL.
Ore-Deposition. By WALTER P. JENNEY. Discussion, by JOHN A. CHURCH.
ear Igneous Contacts. By WALTER HARVEY WEED. Discussion, By W. L. AUSTIN.
and Vein-Enrichment by Ascending Hot Waters. By WALTER HARVEY WEED.
as Guides to Ore-Deposits in the Cripple Creek District, Colorado. By E. A. STEVENS.
atures of the Gold-Production of North America. By W. LINDGREN. Discussion, by W. and W. L. AUSTIN.
actor in Ore-Formation. By HALBERT POWERS GILLETTE.
f Sudbury, Ontario. By CHARLES W. DICKSON.
Copper-Deposits of Clifton-Morenci, Arizona. By W. LINDGREN.
ts at San Jose, Tamaulipas, Mexico. By J. F. KEMP.
gin of Vein-Forming Waters in Southeastern Alaska. By A. C. SPENCER.
ons of the Western Nevada Ores. By J. E. SPURR.
Veins of Silver Peak, Nevada, the Result of Magmatic Segregation? By J. B. HASTINGS.
Stibnite at Steamboat Springs, Nevada. By W. LINDGREN.
ake Superior Geology with Special Reference to Recent Studies of the Iron-Bearing C. K. LEITH.
ations of the Scandinavian Iron-Ores. By H. SJÖGREN.
d Enrichment of Ore-Bearing Veins. (With Supplementary Paper.) By GEORGE J.
the Elements in Igneous Rocks. By H. S. WASHINGTON.
nganese in the Superficial Alteration and Secondary Enrichment of Gold-Deposits of States. By W. H. EMMONS.
s.
of the Science of Ore-Deposits. By J. D. IRVING, H. D. SMITH, and H. G. FERGUSON.

ume contains 1002 pages. Price, bound in cloth, $5; in half $6. Both the Emmons and the Posepny Volumes on Ore- bound in cloth, $8; in half morocco, $10.

## AFFILIATED STUDENT SOCIETIES.

Any society of undergraduates at a technical school, comprising students in any branch of engineering, metallurgy, chemistry, geology, etc., may be recognized by the Board of Directors in its discretion as an Affiliated Student Society. A circular giving details of the plan of affiliation may be obtained on application to the office of the Secretary of the Institute.

The following societies have been placed by authority of the Board of Directors on the above list:

### AFFILIATED STUDENT SOCIETIES.

The Mining Society of the Sheffield Scientific School, Yale University, New Haven, Conn. *President*, Francis S. Winslow; *Secretary*, Roger W. Newberry.

The University of Illinois Student Branch of the American Institute of Mining Engineers, Champaign, Ill. *President*, M. L. Nebel; *Secretary*, Willis Leriche.

The Engineering Society of the University of Nevada, Reno, Nev. *President*, P. E. Raymond; *Secretary*, R. M. Seaton.

The University of Wisconsin Mining Club, Madison, Wis. *President*, Rudolph J. Stengl; *Secretary*, Mack C. Lake.

The Mining and Geological Society of Lehigh University, South Bethlehem, Pa. *President*, Robert C. Mickel; *Secretary*, Edward B. Snyder.

The School of Mines Society of the University of Minnesota, Minneapolis, Minn. *President*, C. A. Walker; *Secretary*, A. C. Bierman.

The Mining Engineering Society of the Massachusetts Institute of Technology. *President*, Ralph E. Wells, Jr.; *Secretary*, Louis W. Currier.

The Student Auxiliary Society of the American Institute of Mining Engineers of the University of Kansas, Lawrence, Kan. *President*, Walter E. Rohrer; *Secretary*, H. R. Brown.

The Associated Miners of the University of Idaho, Moscow, Idaho. *President*, Hallard W. Foester; *Secretary*, Walter P. Scott.

The State College of Washington Mining and Geological Society, Pullman, Wash. *President*, J. V. Quigley; *Secretary*, S. A. Swanson.

The Tejas Technical Society, School of Mines, University of Texas, Austin, Tex. *President*, G. C. Cartwright; *Secretary*, David S. Alley.

The Ohio State University Student Branch of the American Institute of Mining Engineers, Columbus, Ohio. *President*, Hugh B. Lee; *Secretary*, E. P. Elliott.

The Stanford Geology and Mining Society, Stanford University, Cal. *President*, B. E. Parsons; *Secretary*, E. D. Nolan.

The Senior Mining Society of Columbia University, New York, N. Y. *President*, Roger L. Strobel; *Secretary*, Clark G. Mitchell.

Mining Association of the University of California, Berkeley, Cal. *President*, D. M. Drumheller, Jr.; *Secretary*, J. B. Orynski.

Tufts College Chemical Society, Tufts College, Mass. *President*, P. G. Savage; *Secretary*, W. S. Frost.

University of Washington Mining Society, Seattle, Wash. *President*, Oliver P. Searing; *Secretary*, Albert R. Sherman.

Student Branch of the American Institute of Mining Engineers, Iowa State College, Ames, Iowa. *President*, Lyle A. Butler; *Secretary*, A. J. Crawford.

Missouri Mining Association of the Missouri School of Mines, Rolla, Mo. *President*, L. S. Copelin; *Secretary*, L. J. Boucher.

The Pick and Shovel Club of the Case School of Applied Science, Cleveland, Ohio. *President*, M. T. Whelan; *Secretary*, Lloyd A. Collier.

Colorado School of Mines Scientific Society, Golden, Colo. *President*, Alan Kissock; *Secretary*, George Wilfley.

Mining Engineering Society of the University of Arizona, Tucson, Ariz. *President*, J. Gary Lindley; *Secretary*, H. O. Coles.

Kentucky Mining Society, College of Mines and Metallurgy, University of Kentucky, Lexington, Ky. *President*, Oliver W. Smith; *Secretary*, Charles E. Straub.

Mining Society of Pennsylvania State College, State College, Pa. *President*, Ralph E. Kirk; *Secretary*, Willard M. Lindsey.

## LIBRARY.

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS.
AMERICAN SOCIETY OF MECHANICAL ENGINEERS.
AMERICAN INSTITUTE OF MINING ENGINEERS.
UNITED ENGINEERING SOCIETY.

WILLIAM P. CUTTER, Librarian.

ary of the above-named Societies is open from 9 A.M. to 9 P.M. -days, except holidays, from September 1 to June 30, and from P.M. during July and August. The Library contains about umes, including sets of technical periodicals and the publica- entific and technical societies.

of the Institute, with few exceptions, are forced to spend a por- ir time in localities isolated from sources of information. To Library can render valuable service through correspondence; esting information will receive special attention. The Library l to furnish references and copies of articles on mining and al subjects; to determine the existence of mining-maps, and to eral information as to the geology and mineral resources of all

munications should be made as definite as possible so that the received may be what is desired and not include collateral ch may not be of interest. The time spent in searching for such matter will be saved, and the information will be sent more nd in more usable shape.

bers of the Institute can be of service to the Library by for- opies of mining-reports, maps privately issued, and similar hich will be classified, indexed, and made available to other Suggestions for additions to the Library, either by purchase solicitation as gifts, will be welcomed. It is hoped that mem- in the city will use the Library freely, and assurance is given areful service will be rendered to them.

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## LIBRARY RESEARCH-WORK.

he year 1912 there were 160 searches made for members and ers of the Founder Societies, and copies of the references have rved for the use of others. This work has been largely based s sent in by mail, from Japan, South Africa, South America, nada, and England, as well as from different parts of the United e library receives 670 technical periodicals which are available e indexes for this special purpose.

## Library Accessions.

June 1 to June 30, 1913.

[Copies of the list of additions to the Libraries of the American Soci
cal Engineers and the American Institute of Electrical Engineers can
application to the Secretary of the American Institute of Mining Engine

ACADEMY OF NATURAL SCIENCES OF PHILADELPHIA. Index to the Scie
the Journal and Proceedings, 1817–1910. Philadelphia, 1912. (Pu

APPARATUS FOR GAS-ANALYSIS LABORATORIES AT COAL MINES. (Tech
14, U. S. Bureau of Mines.) Washington, 1913.

ASSOCIATION DES INGENIEURS SORTIS DE L'ECOLE DE LIÉGE. Lists des
Liége, 1912. (Exchange.)

K. K. MINISTERIUM FÜR OFFENTLICHE ARBEITEN. Statistik des Bergbau
1911. Wien, 1912. (Exchange.)

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Elton W. Wa

Meetings of this committee were held at the Institute headqu
April 18, May 2 and 15, and June 19.

At the meeting of June 19, the Secretary reported that letters
tion to apply for membership in the Institute had been sent t
alumni of the School of Mines of Columbia University and a
State Mine Inspectors and their deputies. He also reported the
of lists of the alumni of the Harvard Engineering Schools, and of
ing Schools at Houghton, Mich.; Butte, Mont.; State College, Pa.
Wash.; Fayetteville, Ark.; Laramie, Wyo.; Montreal, Canada,
cester and Rensselaer Polytechnic Institutes. Arrangements w
to invite all these to join the Institute. It was suggested that
of the Institute resident in Australia be appointed to this Comm
one from Japan. Attention of the members of the Committee w
to the fact that letters requesting others to join the Institute will
in the offices of the Institute if desired. The applications for me
received recently have been as follows: March, 30; April, 62;
June, 54. The next meeting will be held July 17.

THOMAS T. READ, S

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## BACK VOLUMES OF THE TRANSACTIONS

The Board of Directors has authorized the following offers of se
volumes of the *Transactions*, at considerably reduced prices, to
Libraries, and Scientific Societies:

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9 (1881), . . . . . .

## NTERNATIONAL GEOLOGICAL CONGRESS.

her particulars apply to the Secretary, 12th International Geological Congress, emorial Museum, Ottawa, Canada; cable address, Geocong, Ottawa.)

**oal Resources of the World.**—Three quarto volumes of about s each, 8¼ by 10¾ in., and an atlas of about 70 maps in colors, ½ in., bound in heavy paper cover. Edition limited to 3,000 copies. sand copies will be reserved for members of the International l Congress, and the remainder of the edition will be distributed der in which applications are received. Price, $25 per set. Ready 1913.

ook is an inquiry made upon the initiative of the Executive Com- the Twelfth International Geological Congress, with the assist- eological surveys and mining geologists of different countries, l by Morang & Co., Ltd., Toronto, Canada.

*ters During Session at Toronto.*

adquarters will be at the University of Toronto, where there will ch post-office open at all hours and at which registered mail may ed or despatched, money orders sent or cashed, etc. A bank will shed at which money may be exchanged. A typewriting service aintained, also a telephone service and messenger service. Rail- steamship agents will also be in attendance.

urant service providing luncheon in the middle of the day will ed.

*Special Dates.*

, August 1.....Special day at Ottawa.
ay, August 2.....Special day at Montreal.
ay, August 7.....Opening day of Session at Toronto.
ay, August 14...Last day of Session at Toronto. Excursions C. 1 and C. 2 start.
y, August 26.....Special day at Victoria, British Columbia.

*Privileges.*

fessional members of the Congress and to dependent members of ilies a special form of certificate will be given. The possession of ficate will enable the person to whom it is issued to purchase and g a limited period of time railway tickets at a much reduced nd from any point in Canada.

railway rates are also provided for members attending the Con- n certain points on the Pacific coast of the United States.

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## ITERNATIONAL ENGINEERING CONGRESS.

ternational Engineering Congress of 1915 will be held under the of the American Society of Civil Engineers, American Society of cal Engineers, American Institute of Electrical Engineers, Society Architects and Marine Engineers, and the American Institute of Engineers, in connection with the Panama-Pacific Exposition. nents for this Congress are now proceeding under the General ee, the Institute representatives on which are given on p. xxxv. esting technical program is promised. The time of the Congress 0 to 25, 1915. Members of the Institute are cordially invited to d participate in the proceedings of the Congress.

Spanish American Iron Co., Daiquiri, Cuba, C. F. Rand, President. 19
of explorations at Daiquiri, Cuba.
Present position: Mining Supt., Spanish American Iron Co.

**Burr Arthur Robinson**, New York, N. Y.
Proposed by L. O. Kellogg, Bradley Stoughton, R. Dawson Hall.
Born, 1883, Bradford, Pa. 1888–97, Grammar Schools, Rochester,
Rochester Business Institute, Rochester, N. Y. (Bookkeeping and Stenogr
Masten Park High School, Buffalo, N. Y. 1905–09, Mass. Inst. of Tec
Mass.; B. S. in course in Mining Engineering and Metallurgy. 1899
Pittsburgh Coal & Iron Co., Rochester, N. Y., Stenographer. 1909–10, Mc
Pluma, S. D., Roustabout and costkeeper. 1910–11, Stratton's Independe
Colo.; Shift foreman and chemist in cyanide mill. 1911–13, *Engineering and*
New York, N. Y., Editorial Staff.
Present position: Editorial Staff, American Institute Mining Engineers

**Jacob B. Ross**, Denver, Colo.
Proposed by R. M. Atwater, Jr., Kirby Thomas, H. A. Prosser.
Born, 1878, Silverton, Colo. Graduate of High School, Montrose, C
training from fifteen years' operating. 1898, Vulcan and Gunnison Mine
Colo. 1899, Thomas F. Walsh. 1902, Camp Bird Mine. 1902–04, Camp
1904–13, Organized and operated Ross M. & M. Co., Congress Gold &
Juan Gold Mines Co. and several others in San Juan County, Colo.
Present position: Mine operator.

**Louis Ross**, Boston, Mass.
Proposed by Allen H. Rogers, Lucius W. Mayer, Edmond N. Skinner.
Born, 1859, Bangor, Maine. 1874, Boston Grammar schools. 1887
School, Boston. 1881, Cusihuirrachic Mining Co., Mexico; Kidder, Pea
ton; Otto Hoffman, ——— Russell. 1883–87, Cieneguilla Mining Co.,
1887–00, Examination of Mines, T. W. Lawson, Schofield, Whicher &
strong & Co., F. A. Evans & Co., all of Boston. 1902–06, Managing D
Mining Co., Durango, Mexico.
Present position: Consulting Engineer to A. C. Burrage, Boston.

**Frederick Taber Rubidge**, New York, N. Y.
Proposed by J. F. Kemp, Thomas T. Read, Robert Peele.
Born, 1877, Ontario, Canada. 1899, B. S. (C. E.), University of Col
Columbia School of Mines. 1901–10, Asst. Supt. New Jersey Zinc Co.,
nace N. J.
Present position: 1910 to date, Mgr. of Mines as follows: St. Lawre
Northern Alabama Coal, Iron & Ry. Co., Central Coal Co., Ala., Seab
Co., Ala.

**Andrew F. Sale**, Kimberly, Nev.
Proposed by E. F. Gray, Charles F. Rand, R. M. Atwater, Jr.
Born, 1880, Columbus, Texas. 1897–01, State School of Mines of C
1901–05, General experience for short times ranging from miner to surve
mill-man to chemist of mills and location surveyor.
Present position: 1906 to date, Assayer and Surveyor and later Gene
neer of Giroux Con. Mines Co.

**Eugene Mitchell Sawyer**, Leopold, N. M.
Proposed by Walter Douglas, Cleveland H. Dodge, A. R. Ledoux.
Born, 1882, Bangor, Maine. 1904, A. B., Harvard. 1906, S. B., Harva
Harvard. 1906–07, Austin Teaching Fellow in Mining and Metallurgy at
09, Engineering Dept., Copper Queen Cons. Mining Co., Bisbee, Ariz.
Geologist, Copper Queen Cons. Mining Co., Bisbee, Ariz. 1910–12, Sup
Prospect for Copper Queen Cons. Mining Co.
Present position: Supt., Burro Mountain Copper Co., Leopold, N. M.

**William Graham Scott**, Morenci, Ariz.
Proposed by M. H. McLean, David D. Irwin, John Kiddie.
Born, 1873, Causawayhead, Stirlingshire, Scotland. Dec., 1897–Mar
and Chemistry at Columbia College, New Westminster, B. C. 1900–05,
Int. Correspondence Schools, Scranton, Pa. 1891–97, Miner at various
and California. Mar.–Dec., 1897, In charge of Cromwell and Champ
Lake, British Columbia. Mar.–Aug., 1898, In charge of McKinnen &

oct, British Columbia. Oct., 1893–Apr., 1903, Foreman and Supt. of Queen Alamo, near Sandon, British Columbia. July, 1903–04, Supt., Continental , Alta, Utah. July–Dec., 1904, working property of my own at Bingham, ., 1904–Apr., 1905, Shift Boss at Yampa Mines, Bingham, Utah. Apr., 1905– Foreman for H. Doolittle Properties, Bingham, Utah.
osition: Feb., 1906, to date, Mine Foreman at Morenci for Arizona Copper on, Ariz.

**Iyndman Seip**, Wilkes-Barre, Pa.
by Charles E. Schwarz, H. A. Wheeler, H. A. Buehler.
8, Mauch Chunk, Pa. 1906, Graduated from the Wilkes-Barre High School. ated from the School of Mines of the Pennsylvania State College; B. S. Min- ering. Feb., 1907–Sept., 1908, Employed in Engineering Dept. of the Lehigh Co. at Wilkes-Barre, Pa., as Asst. Transitman under Charles Enzian, Divi- er. Employed by the same company during the two following summers. June, 1913, Asst. Supt. of the North American Smelting Co., Ltd., at Perth Road, a.
osition: Engineer, Consolidation Coal Co., at Jenkins, Letcher County, Ky.

**Monroe Shoop**, Felton, Cuba.
by Frederick Charles Wilson, James E. Little, Charles F. Rand.
'8, Hummelstown, Pa. 1885–90, Public Schools of Lebanon County, Pa. banon High School, Lebanon, Pa. Sept., 1893–Jan., 1895, Pennsylvania Col- burg, Pa. Jan., 1895–Nov., 1900, Private instruction in mathematics and ering under George W. Hayes. C. E., A. I. M. E., Lebanon, Pa. Jan., 1895– Chainman, Rodman and Instrumentman in the employ of George W. Hayes, er, Lebanon, Pa. Nov., 1900–Dec., 1906, Transitman on location of Cuba R. A. Smith, Chief Engineer, and Asst. Engineer on construction with The Cuba A. C. Reed. Jan., 1907–Nov., 1909, Asst. Engineer on construction of Mayari . of the Spanish American Iron Co. Nov., 1900–Feb., 1913, Mine Supt., es, the Spanish American Iron Co., A. C. Reed, Supt.
osition: Asst. to the Supt., Mayari Division, the Spanish American Iron Co.; n, Supt.

**Monell Staples**, Morenci, Ariz,
by M. H. McLean, Ellis W. Honeyman, John Kiddie.
38, Arlington, Neb. 1895–02, Grammar Schools, Boise, Idaho. 1902–06, l, Red Bluff, Cal. 1906–11, College of Mining, University of California; B. , six months sampling underground for Arizona Copper Co., Morenci, Ariz. onths assaying and one month asst. underground surveyor, Arizona Copper Co., iz.
osition: Sept., 1912, to date, in charge of mining costs and head sampler for per Co., Morenci, Ariz.

**r T. Thomson**, Morenci, Ariz.
by M. H. McLean, David D. Irwin, Ellis W. Honeyman.
5, Edinburgh, Scotland. 1885–02, Edinburgh Academy. 1902–06, Richard ., Chartered Accountants, Edinburgh, Scotland. 1906–12, Arizona Copper Co., n, Ariz., and Arizona and New Mexico Ry., Clifton, Ariz. Various positions e years, including Cashier and Purchasing Agt. A. C. Co. and Genl. Supt. and A. & N. M. Ry.
osition: Genl. Mgr., Detroit Copper Mining Co. of Arizona.

**ngier Vernon**, Unsan, Korea.
by J. B. Lower, A. E. Deardorff, Thomas W. Van Ess.
rch 6, 1885, Providence, R. I. 1908, Graduated, Civil Engineering Degree, University. Three summers on U. S. Coast Survey; 3½ years with Oriental l Mining Co.
osition: Asst. Supt. for the above company.

**Ware**, Anaconda, Mont.
by E. P. Mathewson, Frederick Laist, C. D. Demond.
9, Memphis, Mo. 1899, Graduated Missouri Wesleyan College, A. B. 1904–13, ashoe Reduction Works, Anaconda, Mont.
osition: Supt Blast Furnaces, Washoe Reduction Works.

**William Melvin Weigel**, Kingston, Ont.
Proposed by Charles E. Schwarz, H. A. Wheeler, H. A. Buehler.
Born, 1878, Montrose, Ill. 1895–96, Kansas City, Mo., High School. 1896–(
souri School of Mines and Metallurgy, Rolla, Mo.; B. S. 1903, E. M. 1900,
Electric & Mfg. Co., St. Louis, Mo. 1901–02, Morning Star Mining Co. and
Star Ry. Co., Arkansas. 1902–06, St. Louis Smelting & Refining Co., St. Louis,
Collinsville, Ill., Chemist, Asst. Supt. and Supt. 1906, Federal Lead Co., Fla
Mo., drafting and designing. 1906–08, Stanley Smelting Works, Canada, Supt.
Associate Professor of Mining, Pennsylvania State College.
Present position: Genl. Supt. Mines and Smeltery, North American Smelti
Kingston, Ont.

**James Raymond Wester**, Morenci, Ariz.
Proposed by M. H. McLean, Ellis W. Honeyman, John Kiddie.
Born, 1877, Dallas, Ore. 1904, Washington State College, Pullman, Wash.
1905–07, Supt., Morenci Water Co. 1907–10, Genl. Mgr., New York-Arizona
Copper Co., Morenci, Ariz.
Present position: 1910 to date, Shift Boss and Foreman of the Arizona Copper
the Humboldt Mine.

**Harold O. Whitnall**, Hamilton, N. Y.
Proposed by Karl Eilers, Thomas T. Read, John W. Finch.
Born, 1877, Morristown, N. J. 1898–1900, Colgate University; Ph. B. 1900–
gate University; M. A. 1902–03, Harvard University. 1904, Asst. Harvard
School. 1904–06, Asst. in Geology and Biology, Colgate University. 1906–10, In
in Geology, Colgate University. 1905, Field Asst., New York Geological Survey.
position in 1906–07, 1909–12.
Present position: 1910 to date, Associate Professor of Economic Geology an
eralogy, Colgate University.

**Ralph Wilcox**, Miami, Ariz.
Proposed by J. Parke Channing, N. O. Lawton, R. C. Canby.
Born, 1879, Clifton, Keweenaw County, Mich. 1897, High School, Negaunee
1898, One year in Ohio Wesleyan University, Delaware, Ohio; Literary Course.
McDonald Exploration Co. Calumet & Arizona Mining Co. Sullivan Machin
Standard Oil Co. Transvaal Copper Mining Co. (Contracting, three years.)
Iron Mining Co. Wassau West Amalgamated Mines, Gold Coast. U. S. Govt
ragua Survey of 1900. Alaskan Development Co. Santa Eulalia Exploration Co.
Present position: Foreman of Diamond Drills for the Miami Copper Co.

*Associate.*

**Clarence Ivins Bradley**, Bradford, Conn.
Proposed by J. F. McClelland, Joseph W. Roe, L. W. Bahney.
Born, Aug. 13, 1891, Branford, Conn.
Present position: Student in Mining at Sheffield Scientific School, Yale Uni
New Haven, Conn.

*Junior Members.*

**Douglas Calvert Corner**, Madison, Wis.
Proposed by Edwin C. Holden, C. K. Leith.
Born, 1891, Baltimore, Md. 1909, Graduated Baltimore Polytechnic Institute.
13, University of Wisconsin. Summer 1907, Piedmont Soapstone Co., Tye Riv
Summer 1908, American Zinc Ore Separating Co., Platteville, Wis. June, 1909–(
Wisconsin Zinc Co., Platteville. Wis.
Present position: Student, University of Wisconsin.

**Zar T. Crittenden**, Butte, Mont.
Proposed by D. C. Bard, C. H. Bowman, C. W. Goodale.
Born, 1887, Greenville, Mich. 1901–05, Mt. Pleasant, Mich., Public Schools.
13, Montana State School of Mines; E. M.
Present position: Student.

**Chris. G. Dobson**, Butte, Mont.
Proposed by D. C. Bard, C. H. Bowman, C. W. Goodale.
Born, 1889, Sioux City, Iowa. 1909–10, University of Washington. 1911–1
tana State School of Mines. June–Oct., 1910, Miner, Timberman and Timber Bo

eson Coal & Coke Co., Wilkeson, Wash. June, 1911–June, 1912, Miner, Timber-
t the Moonlight, Badger State and Pennsylvania mines, Butte, Mont.
sent position : Student at Montana State School of Mines.

**E. Dodge**, Butte, Mont.
posed by D. C. Bard, C. H. Bowman, C. W. Goodale.
n, 1886, Great Bend, Kan. 1893–00, Grammar schools of Great Bend and Kansas
Kan. 1900–03, High School of Shawnee, Okla. 1903–05, Apprentice machinist
. I. & P. Ry. shops, Shawnee, Okla. 1905–08, A. and M. College of Oklahoma,
ater ; completed chemistry course as a special student. 1908–09, Assayer, El Tigre
g Co., El Tigre, Sonora, Mexico. 1909–11, Engineer, El Tigre Mining Co., and
Engineer of Construction with D. L. H. Forbes at El Tigre. 1911–12, Highway
ission of San Diego County, Cal.
sent position : Special Student at Montana State School of Mines.

**lson Burnes Gatch**, St. Louis, Mo.
posed by Arthur L. Walker, William Campbell, Robert Peele.
n, 1888, St. Joseph, Mo. 1907, Diploma, Smith Academy, St. Louis, Mo. 1911,
Degree, Columbia College.
sent position : Student in Mining.

**gustin S. Horcasitas**, New York City, N. Y.
posed by Howard Eckfeldt, Henry S. Drinker, Benjamin L. Miller.
n, 1890, Chihuahua, Mexico. 1905–06, Fordham University. 1906–09, New York
ary Academy. 1909–13, Lehigh University. (E. M.)
sent position : Engineering Student.

**ed Warren Hyde**, Joliet, Ill.
posed by Arthur L. Walker, William Campbell, J. F. Kemp.
n, 1890, Joliet, Ill. 1905–08, Joliet High School. 1909–13, Columbia University.
sent position : Student in Metallurgy.

**rbert W. Lamb**, South Bethlehem, Pa.
posed by Howard Eckfeldt, Benjamin Miller, Henry S. Drinker.
n, 1891, Adrian, Mich. 1898–08, Public Schools, Adrian, Mich., and Monessen,
1908–09, Kiskiminetas Springs Schools, Saltsburg, Pa. 1909–13, Lehigh Univers-
chool of Mines. 1912, June and July, Mine and railroad surveying.
sent position : Student in Mining Engineering.

**ank H. Madson**, Racine, Wis.
posed by Edwin C. Holden, C. K. Leith.
n, 1890, Racine, Wis. 1904–07, Racine High School, Science Course. 1908–11,
ersity of Wisconsin, Course in Mining Engineering. 1911–12, Miner, Timberman,
er, Anaconda Copper Mining Co., Butte, Mont., in the Mt. View Mine.
sent position : Student, University of Wisconsin.

**rris Kent Petty**, South Bethlehem, Pa.
posed by Howard Eckfeldt, Benjamin L. Miller, Henry S. Drinker.
n, 1891, Pittsburg, Pa. 1906–09, Kiskiminetas Springs School. 1909–13, Lehigh
ersity, Department Mining Engineering. Before entering the Mining School of
gh University I spent one summer as machinist's helper in the railroad shops of the
burgh & Lake Erie R. R., McKee's Rocks, Pa. After entering the Mining School at
gh I spent the summer vacation of my Freshman year in the same employment. Fol-
g summers, Engineering Corps of the Ellsworth Collieries Co., Ellsworth, Pa., and
ngahela R. R. Co., Brownsville, Pa.
sent position : Student, School of Mines, Lehigh University.

**win Allen Pierse**, Great Falls, Mont.
posed by D. C. Bard, C. H. Bowman, C. W. Goodale.
rn, 1887, Montana. 1908, Great Falls High School. 1913, Montana State School of
s. 1909, Worked at B. & M. Smelter, Great Falls. Worked for the Barker Mines
at Barker, Mont.
sent position : Student.

**lyne Francis Pratt**, Butte, Mont.
posed by D. C. Bard, C. H. Bowman, C. W. Goodale.
rn, 1887, Portland, Ore. 1894–01, Portland Public Schools. 1901–05, Portland
School. 1906–07, Oregon Agricultural College. 1909–11, Colorado School of Mines.
-13, Montana School of Mines ; E. M.
sent position : Student.

**John Calvert Scoles**, Granton, Wis.
Proposed by Edwin C. Holden, C. K. Leith.
Born, 1889, Red Cloud, Neb. 1909, Graduated High School, Neillsville, Wis College of Letters and Science, University of Wisconsin. 1911-13, College of Engi University of Wisconsin. Summer 1911, Instrumentman, Township resurvey County, Wis. Employer, R. S. Owen, Madison, Wis. Summer 1912, Asst. E Lands Department, Algoma Central & Hudson Bay R. R., Mgr. J. A. Dresser, Sa Marie, Ont. Engineers with whom I worked, H. H. Bradt, Duluth, Minn.; R. 59 Glasgow St., Toronto, Ont.
Present position: For summer 1913, Geologist, Wisconsin Geological and Natu tory Survey, E. F. Bean, Geologist-in-charge, Madison, Wis.

**Oliver P. Searing**, Seattle, Wash.
Proposed by Joseph Daniels, Milnor Roberts, Henry Landes.
Born, 1888, Jacksonville, Fla. 1903-07, Duval High School, Jacksonvill diploma. 1908-09, Washington and Lee University, Lexington, Va. 1909-13, ity of Washington College of Mines, Mining and Geology. Summer of respecti 1908, clerk, Atlantic Coast Line R. R., Jacksonville, Fla. 1908, Southern Weigh Inspection Bureau, Jacksonville, Fla. 1909, Denny Renton Clay & Coal Co.' Taylor, Wash. 1911, Apex Gold Mine, Berlin, Wash. 1912, Tacoma Smelter, Wash.
Present position: Student Asst., University of Washington, College of Mines. ber Student Affiliated Society.

## CHANGES OF ADDRESS OF MEMBERS.

The following changes of address of members have been received Secretary's office during the month of June, 1913. This list, to with the list published in *Bulletin* Nos. 76 to 78, April to 1913, and the foregoing list of new members, therefore, supplemen annual list of members corrected to Mar. 1, 1913, and brings it up date of July 1, 1913.

ALLEN, JOHN H. ..... Walpol
ALLEN, ROBERT ..... 40 Queens Road, Leigh, Essex, E
AMSDEN, O. B. ..... P. O. Box 693, Idaho Spring
BARNES, BLAKESLEE ..... P. O. Box 493, Shrevep
BEELER, HENRY C. ..... 127 15th St., Denve
BODY, J. FRANCIS, Vice-Prest., The Reinforcement Co., 24 W. 39th St., New Yor
BOWRON, WILLIAM M. ..... Sugar Valley, Gordon
BRETHERTON, W. L. ..... 900 New Hampshire Ave., Los Angel
BREWER, CARL ..... Crystal Falls
BROMLY, ALFRED H., Care Alfred James, 18 Eldon St., London, E. C., E
CARSON, ELLARD W. ..... 233 Consolidated Realty Bldg., Los Angel
CASTO, WILLIAM H., JR. ..... Testing Dept., Anaconda Copper Co., Anaconda
COMSTOCK, CHARLES W. ..... 514 First National Bank Bldg., Denve
COXE, EDWARD H., Genl. Supt., La Follette Coal, Iron, & R. R. Co., La Follette
CROCKER, A. L. ..... 626 Plymouth Bldg., Minneapolis
CLAPP, LAWRENCE R. ..... Federal Lead Co., Flat Riv
CUMMINS, HARLE O. ..... Reddi
DALBURG, FRANK A. ..... Ralst
DAMON, GEORGE G. ..... Beaver Gold Dredging Co., Loon
DE LASHMUTT, IVAN, Mine Supt., Standard Silver-Lead Mining Co., Silverton, B. C.,
DINGWALL, WILLIAM B. A. ..... The Hutchins, 205 Garden St., San Antonio,
DUNN, WILLIAM R. ..... Phillipsburg
DURELL, CHARLES T. ..... Care American Girl, Ogil
EMMEL, RUDOLPH ..... 117 Hamilton St., Butte
EVERED, N. J., Mgr. ..... Keeley Mine, Silver Centre, Ont.,
GRAVE, ERNEST ..... Calzada de la Veronica 34, Mexico City,
GREENWAY, THOMAS J., Min. and Met. Engr., Genl. Mgr., Potters Ore Treatment, Ltd., 60 Queen St., Melbourne, Vic., Au
HAMMER, JOHN G. ..... Standard Brick & Tile Co., 310 Henry Bldg., Portlan
HAUBER, MATHIAS, JR. ..... Instructed to hold a
HEDLEY, ROBERT R. ..... 405 Cotton Bldg., Vancouver, B. C.,

EGINALD E. ..... 44 Lombard St., Toronto, Ont., Canada.
W. H. ..... Little Boar's Head, N. H.
FRANK G., Vice-Prest., E. J. Longyear Co.,
710 Security Bank Bldg., Minneapolis, Minn.
OWEN, The Famatina Co., Ltd., Chilecito, Prov. La Rioja,
Argentine Republic, So. America.
HUN TET. ..... 87 Yan Zai St., Canton, Kwantung, China.
VILLIAM H., Jr. ..... Boyles, Ala.
OHN B. ..... 106 York Ave., Pittston, Pa.
, HAROLD N. ..... 526 Yeon Bldg., Portland, Ore.
S. Paul, Chem., Western Precipitation Co., 1016 W. 9th St., Los Angeles, Cal.
RE, ORLEANS. ..... University Park, Colo.
AMES W. ..... Leonard Hotel, Butte, Mont.
ARTHUR, Supervising Engr., New York Engrg. Co., 2 Rector St., New York, N. Y.
AY, WILLIAM B. ..... 22 Exchange Pl., New York, N. Y.
L, CHARLES ..... 1935 79th St., Brooklyn, N. Y.
AN, MANSFIELD ..... 432 Fourth Ave., New York, N. Y.
T, CHARLES A., Mgr. Steel and Wire Mill, Standard Steel Co.,
1135 Kirkland Ave., Birmingham, Ala.
, FREDERICK H. ..... 1321 Wood Ave., Colorado Springs, Colo.
, BAYARD S., Testing Dept., Washoe Reduction Wks.,
Anaconda Copper Mining Co., Anaconda, Mont.
DUNCAN M. ..... Apartado 37, Guanajuato, Mexico.
R, A. C. ..... Scranton, Tooele Co., Utah.
N, C. E. ..... 26 W. 35th St., Bayonne, N. J.
N, ARTHUR W. ..... 1111 12th Ave., Spokane, Wash.
SAMUEL L. ..... Raymond, Madera Co., Cal.
, HOWARD A. ..... 63 Wall St., New York, N. Y.
ROBERT S. ..... Empire, Canal Zone.
CHARLES F. ..... 215 First National Bank Bldg., San Francisco, Cal.
S, ROBERT P. ..... Mt. Lyell Mining & Railway Co., Queenstown, Tasmania.
S, CHARLES E. ..... Dome Mines Co., So. Porcupine, Ont., Canada.
, MAX ..... Great Neck, L. I., N. Y.
CHARLES E., Fraser & Chalmers, Ltd., 3 London Wall Bldgs.,
London, E. C., England.
LFRED VON DER. ..... 233 Broadway, New York, N. Y.
JOHN A., Genl. Mgr., Shenango Furnace Co., Alworth Bldg., Duluth, Minn.
, MAX C. ..... P. O. Box 332, Eagle Pass, Tex.
V. M. H., Cuidado de Consul Americano, Barranquilla, Colombia, So. America.
, NANABHOY R., Min. Engr., Tata Sons & Co., 82 Wall St., New York, N. Y.
EY, W. H. ..... 527 Waverly St., Palo Alto, Cal.
, JAMES C., Care E. H. Hunter & Co., 1 St. Mary Axe, London, E. C., London.
EDWARD A. ..... Care T. B. Scott, Sinai Mining Co., Ltd., Suez, Egypt.
AMASA D. ..... 73 W. 5th St., Chillicothe, Ohio.
-ROOD, MRS. L. A. ..... Care First National Bank, Pueblo, Colo.
HAYES, MAURICE W. ..... McEnaney Mines, Schumacher, Ont., Canada.
, WILLIAM H. ..... Care C. H. Forbes, Phillips Academy, Andover, Mass.
K K., Min. Engr. ..... Bureau of Industry and Commerce, Canton, China.
LEN, MATTHEW ..... 44 Pine St., New York, N. Y.
W. ROGERS ..... Tyrone, N. M.
JAMES S. C., Care Vincent Franceschini, Calle 73 Santome,
Santo Domingo, Santo Domingo, W. I.
ER, HERMAN ..... 1026 Lake St., Los Angeles, Cal.
S, WILLIAM ..... 510 Dime Bldg., Detroit, Mich.
ELWOOD J. ..... 540 W. 122d St., New York, N. Y.
, WALTER W. ..... Wenden, Yuma Co., Ariz.
ER, HERBERT C. P. ..... P. O. Box 19, Moscow, Russia.
HO ..... 36 Sze Chuan Rd., Shanghai, China.

## ADDRESSES OF MEMBERS AND ASSOCIATES WANTED.

| Name. | Last Address of Record, from which Mail has bee |
|---|---|
| Armstrong, Richard E., | 416 E. So. 2d St., Salt Lake City, |
| Casey, John P., | 1509 Montana St., El Paso, Tex. |
| David, W. M., | Care C. G. Hussey & Co., Second burg, Pa. |
| Ederheimer, Leopold, | Maine Mining & Mfg. Co., 52 Br York, N. Y. |
| Eggers, John H., Jr., | Hughes Creek Mine, Kings Rive |
| Ferguson, Donald, | P. O. Box 644, Goldfield, Nevad |
| Fergusson, Hugh B., | 216 Loo Bldg., Vancouver, B. C. |
| Finletter, John R., | 625 St. Francis Hotel, San Franc |
| Geisendorfer, Henry A., | Ariz-Nev. Copper Co., Hillside, |
| Heywood, William A., | 4 Broad St. Pl., London, E. C., |
| Hollis, R. W., | Silverton, Colo. |
| Horschitz, Richard J., | P. O. Box 453, Haileybury, Ont. |
| Jackson, Walter H., | Mapimi, Dur., Mexico. |
| Johnson, Dion L., | 325 Water St., Pittsburg, Pa. |
| Lampshire, John O., | Vulture Mine, Wickenburg, Ariz |
| MacKay, Philip A., | "Illinois," Winmuera Pl., Melbourne, Vic |
| Malins, Francis A., | Metates Mine, San Marcos, Sin., |
| Moore, Roy W., | P. O. Box 48, Velasco, Tex. |
| Nelson, D. W. C., | Baker City, Ore. |
| Peterson, Frank, | H. W. Hellman Bldg., Los Angel |
| Rathborne, Merwyn R. W., | Amargosa, via Las Vegas, Nev. |
| Reynolds, Llewellyn, | Socorro Mines, Mogollon, N. M. |
| Rickard, Harold, | Foley-O'Brien, Ltd., So. Porcupine, O |
| Russell, Branch E., | Apartado 22, Nacozari, Son., Me |
| Stoddart, A. W., | 638 Salisbury House, London, E. |
| Van Ness, William W., | 622 Salisbury House, London, E. |
| Vinicombe, Robert E. B., | Post Restante, Vladikafkas, So. |
| Watson, Ralph W., | Calloo, Utah, Clifton Mail box. |
| Webster, Erastus H., | Hotel Cosmopolita, Guadalajara, |
| Williams, John R., | Care Standard Bank of So. Afri ents Lane, London, E. C., E |
| Woods, Clarence, | Shawmut, Cal. |

## NECROLOGY.

The deaths of the following members were reported to the S office during the month of June, 1913:

| Date of Election. | Name. | Date |
|---|---|---|
| 1886. | *Bonzano, Adolphus, | May 5 |
| 1903. | *Havard, Francis T., | May 2 |

* Member.

## STANDING COMMITTEES.

*Executive.*

CHARLES F. RAND, *Chairman.*

ES F. KEMP,
ERT R. LEDOUX,
JOSEPH W. RICHARDS,
BENJAMIN B. THAYER.

*Membership.*

BENJAMIN B. THAYER, *Chairman.*

TER R. INGALLS,
N D. IRVING,
JOHN H. JANEWAY, JR.,
HERMAN A. PROSSER.

*Finance.*

EDWARD L. YOUNG, *Chairman.*

L EILERS,
JAMES GAYLEY.

*Library.*

JAMES F. KEMP, *Chairman.*[1]

N HAYS HAMMOND,[3]
X. C. HUMPHREYS,[4]
E. GYBBON SPILSBURY,[2]
BRADLEY STOUGHTON.

*Papers and Publications.*

TOUGHTON, *Ch.*
ALL,
. AUSTIN,
INBINE,
. BLAUVELT,
ANNER,
RUNTON,
ETANI,
AMPBELL,
CARLTON,
,
L H. EMMONS,
NCH,
. FULTON,
JAMES GAYLEY,
C. WILLARD HAYES,
HEINRICH O. HOFMAN,
HENRY M. HOWE,
LOUIS D. HUNTOON,
WALTER R. INGALLS,
JAMES F. KEMP,
CHARLES KIRCHHOFF,
CHARLES K. LEITH,
RICHARD MOLDENKE,
SEELEY W. MUDD,
HENRY S. MUNROE,
R. V. NORRIS,
EDWARD W. PARKER,
EDWARD D. PETERS,
ROSSITER W. RAYMOND,
JOSEPH W. RICHARDS,
ROBERT H. RICHARDS,
HEINRICH RIES,
E. F. ROEBER,
ALBERT SAUVEUR,
CHARLES H. SHAMEL,
HENRY L. SMYTH,
GEORGE C. STONE,
RALPH H. SWEETSER,
FELIX A. VOGEL,
ARTHUR L. WALKER,
ROLLA B. WATSON.

## COMMITTEE ON INCREASE OF MEMBERSHIP.

C. R. CORNING, *Chairman.*

ADOLPHE E. BORIE, *First Vice-Chairman.*

THOMAS T. READ, *Secretary*, Woolworth Bldg., New York, N. Y.

*Vice-Chairmen.*

N H. ALLEN,
HARD M. ATWATER, JR.,
RGE D. BARRON,
HESTER BEATTY,
ARKE CHANNING,
GEORGE M. COLVOCORESSES,
ROBERT PEELE,
CHARLES P. PERIN.
JOSEPH A. VAN MATER,
ARTHUR L. WALKER.

NEDICT,
ANNER,
RTER,
LARK,
EE,
CRAWFORD,
SON,
ILLIERS,
UGLAS,
UGLAS,
. EAVENSON,
HOWARD ECKFELDT,
R. C. GEMMELL,
F. LOUIS GRAMMER,
ERNEST A. HERSAM,
EDWIN C. HOLDEN,
WILLIAM L. HONNOLD,
REGINALD E. HORE,
C. COLCOCK JONES,
EUGENE P. KENNEDY,
CHESTER F. LEE,
RICHARD S. McCAFFERY,
JAMES F. McCLELLAND,
MILTON H. McLEAN,
PHILIP N. MOORE,
T. H. O'BRIEN,
JAMES J. ORMSBEE,
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JOHN B. PORTER,
R. M. RAYMOND,
ROBERT H. RICHARDS,
LEROY SALSICH,
HENRY LLOYD SMYTH,
F. W. TRAPHAGEN,
ELTON W. WALKER.

---

Feb., 1914. [2] Until Feb., 1915. [3] Until Feb., 1916. [4] Until Feb., 1917.

## TECHNICAL COMMITTEES.

### *Iron and Steel.*

CHARLES KIRCHHOFF, *Chairman.*
ALBERT SAUVEUR, *Vice-Chairman.*
A. A. STEVENSON, *Vice-Chairman.*
HERBERT M. BOYLSTON, *Secretary*, Harvard University, Cambridge, Mas

JOHN BIRKINBINE,
WILLIAM H. BLAUVELT,
JAMES GAYLEY,
HENRY D. HIBBARD,
HENRY M. HOWE,
ROBERT W. HUNT,
J. ESREY JOHNSON, JR.,
WILLIAM KELLY,
RICHARD MOLDENKE
JOSEPH W. RICHARDS
E. GYBBON SPILSBUR
FELIX A. VOGEL,
LEONARD WALDO,
WILLIAM R. WALKER
WILLIAM R. WEBSTE
FREDERICK W. WOOD

### *Precious and Base Metals.*

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L. D. RICKETTS, *Vice-Chairman.*
DARSIE C. BARD, *Secretary*, Montana State School of Mines, Butte, Mont.

LEONARD S. AUSTIN,
DAVID W. BRUNTON,
THEODORE B. COMSTOCK,
STANLEY A. EASTON,
SAMUEL S. FOWLER,
THOMAS J. GRIER,
HENNEN JENNINGS,
EDMUND B. KIRBY,
CHARLES W. MERRIL
WILLET G. MILLER,
ALBERT F. SCHNEIDE
GEORGE C. STONE,
BENJAMIN B. THAYE

### *Mining Geology.*

JAMES F. KEMP, *Chairman.*
JOHN W. FINCH, *Vice-Chairman*,
R. A. F. PENROSE, JR.,
L. C. GRATON, *Secretary*, Harvard Geological Museum, Cambridge, Mass.

RALPH ARNOLD,
H. FOSTER BAIN,
JOHN M. BOUTWELL,
H. A. BUEHLER,
MARIUS R. CAMPBELL,
WILLIAM H. EMMONS,
HENRY LANDES,
ALFRED C. LANE,
CHARLES K. LEITH,
R. V. NORRIS,
EZEQUIEL ORDONEZ,
WILLIAM B. PHILLIP
JOSEPH H. PRATT,
F. L. RANSOME,
HEINRICH RIES,
RENO H. SALES,
WILLIAM G. SHARP,
HENRY L. SMYTH,
JOSIAH E. SPURR,
HORACE V. WINCHEL

### *Mining Methods.*

J. PARKE CHANNING, *Chairman.*
DAVID W. BRUNTON, *Vice-Chairman*,
H. C. HOOVER, *Vice-Chairm*
——— ———, *Secretary.*

TRUMAN H. ALDRICH,
R. B. BRINSMADE,
DOUGLAS BUNTING,
LOUIS S. CATES,
CHARLES CATLETT,
R. M. CATLIN,
CHARLES A. CHASE,
S. B. CHRISTY,
F. W. DENTON,
E. V. D'INVILLIERS,
STANLEY A. EASTON,
J. R. FINLAY,
R. C. GEMMELL,
JOHN GILLIE,
HARRY A. GUESS,
THOMAS J. GRIER,
JOSEPH A. HOLMES,
WILLIAM L. HONNOLD,
JAMES E. JOPLING,
ROBERT A. KINZIE,
C. B. LAKENAN,
THOMAS H. LEGGETT,
J. F. McCLELLAND,
JAMES MAC NAUGHTON,
SEELEY W. MUDD,
H. S. MUNROE,
W. J. OLCOTT,
JOHN B. PORTER
MILNOR ROBERT
OSCAR ROHN,
W. L. SAUNDERS,
H. L. SMYTH.
ARTHUR THACH
B. W. VALLAT,
SAMUEL D. WAR
GEORGE WEIR,
BULKLEY WELLS

## EXECUTIVE COMMITTEES OF LOCAL SECTIONS.

*New York.*

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E. MALTBY SHIPP, *Treasurer.*
RGE F. KUNZ, ——— ———.

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ROBERT H. RICHARDS, *Chairman.*
ALBERT SAUVEUR, *Vice-Chairman.*
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OTHY W. SPRAGUE, HENRY A. WENTWORTH.

*Spokane.*

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GLENVILLE A. COLLINS, *Vice-Chairman.*
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NCIS A. THOMSON, JAMES F. McCARTHY.

*Puget Sound.*

CHESTER F. LEE, *Chairman.*
C. R. CLAGHORN, *Vice-Chairman.*
JOSEPH DANIELS, *Secretary-Treasurer*, Univ. of Washington, Seattle, Wash.
. BUTLER, M. K. RODGERS.

*Southern California.*

THEODORE B. COMSTOCK, *Chairman.*
SEELEY W. MUDD, *Vice-Chairman,*
CK J. H. MERRILL, *Secretary-Treasurer*, 610 Citizens National Bank Bldg., Los Angeles, Cal.
OLCOCK JONES, FRANK ROBBINS.

## COMMITTEE ON THE USE OF ELECTRICITY IN MINES.

WILLIAM KELLY, *Chairman.*
O. BLACKWELL, JOHN LANGTON, LOUIS S. NOBLE,
W. GOODALE, THOMAS H. LEGGETT, DAVID B. RUSHMORE.
FREDERICK W. O'NEIL.

## NOMINATING COMMITTEE.

JOHN HAYS HAMMOND, *Chairman.*
DERICK G. COTTRELL, JAMES W. MALCOLMSON,
RLES W. GOODALE, RICHARD S. McCAFFERY,
STER F. LEE, ROBERT H. RICHARDS.

## INSTITUTE REPRESENTATIVES.

*United Engineering Society Trustees.*

E DWIGHT,[1] JAMES F. KEMP,[2] JOSEPH STRUTHERS,[3]

*John Fritz Medal Board of Award.*

. NORRIS,[1] JAMES F. KEMP,[3]
RLES KIRCHHOFF,[2] ALBERT SAUVEUR,[4]

*General Conference Committee of Engineering Societies.*

JAMIN B. LAWRENCE, J. PARKE CHANNING.

*Committee of Management, International Engineering Congress*, 1915.

H. BENJAMIN, NEWTON CLEAVELAND, CHARLES F. RAND,
R BAIN, WILLIAM S. NOYES, BRADLEY STOUGHTON.

*American Association for the Advancement of Science.*

NRICH O. HOFMAN, JOHN D. IRVING.

*Committee on Patent Law Legislation.*

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*Advisory Board of the National Conservation Congress.*

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# Reducibility of Metallic Oxides as Affected by Heat Treatment.

BY WOOLSEY McA. JOHNSON, HARTFORD, CONN.

(Butte Meeting, August, 1913.)

tallurgical circles it is known widely, but somewhat vaguely, ease of reduction of metallic oxides depends largely on the have been prepared. It is likewise known that different carbon have different and greatly varying reducing powers. purpose of this paper to show somewhat more definitely and ally these facts.

years ago the writer published an account of pyrometric de- ions of the reduction temperature of zinc oxides variously and of several kinds.[1] In said paper it was shown that the ure at which zinc oxide evolves zinc according to the equa-

$$ZnO + C \rightleftharpoons Zn + CO$$

upon the nature of the two reacting bodies. The determina- a range of accuracy of from 2° to 6° C., whereas differences were amount to an extreme of 90° C. So the differences are real must have a definite cause. The equation of boiling water,

$$(H_2O)_4 = 4\ (H_2O)$$

ed temperature for equilibrium of 100° C. at 760 mm. pres- n general, all equations reacting in the liquid or the gaseous ave well-defined constant physical conditions which govern ibrium point and reaction velocities.

eaction where we have one or more solid phases, the tempera- ditions requisite for the unbalancing of the stable system are vary. These variations are due to the fact that in a solid a very complex molecule and it is not a question solely of the way of two interlocked atoms, but also of the destruction of x molecule containing hundreds, or possibly thousands, of nolecules.

ner words, there are two forces to be overcome: a certain

[1] *Transactions of the American Electrochemical Society*, April 1904, vol. v., p. 211 (1904).

chemical force, $F_c$, and a certain "physical" force, $F_p$. The
cates that $F_c$ is constant and $F_p$ is variable. $F_p$ depends on
sical configuration of the small molecules of the metallic oxi

The writer's researches in this line have been confined to di
caused by heat treatment of zinc oxide and reducing agents.
zinc business we all know that roasted blendes of various kin
differently in retorting. Roasted blendes from one mine work

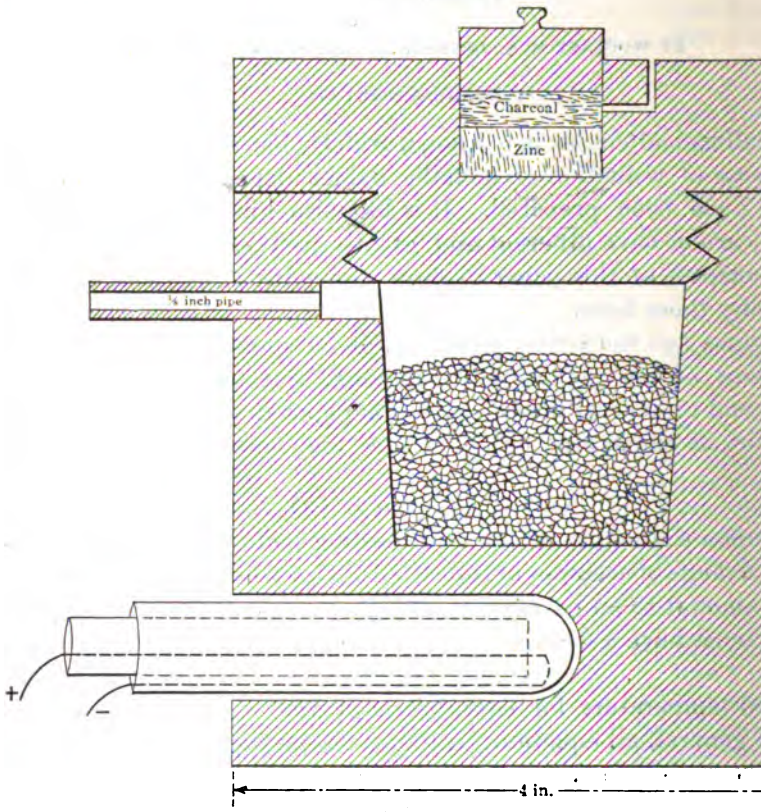


FIG. 1.—JOHNSON "REACTION TESTER."

at a low temperature, whereas roasted blendes from another m
of the same analysis and with the same reducing agent, wo
and slow even with high firing. As the furnaceman says, su
need a touch of "high life."

Fig. 1 shows the Johnson "reaction tester." This is made
son graphite electrode by machining 4-in. round stock in a lat
pyrometer is inserted in a hole very near the charge, so that
but a small temperature difference between it and the charg

of the reaction is very sharp. The only precaution necessary that the reaction, which starts at, say, 1,040° C., is active at C., energetic at 1,045° C. and violent at 1,050° C. As a mil- of zinc will give quite a flame, a false value of the sporadic of reagents, not typical of the mass, will result unless this on be used. To avoid misunderstanding let us suppose that 0 g. of zincy material in the tester, 5 mg. were of a soft na- ducible at 20° C. below the real reduction temperature of ; this would give a false value unless reaction proceeded to e extent and was well marked.

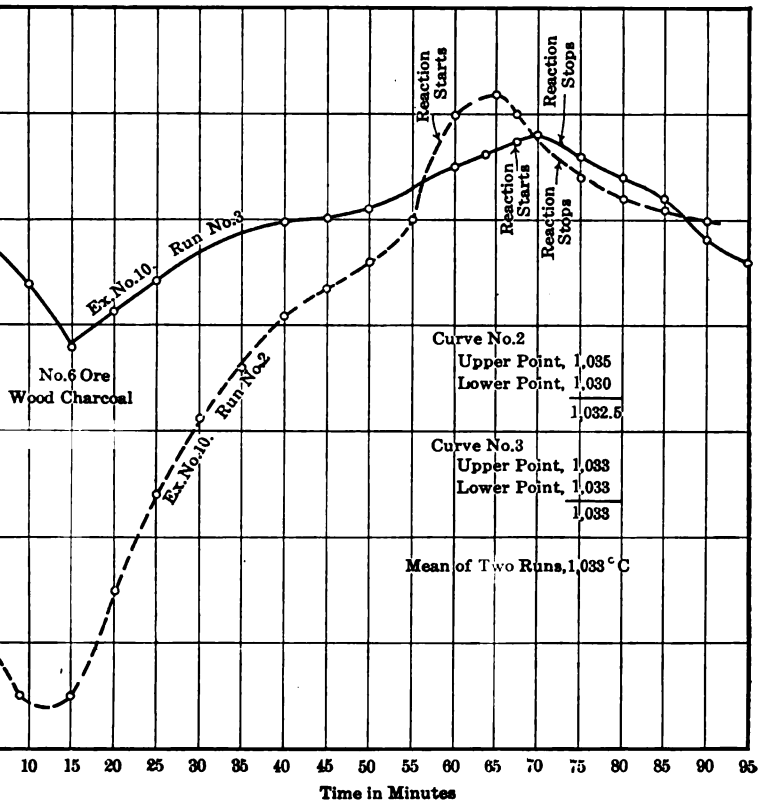


FIG. 2.—REDUCTION TEMPERATURES.

oling down, the reaction stops at nearly the same temperature it began. The mean of starting and ending is taken, and hree determinations of each test are made. With careful and increase of temperature the mean of means does not vary e extremes by over 2° C. A curve of a test better than the is shown in Fig. 2.

The pyrometer was checked by determining the 920° C. the boiling point of zinc. In the latter tests this was so that we could determine the differences in atmospheric pre changes in the boiling point. The zinc was put in a 0.5-in. h a plug in the tester and well covered with charcoal. Unless done the air will rush in at 900° C. and burn the zinc vapo exists there in quantities, making a partial vacuum by the fo of solid zinc oxide, and the boiling points will vary widely.

As the 920° C. point is taken with each determination, little chance for the pyrometer to get wrong.

Looking at Table I., we see that Test 102, with C.P. zin made by the wet method and soft 600° C. charcoal, gave a temperature of 1,022° C.; 48-hour Indian Territory coke ga the same substance in Test 101, a temperature of 1,029.5° C Acheson graphite turnings the reduction temperature was 1,

The same substance, C.P. zinc oxide, was treated by hea definite temperatures for 12 hours with soft coke and its temperature was determined. The three values of C.P. wet- zinc oxide are as follows:

| | Reduction Temperatur |
|---|---|
| No previous treatment | 1,029.5° C. |
| 1,100° C. treatment | 1,048° C. |
| 1,300° C. treatment | 1,061° C. |

The effect of the different reducing agents is shown as follo

*C.P. Cadmium Oxide.*

| | |
|---|---|
| Wood charcoal burnt at 600° C. | 767° C. |
| Wood charcoal burnt at 1,100° C. | 772° C. |
| Indian Territory coke | 813° C. |
| Acheson graphite turnings | 849° C. |

Referring to Table I., it will be seen that this law can be d *The higher either of the reacting bodies is previously heated, the high reduction temperature.*

The fact that the values are concordant with this law, and that the reductivity of the zinc charge varies with the previo ment, as determined by measuring the reaction velocity acco another method of the writer's, shows that the heat treatmen reacting bodies modifies the physical-chemical nature profoun

Exactly how the zinc oxide parts with its oxygen and exac the oxygen gets to the carbon of the reducing agent is a myst no more a mystery than any reaction between solid particles. ural assumption is that zinc oxide has a slight vapor pressure

de heated to a high temperature in a crucible will slowly
e), that carbon has also an infinitesimal vapor pressure, and
n the sum of these vapor pressures equals a certain figure,
oxide dissociates into oxygen, which combines with the car-
metallic zinc. Carbon monoxide is formed and both gases
ed. This speculation, if continued, would lead us into the
f molecular physics; but that which the writer has given,
heoretical in its nature, has a strict practical bearing. There
n zinc metallurgists who have the opinion that zinc oxide re-
goes into two stages.

$$\text{I.} \qquad ZnO + CO = Zn + CO_2.$$
$$\text{II.} \qquad CO_2 + C = 2\,CO.$$

his view the writer is unable to agree. It is of course plain
will reduce zinc oxide, for many experiments, including the
have proved this, but not to zinc directly to any extent, for
ion

$$ZnO + CO \rightleftharpoons Zn + CO_2$$

ble and $CO_2$ can exist only in small amounts at 1,100° C.—
t 1 per cent.—without oxidizing zinc vapor. The direct
"push," or the atomic kick that starts the reaction and
going, comes from the carbon molecule. The fact that varia-
he physical nature of the reducing agent make a difference
C. in the critical temperature at which this kick becomes
indicates that this kick comes from the carbon molecule, and
the secondarily-generated carbon monoxide.

riter does not argue that the reactions I. and II. do not occur,
would be as foolish as to say that it is impossible to go to Bos-
pt on the trains of the New Haven railroad. It would of
possible to take the New York Central train to Albany and
Boston and Albany to Boston, but it would cost more. And
oes not waste enery in an endothermic reaction. The writer
h regard for this endothermic reaction:

$$ZnO + C = Zn + CO.$$

tion takes the path of least resistance. The fact that reduc-
eeds very much faster in the writer's electric zinc-smelting
when only enough carbon for reduction is added, than in re-
ice would seem to dismiss most of the importance attached
ons I. and II.

nclusion follows that, when the temperature of the reacting
considerably above the initial reduction temperature, the di-
nical path is so crowded that the double reaction I. and II.
to some extent: When the New Haven railroad trains are
would go to Boston via Albany.

Taking up the question of the changes in the nature of tl oxide as affecting reduction temperature, it might be well to c zinc ferrite, zinc silicate, and zinc aluminate. All these hav made by the writer by heating $Fe_2O_3$, $SiO_2$, and $Al_2O_3$ mixe molecular equivalents of zinc oxide. They are insoluble in nium chloride and dilute acids, and, if strongly heated, insol concentrated acids. They are also inert to reduction by carb have high reduction temperatures.

We can, however, derive a useful lesson from these comp For, *if zinc oxide can combine with metallic oxides as aluminum ox iron oxides, why can it not combine with another metallic oxide: to oxide?* Accordingly, we can say that, when zinc ores are heat strongly, "zinc zincate" is formed, and that the energy of th pound must be supplied to break it up before the reaction ca ceed.

The writer has found that these experimental facts agreed w practical experience of himself and many others in the zinc bu for instance, it is a fact that "spiegel oxide," which is formed downcomers and in the hot-blast stoves of the New Jersey Zin spiegeleisen furnaces working on residues from the "oxide" fu and which is thus made at a low temperature, requires little "retorting," whereas "refuse oxide," made in the Wetherill fu of the same company at a temperature vastly higher, requires higher temperature to "retort." "Dead" coal, or coal from th pits of western Missouri, has been known to reduce zinc oxide more easily than Arkansas semi-anthracite, as the tests show natural to expect that the weathering action makes the carbon active.

The tests show, of course, only the initial temperature of re and say nothing about the rate of reduction. About 150 tes made to determine this value by heating different mixes for a time and determining reduction by analyzing the mix before an test. With equivalent amount of fixed carbon and ash in the r rate of reaction is roughly proportional to the initial temperat statical point. This is in accordance with theory.

Before any chemical action takes place in the highly endot reaction, the reduction of zinc oxide by solid carbon, a definit mochemical potential must first be obtained. Of course, in reaction the free energy determines the velocity of reaction acc to the Gibb-Helmholz equation.

The chemical potential of the reverse reaction is higher tha of reduction before this thermochemical potential is attained

ıce of potential is a function of the difference of temperature n the "reaction temperature" and the temperature of the re- bodies. Thus, should we have the reduction temperature at C. and the function be a linear one, then, when we have the g bodies at 1,100° C., the reaction velocity will be five times han it would be should the reacting bodies be at 1,050° C.

n Ohm's law Amperes $= \frac{\text{Volts}}{\text{Ohms}}$, so Professor Nernst has shown n like manner, the rate of chemical change is equal to $\frac{\text{mical force}}{\text{cal resistance.}}$ The rate at which zinc is reduced in a retort is inable should we know the chemical force or chemical poten- cause then the reaction velocity can be found by means of the n given above.

ll heat treatment the question of time is important, for the on of the "zinc zincate" proceeds very slowly. The heating ıally for 12 hours, or over night.

ıe classical work of the late Sir Lowthian Bell, *Chemical Phe- of Iron Smelting*, on p. 17, *et seq.*, it will be seen that he experi- l with iron oxides. The substances he treated were spread out ighed dish so as to secure free access to the gas. The following were obtained by him by exposing the substances enumerated to the action of pure CO for 7 hours. The temperatures were ed by the use of pure lead, zinc, and antimony test pieces.

| Exposure. | | Percentage of Original Oxygen Removed. |
|---|---|---|
| 26. Calcined Cleveland ore to | CO at 417° C. | 9.4 |
| 27. Pumice stone with $Fe_2O_3$ | CO at 417° C. | 23.8 |
| 28. Calcined $FeSO_4$ ($Fe_2O_3$] | CO at 417° C. | 61.7 |
| 29. $Fe_2O_3$ precipitated | CO at 417° C. | 66.7 |
| 30. $Fe_2O_3$ from nitrate | CO at 417° C. | 72.7 |
| Exposure for 7.5 hours at the temperature of softened zinc (410° C.): | | |
| 31. Lancashire hematite, | 40.3 per cent. of Fe | 37.1 |
| 32. $Fe_2O_3$ from nitrate | | 59.9 |
| Exposure for 7.5 hours at the temperature of softened zinc (410° C.): | | |
| 33. Calcined Cleveland ore, | 40 per cent. of Fe | 20.7 |
| 34. Calcined spathose ore, | 51.7 per cent. of Fe | 28.4 |
| 35. Lancashire hematite ore, | 40.3 per cent. of Fe | 57.4 |
| Exposure for 6 hours at a temperature of about 410° C.: | | |
| 36. Calcined Cleveland ore, | 40 per cent. of Fe | 37.3 |
| 37. Lancashire hematite, | 66.6 per cent. of Fe | 35.9 |
| 38. Elba specular ore, | 66.8 per cent. of Fe | 16.8 |
| 39. Calcined spathose ore, | 51.7 per cent. of Fe | 15.4 |
| 40. Pure $Fe_2O_3$, precipitated, | 70.0 per cent. of Fe | 49.2 |
| 41. Pure $Fe_2O_3$, from calcined $FeSO_4$ | 70.0 per cent. of Fe | 60.8 |

and on p. 48 of the same work, by treating with CO gas:)

| | C Per 100 of Stuff. | C Per 100 |
|---|---|---|
| Exp. 201. $Fe_2O_3$ from the nitrate | 56.3 | 144. |
| Exp. 202. Precipitated $Fe_2O_3$ | 45.5 | 95. |
| Exp. 203. $Fe_2O_3$ from calcined $FeSO_4$ | 31.9 | 54. |
| Exp. 204. $Fe_2O_3$ from calcined $FeSO_4$ on pumice-stone, | 1.69 | 14. |
| Exp. 205. Calcined Cleveland ore | 0.11 | 0. |

Exposure for 7 hours at a temperature not above 420° C.

We see that the effect of heat treatment in each of these c to decrease the chemical activity. The writer's research was to see whether or not the same thing happened with zinc ox Bell found with iron oxides.

About the most remarkable case was seen by the writer at th oratory of F. A. J. Fitzgerald: Lime was fused in the electri nace, and neither water nor acids had any effect on the samples hours, and in 6 or 7 months the lime was not slaked. Here the of heat treatment on the chemical activity of a solid was so m as to constitute a chemical joke. Electrically sintered magnes the Norton Co. shows the same effect.

Every item of experimental evidence and theoretical specu indicates that the reducibility of zinc oxide, or any other m oxide, depends on the way in which the reducing agent and th terial to be reduced are heat treated. Practically we find this the case in retorting and also in smelting in the writer's electri nace. In the latter the reactions proceed in an entirely different ner, for the alternating current is a catalyzer and the slag smelting zone introduces new features besides, and only enoug bon is added to reduce the zinc oxide.

Since researches prove that such practical changes as roasting low a temperature as will give a "dead" roast increase the e retorting to such an extent that it is directly and distinctly se to the workman, it is but logical to reason conversely and to att to the practical judgment of the workman a substantial deg accuracy. Now, any practical zinc man knows that ores fro mine will "retort" easily and ores from another will "retort" s So it is hardly to be doubted that an investigation of the tion temperature of zinc blende roasted under standard tions, and with a standard reducing agent, in a Johnson "re tester" would determine the geological history of such mines. writer knows little about geology, but he is sure that blende dep primarily would give when roasted a reduction temperature 1 and possibly 20° C., higher than a blende deposited in the same secondarily and tested under same conditions.

| No. | Material to be Reduced. | Reducing Material. | Mean. Degrees C. | Mean. Degrees C. | Mean. Degrees C. | Means. Degrees. C. | Remarks. |
|---|---|---|---|---|---|---|---|
| 10 | Joplin ore | Common charcoal, Joplin, Mo | 1,032.5 | 1,033 | | 1,033 | All stuff finer than 20 mesh |
| 56 | Joplin ore | Soft burned charcoal, 600° | 1,049 | 1,048 | | 1,048 | |
| 57 | Joplin ore | Indian Territory coke | 1,063 | 1,060 | | 1,062 | |
| 24 | Joplin ore | Indian Territory coke | 1,066 | 1,065 | | 1,066 | |
| 55 | Joplin ore | Hard burned charcoal, 1,100° | 1,060 | 1,068 | 1,070 | 1,068.5 | |
| 22 | Joplin ore | Indian Territory coke | 1,072 | 1,080 | | 1,076 | |
| 26 | Joplin ore | Acheson graphite | 1,088 | 1,085 | | 1,087 | |
| 29 | Joplin ore | Acheson graphite turnings | 1,103 | 1,107 | | 1,105 | |
| 109 | Roasted Joplin ore, 900° C | Arkansas semi anthracite | 1,006 | 1,014 | | 1,010 | |
| 110 | Roasted Joplin ore, 900° C | Dead coal, 1,400° C | 1,021 | 1,021 | | 1,021 | |
| 114 | Roasted Joplin ore, 900° C | Wood charcoal | 1,021 | 1,024 | | 1,023 | |
| 115 | Roasted Joplin ore, 900° C | Indian Territory coke | 1,049 | 1,058 | | 1,054 | |
| 116 | Roasted Joplin ore, 900° C | Graphite | 1,106 | 1,106 | | 1,105.6 | |
| 28 | Colorado ore | Indian Territory coke | 1,003 | 1,010 | | 1,007 | |
| 27 | Colorado ore | Indian Territory coke | 1,017 | 1,020 | | 1,019 | |
| 29 | Colorado ore | Soft charcoal | 1,042.5 | 1,065 | 1,065 | 1,065.4 | |
| 30 | Colorado ore | Acheson graphite turnings | 1,108 | 1,111 | 1,107 | 1,108.3 | |
| 120 | Roasted Colorado ore, 900° | Wood charcoal | 1,014 | 1,019 | | 1,017 | |
| 121 | Roasted Colorado ore, 900° | Indian Territory coke | 1,029 | 1,021 | | 1,025 | |
| 111 | Roasted Colorado ore, 900° | Arkansas semi-anthracite | 1,026 | 1,033 | | 1,030 | |
| 122 | Roasted Colorado ore, 900° | Acheson graphite turnings | 1,081 | 1,061 | | 1,071 | |
| 102 | C. P. zinc oxide, wet method | Wood charcoal, soft, 600° C | 1,022 | 1,024 | 1,020 | 1,022 | |
| 101 | C. P. zinc oxide, wet method | Indian Territory coke | 1,029 | 1,030 | | 1,029.5 | |
| 104 | C. P. zinc oxide, wet method | Acheson graphite turnings | 1,082 | 1,085 | 1,085 | 1,085 | |
| 107 | C. P. ZnO, calcined at 1,100° | Indian Territory coke | 1,049 | 1,046 | 1,048 | 1,048 | |
| 108 | C. P. ZnO, calcined at 1,300° | Wood charcoal | 1,024 | 1,025 | | 1,025 | |
| 106 | C. P. ZnO, calcined at 1,300° | Indian Territory coke | 1,060 | 1,061 | | 1,061 | |
| 113 | C. P. ZnO, calcined at 1,400° | Dead coal | 1,019 | 1,025 | | 1,022 | |
| 112 | C. P. ZnO, calcined at 1,400° | Arkansas semi-anthracite | 1,054 | 1,056 | | 1,055 | |
| 75 | Boiling point of cadmium | | 759 | 757 | 754<br>753 | 756 | Henry Heil & Co. |
| 80 | C. P. cadmium oxide, wet method | Wood charcoal, 1,100° C | 772 | 770 | 774 | 772 | |
| 76 | C. P. cadmium oxide, wet method | Indian Territory coke | 798 | 828 | | 813 | |
| 87 | C. P. cadmium oxide, wet method | Indian Territory coke | 813 | | | 813 | |
| 81 | C. P. cadmium oxide, wet method | Acheson graphite turnings | | 848 | 850 | 819 | |
| 21 | $CaSO_4$ | Common charcoal, Joplin, Mo | 870 | 865 | | 868 | By CO flame |
| 61 | Joplin ore | Iron filings | 1,105 | 1,105 | | 1,105 | False, graphite reacted. |
| 62 | Joplin ore | Iron filings | 1,182 | | | 1,182 | Fire clay tester |
| 64 | Joplin blende | Iron filings | 1,187 | 1,188 | | 1,188 | Fire clay tester |

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# Roasting and Leaching Tailings at Anaconda, Mont.

BY FREDERICK LAIST, ANACONDA, MONT.

(Butte Meeting, August, 1913.)

E remodeling No. 1 section of the concentrator at the Washoe on Works of the Anaconda Copper Mining Co. during the of 1912, for the purpose of ascertaining what additional ecoveries could be made by improvements in concentration which had been developed by the staff of the Boston & Mon- duction Department, at Great Falls, it was deemed advisable on some experiments on the treatment of the regular mill by roasting and leaching, to see whether the losses of metal ight not be still further decreased.

e course of these experiments about 5,000 tons of tailings were and of the resulting calcine about 200 tons were leached. ults were so satisfactory that it was decided to continue the a larger scale, particularly as regards the leaching end of ess, and construction work on an 80-ton roasting and leach- t was accordingly commenced during the latter part of Feb- 913, and is being pushed as rapidly as possible.

## *80-Ton Leaching Plant.*

plant will consist of one 20-ft. roasting furnace of the Mac- type, specially adapted to the work of roasting for leaching; ching tanks, 32 ft. in diameter by 12 ft. deep, together with essary solution tanks, air lifts, precipitating launders, etc.

been decided to precipitate the copper from solution on scrap as not to hamper the roasting and leaching departments by kness that might develop in the precipitating department. wever, the intention to experiment with a number of different of precipitation; for example, by electrolysis and by hydro- hide.

cid required for the operation of this plant will be purchased, quantity required is not sufficient to justify the installation of plant. An acid plant will, however, be established in con-

nection with any large units that may be erected, the sulphur for the acid plant being obtained by the roasting of fine concent The commercial success of the scheme in question, for the treat of tailings or low-grade material of any kind, depends on being to secure cheap acid. Where operations are conducted on a sc several thousand tons per day, a large acid plant will be neede the maximum economy in the manufacture of acid can be rea Given roasting-furnace gases containing 6 per cent. or more of the cost of making acid should not exceed $4 per ton of 60° Bé.

The plan and elevation of the 80-ton leaching plant are show Figs. 1 and 2. The general arrangement does not differ mate from a cyanide plant for the treatment of gold ores. The plant be used for the treatment of sand and slime tailings, which wi mixed in the proper proportions before entering the roasting fur Experiments have shown that a mixture of four parts sand tai and one part slime tailings percolates without difficulty after ing, the percolation rate after roasting being about three times a as before roasting, due to destruction of colloid in the furnace.

Fig. 3 shows a sectional elevation of the 20-ft. roasting fu which is being constructed. The furnace is, in general, a six-h furnace of the MacDougall type. A seventh water-jacketed floo been added for cooling the calcine preliminary to dropping it the belt conveyers leading to the leaching tanks.

There are two fire boxes, the flames from which pass over the floor from the top. On the upper three floors the tailings are p roasted and brought to a temperature of about 1,000° F., at w temperature they drop on to the fourth floor, where about 1 per of salt is added. During its passage over the fourth, fifth, and floors the remaining copper, as well as silver, is chloridized, the in the ore, which is kept from dissipating as much as possible, sufficient for the reactions. A very small volume of air is d through the chloridizing hearths of the furnace by means of which exhausts into an absorption tower so as to catch any copp silver which may have volatilized. The furnace is expected to a capacity of at least 80 tons per day, on our material, and sl produce somewhat better results, metallurgically, than the 16-ft. Dougall in which the preliminary experiments were made.

*Roasting Experiments in MacDougall No. 64.*

This furnace was one of our regular MacDougall furnaces (No which we equipped with two fire boxes. These had grates 3 2 ft. 6 in., and were arranged so that the flames could be ma

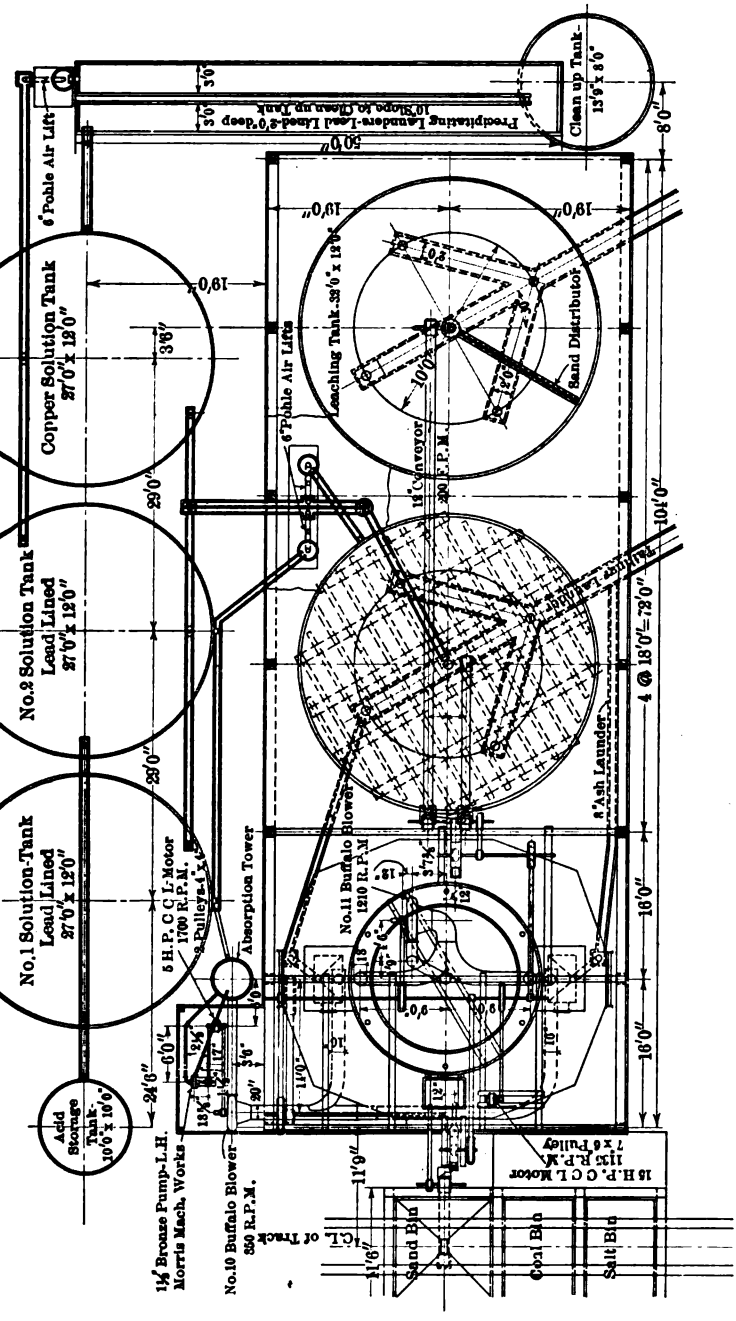
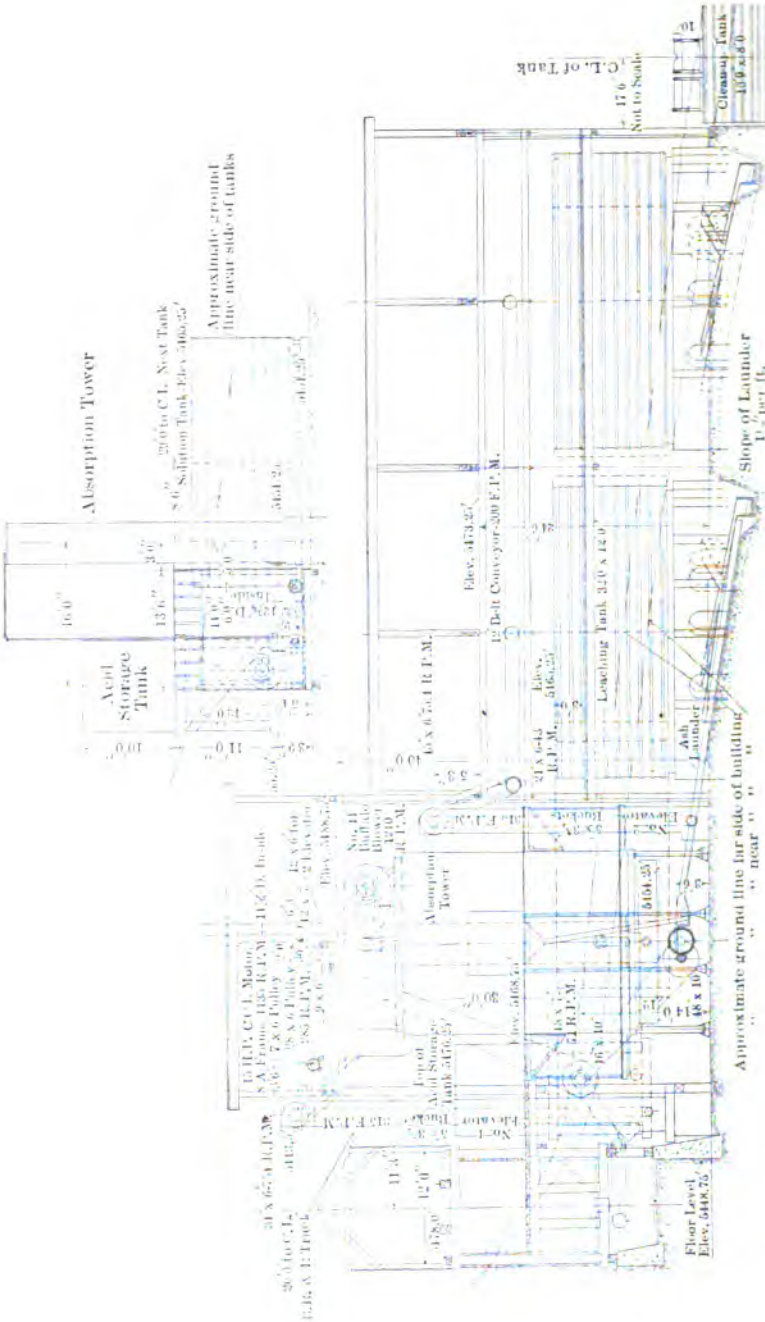



FIG. 1.—PLAN OF 80-TON LEACHING PLANT AT WASHOE REDUCTION WORKS, ANACONDA, MONT.


Absorption Tower
Acid Storage Tank
Approximate ground line near side of tanks
Leaching Tank
Ash Launder
Slope of Launder
Approximate ground line far side of building
Floor Level
Not to Scale
Cleanup Tank

n either the third or the fourth floor from the top. Numerous tions were made to ascertain which was the better arrange- nd it was finally decided that the third floor was better. When nace was put on oxy-chloride roasting this was the only floor

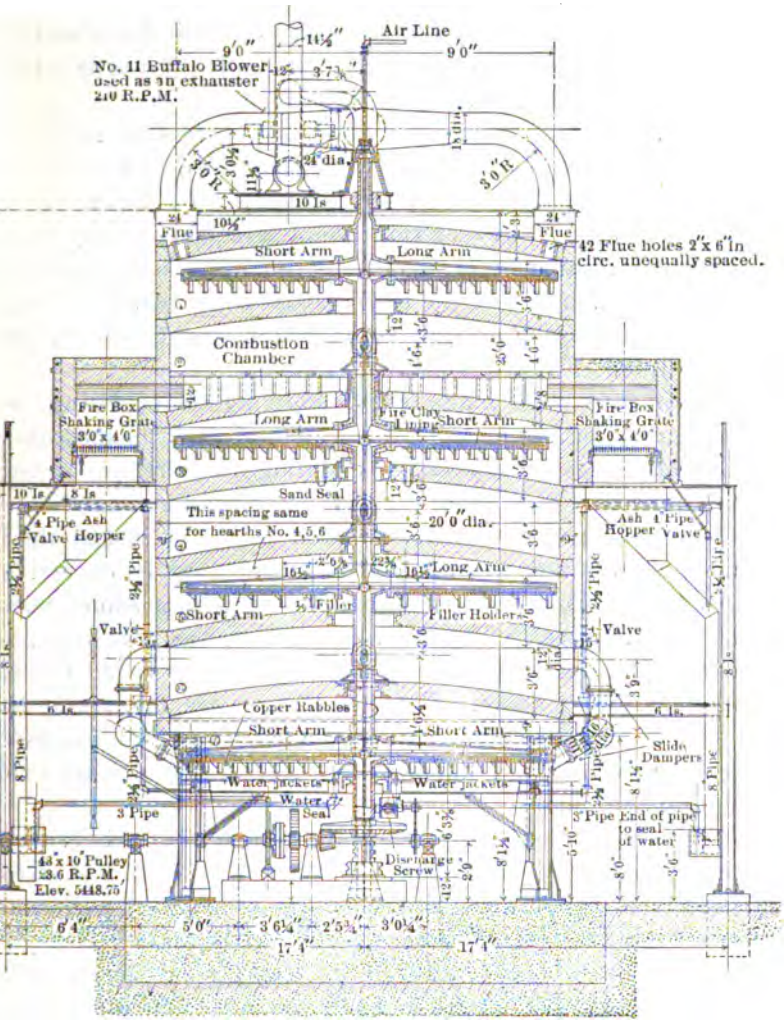



FIG. 3.—SECTIONAL ELEVATION OF 20-FT. ROASTING FURNACE.

uld be considered. Experiments were also made with different for the rabble arms. The highest speed tried was one revolu- 23 sec.; the slowest, one revolution in 60 sec. Most of the vas done with one revolution in 37 sec., but one revolution in

45 sec. gave just about the same results. Too high a speed the material through the furnace too rapidly and does not gi cient time for oxidation. Too slow a speed causes too thick calcine on the floors and does not permit the heat to penetrate

When the furnace was used for oxy-chloride roasting, th were removed from the short arm on each of the three lowe This caused a much thicker bed to form on these floors and ga one-half as much stirring, all of which tended towards a bet servation of heat and better chloridizing.

The arms of the furnace and the shaft were air-cooled, ex the first and sixth floors, which were not cooled at all.

The furnace operated smoothly and very little labor was on it, no barring off of accretions being necessary at any time firing was regulated by means of a pyrometer inserted over th floor. It was found that the temperature could readily be kep a few degrees of the desired point, which was 1,200° F. for ing and 1,000° F. for oxy-chloride roasting.

Experiments were made with oxidizing, chloridizing, a chloride roasting. The last-named method gave the most sati results, all things considered. The straight oxidizing was qui factory, but only one-half of the silver was rendered soluble b the copper extraction was not so good as by the last method straight chloridizing was not satisfactory, owing to excessive copper by volatilization and difficulty of catching same whe with fire box gases. The salt consumption was, moreover, t

In carrying out the oxy-chloride roasting, the upper three serve as oxidizing floors. Here the tailings are dried and he the proper temperature. All of the extra atom of sulphur $FeS_2$ is burned off and much of the FeS and $Cu_2S$ are oxi oxides and sulphates.

When the ore drops from the third floor it is in excellent c for chloridizing, which is done on the fourth, fifth, and sixth by means of salt, which, to the extent of about 1 per cent weight of calcine, is introduced on the fourth floor. Very li is required here, because most of the sulphur has been expell no firing is needed on these floors, because they are well i and are kept hot by the heated ore constantly coming down volatilized copper is small in amount, because most of the C been converted into CuO before coming in contact with the is, therefore, not chloridized, and whatever is volatilized easily caught by means of absorption towers, through whic pass only the small volume of gas from the lower floors.

The results obtained are given in Tables I., II., and III.

BLE I.—*Test Period No.* 1, *Oxidizing Roast on Mill Tailings.*

**Period of Test, June 4 to June 14, inclusive.**

| 1912. | Feed. Wet Weight in Tons. | Feed. Dry Weight in Tons. | Calcine. Dry Weight in Tons. | Coal. Wet Weight in Tons. | Copper Remaining in Tailings After Leaching. As $Cu_2S$. Lb. Per Ton. | As Ferrite. Lb. Per Ton. | As Total. Lb. Per Ton. | Silver in Tailings After Leaching. | $Fe_2O_3 + Al_2O_3$ Dissolved. Per Cent. | Acid Consumption. Lb. $H_2SO_4$ Per Ton. | Copper in Feed. Lb. Per Ton. | Copper in Calcine. Lb. Per Ton. | Moisture in Feed. Per Cent. |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| June. | | | | | | | | | | | | | |
| 4 | 74.3 | 69.0 | 67.6 | 2.40 | 1.50 | 0.44 | 1.94 | 0.26 | 0.77 | 41.3 | 12.4 | 13.0 | 7.14 |
| 5 | 63.8 | 60.0 | 69.3 | 2.56 | 2.56 | 0.24 | 2.80 | 0.17 | 1.00 | 45.0 | 12.6 | 12.8 | 6.00 |
| 6 | 72.3 | 67.6 | 57.3 | 2.32 | 1.56 | 0.22 | 1.78 | 0.21 | 0.70 | 41.9 | 12.6 | 13.2 | 6.50 |
| 7 | 70.9 | 65.9 | 57.6 | 2.34 | 1.00 | 1.00 | 2.00 | 0.23 | 0.83 | 40.2 | 12.4 | 12.8 | 7.06 |
| 8 | 53.9 | 50.6 | 56.1 | 2.26 | 1.20 | 0.80 | 2.00 | 0.29 | 0.55 | 30.0 | 12.8 | 12.8 | 6.11 |
| 9 | 73.3 | 68.3 | 66.3 | 2.37 | 1.78 | 0.92 | 2.70 | 0.22 | 0.72 | 35.0 | 12.6 | 13.2 | 6.79 |
| 10 | 68.4 | 63.6 | 66.0 | 2.72 | 2.44 | 0.30 | 2.74 | 0.16 | 0.73 | 36.0 | 11.8 | 12.0 | 6.95 |
| 11 | 73.1 | 68.4 | 66.9 | 2.58 | 1.60 | 0.62 | 2.22 | 0.26 | 0.58 | 35.0 | 12.2 | 12.6 | 6.50 |
| 12 | 65.6 | 61.4 | 58.9 | 2.32 | 1.20 | 0.38 | 1.58 | 0.28 | 0.75 | 40.8 | 12.0 | 12.6 | 6.50 |
| 13 | 53.0 | 49.5 | 57.8 | 2.26 | 0.84 | 0.58 | 1.42 | 0.24 | 0.70 | 39.0 | 11.8 | 11.8 | 6.50 |
| 14 | 73.7 | 69.0 | 66.1 | 2.22 | 1.12 | 0.50 | 1.62 | 0.26 | 0.53 | 36.0 | 11.2 | 11.4 | 6.50 |
| ..... | 742.3 | 683.3 | 689.9 | 26.35 | | | | | | | | | |
| ..... | 67.5 | 62.1 | 62.7 | 2.41 | 1.53 | 0.55 | 2.07 | 0.23 | 0.71 | 38.2 | 12.2 | 12.5 | 6.60 |

SUMMARY.

| | |
|---|---|
| ds copper contained in feed | 7,600 |
| ds copper contained in calcine | 7,850 |
| age | 250 |
| entage of coal used to wet weight of feed | 3.57 |
| entage of coal used to dry weight of feed | 3.88 |
| entage of copper lost in tailings | 16.50 |
| entage of copper recoverable | 83.50 |
| entage of silver lost in tailings | 49.10 |
| entage of silver recoverable | 50.90 |
| of coal per ton of tailings (dry weight). (Diamondville coal at $1.90 per ton) | $0.190 |
| of acid per ton of tailings (60° Bé. acid at $3.51 per ton = $4.18 per ton $H_2SO_4$) | $0.082 |
| e of silver recoverable per ton of tailings | $0.163 |

ng the months of July and August, during which two of the ns and practically all of the calcine for the leaching plant were our concentrator tailings assayed lower in copper than normally ut 0.05 per cent., or 1 lb. per ton.

e is no trouble whatever in maintaining a perfectly uniform provided a uniform feed is maintained. Occasional "bad are generally due to the feeder failing to act properly and g the tailings to "run."

ll be noted that the flue-dust production was remarkably small, ly having been less than 2 per cent. Even when a mixture of tailings and 1 part slime was roasted, the weights of feed and checked each other almost exactly after allowances were made of sulphur, weight of salt, etc., showing that the flue-dust pro- must have been very small.

TABLE II.—*Test Period No. 2, Oxidizing Roast on Mill Tailings.*

**Period of Test, July 8 to July 31, inclusive.**

| Date, 1912. | Feed. Wet Weight in Tons. | Feed. Dry Weight in Tons. | Calcine. Dry Weight in Tons. | Coal. Wet Weight in Tons. | Copper Remaining in Tailings After Leaching. As $Cu_2S$. Lb. Per Ton. | As Ferrite. Lb. Per Ton. | As Total. Lb. Per Ton. | Oz. Per Ton Silver in Tailings After Leaching. | $Fe_2O_3 + Al_2O_3$ Dissolved. Per Cent. | Acid Consumption. Lb. $H_2SO_4$ Per Ton. | Copper in Feed. Lb. Per Ton. | Copper in Calcine. Lb. Per Ton. | Moisture in Feed. Per Cent. |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| July. | | | | | | | | | | | | | |
| 8 | 62.7 | 58.8 | 50.6 | 2.54 | 1.43 | 0.12 | 1.55 | 0.22 | 0.75 | 43.3 | 11.2 | 10.4 | 6.25 |
| 9 | 62.7 | 58.5 | 70.3 | 2.33 | 1.10 | 0.76 | 1.86 | 0.22 | 0.61 | 42.6 | 10.8 | 11.6 | 6.70 |
| 10 | 82.6 | 76.8 | 60.3 | 2.39 | 1.00 | 0.68 | 1.68 | 0.21 | 0.70 | 43.0 | 10.8 | 11.2 | 6.97 |
| 11 | 63.2 | 59.0 | 61.5 | 2.39 | 1.44 | 0.96 | 2.40 | 0.24 | 0.75 | 38.2 | 12.2 | 11.8 | 6.62 |
| 12 | 75.5 | 70.5 | 64.0 | 2.37 | 1.56 | 1.30 | 2.86 | 0.18 | 0.75 | 38.0 | 11.8 | 13.0 | 6.62 |
| 13 | 75.0 | 70.5 | 65.7 | 2.39 | 2.20 | 0.74 | 2.94 | 0.14 | 0.83 | 36.5 | 11.8 | 12.8 | 6.00 |
| 14 | 64.7 | 61.5 | 60.4 | 2.31 | 1.60 | 0.76 | 2.36 | 0.15 | 0.95 | 39.5 | 10.8 | 11.6 | 5.00 |
| 15 | 75.8 | 72.3 | 67.2 | 2.52 | 2.46 | 0.72 | 3.18 | 0.21 | 0.95 | 39.0 | 11.0 | 11.6 | 4.59 |
| 19 | 65.0 | 59.7 | 49.4 | 2.39 | 1.86 | 0.74 | 2.60 | 0.34 | 0.79 | 44.0 | 11.6 | 12.4 | 8.20 |
| 23 | 78.6 | 75.1 | 70.4 | 2.80 | 2.00 | 0.94 | 2.94 | 0.22 | 0.76 | 39.0 | 10.8 | 11.6 | 4.59 |
| 24 | 75.2 | 71.9 | 59.8 | 2.40 | 1.80 | 0.98 | 2.78 | 0.26 | 0.82 | 39.0 | 11.8 | 12.8 | 4.37 |
| 25 | 66.1 | 63.3 | 70.2 | 2.29 | 1.50 | 1.02 | 2.52 | 0.24 | 0.75 | 37.0 | 12.6 | 14.0 | 4.30 |
| 26 | 51.2 | 48.7 | 39.4 | 2.16 | 1.44 | 0.90 | 2.34 | 0.28 | 0.76 | 42.8 | 12.2 | 14.0 | 4.90 |
| 27 | 43.4 | 41.3 | 48.2 | 2.14 | 1.30 | 0.88 | 2.18 | 0.27 | 0.90 | 45.5 | 11.4 | 12.2 | 4.83 |
| 28 | 55.2 | 52.0 | 39.3 | 2.22 | 1.20 | 0.82 | 2.02 | 0.23 | 0.78 | 45.0 | 10.4 | 11.8 | 5.71 |
| 29 | 65.2 | 62.2 | 70.2 | 2.25 | 1.74 | 0.52 | 2.26 | 0.23 | 0.79 | 38.0 | 11.2 | 11.6 | 4.59 |
| 30 | 61.2 | 58.7 | 48.4 | 2.11 | 1.20 | 0.80 | 2.00 | 0.25 | 0.58 | 40.8 | 10.6 | 12.2 | 3.99 |
| 31 | 64.6 | 60.4 | 62.8 | 2.07 | 1.76 | 0.62 | 2.42 | 0.24 | 0.80 | 36.7 | 12.0 | 12.2 | 6.51 |
| Total | 1,187.9 | 1,121.2 | 1,058.1 | 42.07 | | | | | | | | | |
| Averages | 65.9 | 62.3 | 58.8 | 2.34 | 1.59 | 0.79 | 2.28 | 0.23 | 0.78 | 40.4 | 11.4 | 12.2 | 5.58 |

REMARKS.—Furnace down from 16th to 17th, changing flame from 4th to 3d floor. Heating on 18th. Furnace down on 20th and 21st due to cave in flue.

SUMMARY.

| | |
|---|---|
| Pounds copper contained in feed | 12,800 |
| Pounds copper contained in calcine | 12,900 |
| Overage | 100 |
| Percentage of coal used to wet weight of feed | 3.55 |
| Percentage of coal used to dry weight of feed | 3.75 |
| Percentage of copper lost in tailings | 19.5 |
| Percentage of silver lost in tailings | 44.2 |
| Percentage of copper recoverable | 80.5 |
| Percentage of silver recoverable | 55.8 |
| Percentage of copper in calcine | 100.8 |
| Cost of coal per ton of tailings (dry weight). (Diamondville coal at $4.90 per ton) | $0.181 |
| Cost of acid per ton of tailings (60° Bé. acid at $3.45 per ton = $4.48 per ton $H_2SO_4$) | $0.086 |
| Value of silver recoverable per ton of tailings | $0.176 |

It is of the greatest importance in roasting for leaching to avoid overheating of the ore, since to do so means to render insoluble a portion of the copper. According to Professor Hofman, this copper exists as ferrite. It is insoluble not only in the leaching solution, but also in concentrated nitric and hydrochloric acids, which makes its presence in the tailings very easy to overlook if care be not taken to effect the decomposition with hydrofluoric acid.

Following were the temperatures on the various floors of Furnace No. 64, and in flues and hopper:

TABLE III.—*Test Period No. 3, Oxide-Chloride Roast on Mill Tailings.*

Period of Test, Aug. 2 to Aug. 29, inclusive.

| Date, 1912. | Feed. Wet Weight in Tons. | Feed. Dry Weight in Tons. | Calcine. Dry Weight in Tons. | Coal. Wet Weight in Tons. | Copper Remaining in Tailings after Leaching: As $Cu_2S$. Lb. Per Ton. | As Ferrite. Lb. Per Ton. | As Total. Lb. Per Ton. | Oz. Silver Per Ton in Tailings after Leaching | $Fe_2O_3+Al_2O_3$ Dissolved. Per Cent. | Acid consumption, Lb. $H_2SO_4$ Per Ton. | Copper in Feed. Lb. Per Ton. | Copper in Calcine. Lb. Per Ton. | Moisture in Feed. Per Cent. |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| August. | | | | | | | | | | | | | |
| 2 | 74.9 | 70.7 | 61.2 | 2.10 | 1.30 | 0.95 | 2.25 | 0.08 | 0.78 | 45.0 | 10.4 | 11.0 | 5.59 |
| 3 | 43.7 | 41.1 | 51.8 | 1.96 | 1.26 | 0.94 | 2.20 | 0.06 | 0.94 | 44.0 | 10.8 | 9.6 | 5.84 |
| 4 | 74.6 | 70.3 | 61.9 | 2.15 | 1.30 | 0.88 | 2.18 | 0.04 | 0.79 | 42.5 | 10.6 | 9.6 | 5.75 |
| 5 | 65.5 | 61.5 | 58.6 | 2.14 | 1.06 | 0.62 | 1.68 | 0.03 | 0.80 | 38.5 | 11.0 | 9.8 | 6.00 |
| 6 | 64.2 | 60.7 | 67.2 | 1.98 | 1.10 | 0.38 | 1.48 | 0.03 | 0.75 | 39.0 | 10.4 | 10.0 | 5.45 |
| 7 | 63.7 | 60.5 | 58.4 | 1.92 | 1.30 | 0.10 | 1.40 | 0.03 | 0.77 | 42.0 | 11.6 | 10.6 | 4.95 |
| 8 | 64.3 | 61.1 | 47.5 | 2.32 | 1.06 | 0.26 | 1.32 | 0.03 | 0.68 | 42.2 | 10.8 | 10.6 | 5.06 |
| 9 | 71.8 | 68.0 | 66.6 | 2.40 | 0.96 | 0.68 | 1.64 | 0.03 | 0.80 | 42.0 | 11.2 | 10.0 | 5.25 |
| 10 | 66.0 | 62.7 | 67.7 | 2.26 | 1.20 | 0.48 | 1.66 | 0.03 | 0.76 | 45.1 | 11.8 | 10.8 | 4.88 |
| 11 | 64.9 | 60.5 | 58.4 | 2.13 | 1.00 | 0.20 | 1.20 | 0.03 | 0.81 | 45.6 | 10.8 | 11.0 | 6.80 |
| 12 | 65.7 | 62.5 | 72.0 | 2.25 | 1.66 | 0.30 | 1.96 | 0.01 | 1.30 | 54.0 | 11.2 | 11.2 | 4.85 |
| 13 | 77.8 | 73.1 | 63.7 | 2.55 | 1.41 | 0.56 | 2.00 | 0.05 | 0.96 | 48.0 | 11.4 | 11.8 | 6.00 |
| 14 | 76.6 | 72.9 | 73.7 | 2.76 | 1.86 | 0.38 | 2.24 | 0.04 | 0.95 | 50.5 | 11.4 | 11.4 | 4.75 |
| 15 | 66.5 | 63.2 | 65.6 | 2.19 | 1.16 | 0.54 | 1.70 | 0.03 | 0.95 | 47.0 | 12.0 | 11.4 | 5.00 |
| 16 | 51.0 | 48.3 | 54.6 | 1.98 | 0.94 | 0.64 | 1.58 | 0.03 | 0.84 | 41.5 | 12.6 | 11.0 | 5.14 |
| 17 | 75.3 | 71.2 | 55.5 | 1.98 | 1.32 | 0.60 | 1.92 | 0.03 | 0.80 | 43.0 | 11.8 | 11.8 | 5.43 |
| 18 | 52.2 | 48.5 | 55.7 | 2.03 | 1.40 | 0.60 | 2.00 | 0.05 | 0.80 | 45.0 | 12.4 | 11.8 | 7.12 |
| 19 | 63.6 | 60.4 | 56.0 | 2.05 | 1.10 | 0.38 | 1.48 | 0.01 | 0.85 | 44.0 | 12.6 | 11.2 | 5.07 |
| 20 | 59.8 | 56.8 | 55.6 | 2.06 | 1.24 | 0.48 | 1.72 | 0.04 | 0.88 | 50.2 | salt'd | 11.2 | 5.00 |
| 21 | 58.6 | 50.9 | 55.3 | 2.12 | 1.14 | 0.54 | 1.68 | 0.03 | 0.79 | 45.0 | 11.6 | 11.0 | 5.00 |
| 24 | 75.3 | 71.6 | 64.8 | 2.27 | 1.26 | 0.38 | 1.64 | 0.03 | 0.80 | 47.0 | 11.4 | 11.4 | 4.88 |
| 25 | 74.0 | 70.9 | 64.7 | 2.56 | 1.54 | 0.54 | 1.98 | 0.03 | 0.85 | 46.5 | 11.4 | 11.4 | 4.23 |
| 26 | 73.4 | 70.4 | 75.8 | 2.46 | 1.70 | 0.18 | 1.88 | 0.03 | 0.90 | 45.0 | 10.6 | 11.0 | 4.08 |
| 27 | 64.2 | 61.1 | 57.9 | 2.10 | 1.10 | 0.50 | 1.60 | 0.02 | 0.80 | 47.0 | 10.0 | 10.0 | 4.80 |
| 28 | 42.5 | 40.0 | 33.0 | 1.99 | 0.84 | 0.62 | 1.46 | 0.02 | 0.81 | 47.0 | 11.2 | 10.0 | 5.97 |
| 29 | 54.1 | 50.5 | 48.6 | 2.06 | 0.84 | 0.48 | 1.32 | 0.02 | 0.80 | 43.5 | 11.0 | 10.0 | 6.57 |
| Total | 1,678.2 | 1,589.4 | 1,556.3 | 56.77 | | | | | | | | | |
| Averages | 64.7 | 61.3 | 59.9 | 2.18 | 1.23 | 0.51 | 1.74 | 0.035 | 0.84 | 45.0 | 11.3 | 10.8 | 5.29 |

REMARKS.—Furnace down from 22d to 23d, for replacing broken grate bars.

SUMMARY.

| | |
|---|---|
| Pounds copper contained in feed | 17,950 |
| Pounds copper contained in calcine | 16,800 |
| Pounds copper volatilized and in flue dust | 1,150 |
| Percentage of coal used to wet weight of feed | 3.87 |
| Percentage of coal used to dry weight of feed | 3.56 |
| Percentage of copper lost in tailings | 15.10 |
| Percentage of silver lost in tailings | 6.60 |
| Percentage of copper recoverable | 84.90 |
| Percentage of silver recoverable | 93.40 |
| Percentage of copper volatilized and in flue dust | 6.41 |
| Percentage of copper in calcine | 93.59 |
| Cost of coal per ton of tailings (dry weight). (Diamondville coal at $1.90 per ton) | $0.175 |
| Cost of acid per ton of tailings (60° Bé. acid at $3.54 per ton = $4.48 per ton $H_2SO_4$) | $0.096 |
| Cost of salt per ton of tailings (1 per cent. salt at $7 per ton) | $0.070 |
| Value of silver recoverable per ton of tailings | $0.30 |

*Temperatures.*

| Floor No. | Degrees Centigrade. | Degrees Fahrenheit. |
|---|---|---|
| 1 | 71 | 160 |
| 2 | 282 | 540 |
| 3 | 693 | 1,280 |
| 4 | 571 | 1,060 |
| 5 | 450 | 840 |
| 6 | 326 | 620 |
| Flue | 249 | 480 |
| Calcine in hopper | 277 | 530 |

There were drawn through the furnace per 24 hr. a littl than 6,000,000 cu. ft. of air, as compared with 5,500,000 cu our regular furnaces. Much of this came through the lowe and tended to cool these and remove the chloridizing fum detracting from their efficiency and from the heat economy furnace. This will be remedied in the furnace now under co tion.

Everything entering and leaving the furnace was carefully s and the amount of recoverable copper in the unground calc determined for each shift by treating a 200-g. sample with 15 a leaching solution, containing 8 per cent. of $H_2SO_4$ and 10 p of NaCl, at a temperature of about 85° C. for 2.5 hr.

*Leaching Experiments on Calcine from MacDougall No. 6*

A small leaching plant capable of treating about 20 tons at a was erected, for the purpose of ascertaining what results co obtained on the roasted tailings on a working scale. The shown in plan and photograph in Figs. 4 and 5. The copp dissolved by percolation with solutions containing sulphuric a common salt, and precipitated by means of sulphuretted hy made by the action of dilute sulphuric acid on an iron s matte, the resulting precipitate being collected in a small filte

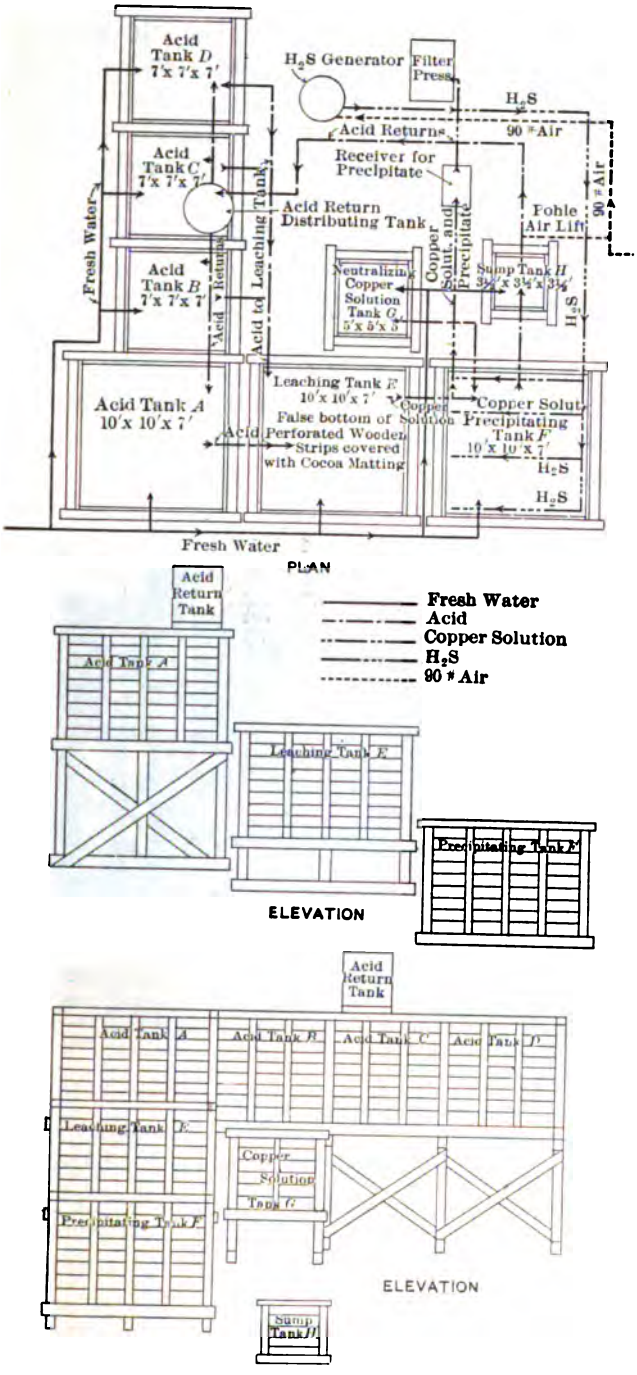
There were four solution tanks. The dimensions of the large 10 by 10 by 7 ft. and it contained the strong, or No. 2, solution carried about 6 per cent. of $H_2SO_4$ and 10 per cent. of NaCl. were three smaller tanks, of dimensions of 7 by 7 by 7 ft. these held the weak, or No. 1, solution, which carried about cent. of $H_2SO_4$ and 10 per cent. of NaCl. The other two wer for wash water, to be used for washing the tailings after leach

There was another large tank, 10 by 10 by 7 ft., which w for leaching purposes. This tank had a false bottom, whi covered with cocoa matting.

The precipitation tank was also 10 by 10 by 7 ft., was e with an agitator, and had a system of lead pipes in the bot carrying and distributing the $H_2S$ gas through the solution.

A small tank for containing excess copper solution was buil was not found necessary to use it. In case, at any time, the tation of the copper sulphide was carried too far, and the absorbed excess $H_2S$, which made it unfit for leaching purpo copper solution was to be drawn into it, to use up this excess

All circulation of solution was accomplished by means of an built of 4-in. lead pipe, drawing from a sump 4 by 4 by 4 f




G. 4.—PLAN AND ELEVATIONS OF EXPERIMENTAL LEACHING PLANT.

operated by air at 15 lb. pressure per square inch. This air lift very satisfactory and never gave any trouble. It discharged in distributing box above the solution tanks, and, by means of v manipulation, could be made to lift solution from any tank to other tank in the plant. All the pipes which carried acid or co solutions were of lead. During the time of operation, about t months, these lead pipes and fittings stood up very well and sho very little corrosion, if any, while any copper or brass fittings used a very short life. Large pinch cocks on short pieces of rubber were used as valves. They were quite satisfactory.

The $H_2S$ generator was an iron drum built to stand 90 lb. pres per square inch, lead lined, and of an inside diameter of 2 ft., an inside height of 4 ft. The iron sulphide was introduced throu hand hole in the top. There were both steam and air connect on the drum.

FIG. 5.—VIEW OF EXPERIMENTAL LEACHING PLANT.

The sulphuric acid was introduced in a concentrated condit through an iron pipe, the container being an iron barrel set abou ft. above the generator. The gas was carried through a lead pip the pipes mentioned above, which lay in the bottom of the pre tating tank. These pipes had small holes in them, spaced at reg intervals, for the escape of the gas.

In the bottom of the generator was a hole closed by a bronze p When the action ceased, this plug was removed, and the 90 lb pressure applied. This blew out all of the water and ferrous phate, and the generator was ready for charging again.

filtration of the copper sulphide was done in a small wooden ›ress. The inside dimensions of the frames were 12 by 12 by Ordinary filter paper was used as filtering medium with canvas for a backing. The copper sulphide was allowed to ›ver night and the clear solution then drawn off, so that the al for the filter press was quite dense. This thickened mate- ıs drawn into a lead-lined drum of the same dimensions as the tor, and forced through a 0.5 in. lead pipe to the filter press, 0 lb. air pressure.

scheme of operation is very simple, depending on the two cal formulæ:

$$H_2SO_4 + CuO = CuSO_4 + H_2O.$$
$$CuSO_4 + H_2S = CuS + H_2SO_4.$$

weak solution, carrying about 3.5 per cent. of $H_2SO_4$ and 10 per f NaCl, was run on the tailings first, to get out the bulk of the . This solution was the only one precipitated, and about lb. of it were used to about 24 tons of calcined tailings. The mount of strong solution, carrying 6 per cent. of $H_2SO_4$ and cent. of NaCl, was then used and returned, to be used as the or weak, solution in the next leach.

No. 1 solution remained in contact for 14 hr.; the No. 2 solu- r 72 hr. The values were washed out with a weak solution en with fresh water.

final tailings were sluiced out of the tank with a 2-in. hose, and l to waste by means of a launder carrying a 3-in. stream of

solution to be precipitated was heated to 45° C. and the $H_2S$ rned in. It is very necessary that there be a good agitation precipitation. After the amount of copper in solution was d to about 0.06 per cent., the precipitate was allowed to settle ight and the clear solution decanted and lifted to the strong n tank, to be used for the next leach. The solution was ıt up to strength in both acid and salt during precipitation.

as found that no salt was required to be added to the solutions, y picked up salt from the tailings. When the clear solution ecanted as close to the precipitate as possible, the remaining as ready for filtering. It was heated and mixed, and drawn lead-lined drum, and forced through the filter press under 90 pressure, giving the final product—a black sulphide of copper, ning some impurities.

little plant proved that fully as good results could be ob- by leaching on a moderately large scale as by the method n the daily samples in the laboratory, previously described.

summarized results of two months' operations are as follows:

TABLE IV.—*Copper and Silver Recoveries.*

| | Copper. Lb. per Ton. | S Oz |
|---|---|---|
| In feed to roasting furnace, | 11.30 | ( |
| In calcine to leaching plant, | 10.40 | ( |
| In volatilized but recoverable, | 0.90 | ( |
| In tailings from leaching plant, | 1.70 | ( |
| Percentage recoverable copper, | | 8 |
| Percentage recoverable silver, | | 9 |

*Miscellaneous Data.*

Percentage of dissolved copper in No. 1 solution, . . . . . 9
(Treatment time, 14 hr. Acid consumed per ton calcine, 13.6 lb. $H_2SO_4$.)

Percentage of dissolved copper in No. 2 solution, . . . . . .
(Treatment time, 72 hr. Acid consumed per ton calcine, 19.1 lb. $H_2SO_4$.)

Pounds $Fe_2O_3 + Al_2O_3$ dissolved per ton of calcine . . . . . .

The analysis of the tailings treated was as follows:

| Cu. Per Cent. | Ag. Oz. | Au. Oz. | $SiO_2$. Per Cent. | Fe. Per Cent. | S. Per Cent. | $Al_2O_3$. Per Cent. | ( Per |
|---|---|---|---|---|---|---|---|
| 0.60 | 0.55 | 0.002 | 82.2 | 1.9 | 2.2 | 9.4 | |

Tables V. and VI. give the total sulphur in feed and calci the sulphate sulphur in the calcine from No. 64 MacDougall Test Periods 2 and 3, respectively.

TABLE V.—*Total Sulphur in Feed and Calcine, No. 64 Mac and Sulphate Sulphur in Calcine.*

| Date. July. | Feed. Total Sulphur. Per Cent. | Calcine. Total Sulphur. Per Cent. | Calcine. Sulphate Sulph Per Cent. |
|---|---|---|---|
| 8 | 2.30 | 0.52 | 0.26 |
| 9 | 2.21 | 0.33 | 0.20 |
| 10 | 2.32 | 0.42 | 0.22 |
| 11 | 2.67 | 0.48 | 0.25 |
| 12 | 2.36 | 0.42 | 0.22 |
| 13 | 2.42 | 0.49 | 0.40 |
| 14 | 2.50 | 0.55 | 0.30 |
| 15 | 2.60 | 0.59 | 0.21 |
| 16 | 2.50 | 0.55 | 0.33 |
| 17 | 2.50 | 1.21 | 0.18 |
| 18 | 2.70 | 0.59 | 0.30 |
| 19 | 2.85 | 0.74 | 0.38 |
| 20 | 2.50 | 0.62 | 0.42 |
| 21 | 2.52 | 0.59 | 0.22 |
| 22 | 2.65 | 0.59 | 0.30 |
| 23 | 2.60 | 0.56 | 0.32 |
| 24 | 2.50 | 0.56 | 0.36 |
| 25 | 2.52 | 0.53 | 0.27 |
| 26 | 2.42 | 0.45 | 0.23 |
| 27 | 2.35 | 0.30 | 0.28 |
| 28 | 2.20 | 0.32 | 0.26 |
| 29 | 2.30 | 0.45 | 0.25 |
| 30 | 2.28 | 0.59 | 0.26 |
| 31 | 2.50 | 0.51 | 0.29 |
| Average, | 2.43 | 0.54 | 0.28 |

I.—*Total Sulphur in Feed and Calcine, No. 64 MacDougall, and Sulphate Sulphur in Calcine.*

| Date. August. | Feed. Total Sulphur. Per Cent. | Calcine. Total Sulphur. Per Cent. | Calcine. Sulphate Sulphur. Per Cent. |
|---|---|---|---|
| 2 | 2.47 | 0.50 | 0.34 |
| 3 | 2.30 | 0.48 | 0.37 |
| 4 | 2.20 | 0.47 | 0.34 |
| 5 | 2.37 | 0.46 | 0.34 |
| 6 | 2.42 | 0.50 | 0.40 |
| 7 | 2.47 | 0.51 | 0.40 |
| 8 | 2.47 | 0.53 | 0.45 |
| 9 | 2.54 | 0.51 | 0.37 |
| 10 | 2.40 | 0.48 | 0.40 |
| 11 | 2.54 | 0.50 | 0.40 |
| 12 | 2.61 | 0.59 | 0.32 |
| 13 | 2.70 | 0.51 | 0.34 |
| 14 | 2.70 | 0.52 | 0.40 |
| 15 | 2.60 | 0.52 | 0.38 |
| 16 | 2.50 | 0.51 | 0.41 |
| 17 | 2.60 | 0.52 | 0.46 |
| 18 | 2.50 | 0.55 | 0.49 |
| 19 | 2.55 | 0.52 | 0.44 |
| 20 | 2.70 | 0.55 | 0.49 |
| 21 | 2.60 | 0.56 | 0.48 |
| 22 | 2.40 | 0.56 | 0.46 |
| 23 | 2.60 | 0.56 | 0.34 |
| 24 | 2.60 | 0.55 | 0.44 |
| 25 | 2.50 | 0.51 | 0.40 |
| 26 | 2.50 | 0.53 | 0.41 |
| 27 | 2.40 | 0.53 | 0.48 |
| 28 | 2.10 | 0.48 | 0.42 |
| 29 | 2.20 | 0.44 | 0.37 |
| Average, | 2.48 | 0.52 | 0.44 |

matte for the generation of $H_2S$ was made by fusing with e and oxidized iron ore a gold-bearing pyrite ore of the fol- nalysis:

| Cu. Per Cent. | $SiO_2$. Per Cent. | Fe. Per Cent. | CaO. Per Cent. | S. Per Cent. |
|---|---|---|---|---|
| 1.2 | 23.0 | 31.6 | 1.4 | 35.9 |

esulting matte analyzed as follows:

| Cu. Per Cent. | $SiO_2$. Per Cent. | Fe. Per Cent. | S. Per Cent. |
|---|---|---|---|
| 2.23 | 0.8 | 61.3 | 31.1 |

omposed readily with dilute acid and yielded practically the al quantity of $H_2S$. About 20 per cent. excess of $H_2S$ was in precipitating (under conditions where all of the gas enter- solution was absorbed) in order to make up losses due to ts in the copper solution. The acid for generating $H_2S$ was,

of course, regenerated in the leaching solution. The prec copper sulphide analyzed as follows:

| Cu. Per Cent. | Ag. Oz. | Au. Oz. | S. Per Cent. |
|---|---|---|---|
| 58 | 69.5 | 0.04 | 29.0 |

The press cakes were firm and easily handled, but containe theless, 48 per cent. of moisture.

*Conclusion.*

There is a good deal to be said in favor of a leaching pro finishing process following water concentration, since this n unquestionably capable of giving a greater recovery of cop any other system in general use at the present time.

The treatment of high-grade (say 3 per cent.) easily con material directly in a leaching plant is, in my opinion, not a owing to greater consumption of acid and chemicals, smaller capacity, and greater tailings loss. In order to prepare ma treatment by roasting and leaching, it is necessary to crus least 15 mesh. Whatever copper is set free down to this poi be taken out as high-grade concentrates and reduced by sme is when we come to the fine grinding and concentrating pa mill that the trouble commences. Here we are between and the deep sea. We must grind fine in order to free the but in doing so we cannot avoid the production of a large p of slimes on which the concentrating machines now avai make but a very unsatisfactory recovery.

The manufacture of sulphuric acid at a smelting plant can b carried on and offers no special difficulties. From the "sn sance" standpoint, it is an advantage to the smelter to be in this industry. Roasting and leaching, under condition exist at Anaconda, can be readily done on any scale. In s entire matter resolves itself into a question of costs.

We have reason to believe that tailings material at Anac be roasted and leached for not more than 70 cents per to means that copper can be made from tailings yielding only ton for 7 cents per pound. The recovered silver will pay for marketing, and refining plus about 0.5 cent, making the net c copper about 6.5 cents per pound. These costs are made p the fact that the material in question is already mined, crus sized, and can, of course, be obtained only by making ch from roaster gases and operating on a very large scale, sa than 3,000 tons per day.

ANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.
[SUBJECT TO REVISION].

ION OF THIS PAPER IS INVITED. It should preferably be presented in person at the g, Aug. 18 to 21, 1913, when an abstract of the paper will be read. If this is impossible, ion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 reet, New York, N. Y., for presentation by the Secretary or other representative of its ess special arrangement is made, the discussion of this paper will close Oct. 1, 1913, when f the *Transactions* will go to press. Any discussion offered thereafter should preferably be of a new paper for publication in Vol. XLVII. (with suitable cross references in both

# n the Electrolytic Refining of Copper Precipitate Anodes.

BY W. F. BURNS, GREAT FALLS, MONT.

(Butte Meeting, August, 1913.)

IPTS were made in 1908, at the Great Falls Works, to pro-
ots direct from the Butte precipitate by smelting the mate-
reverberatory refining furnace. The ingots produced in this
averaged 99.4 per cent. of copper and 0.40 per cent. of ar-
l antimony.

opper content being considered too low, it was decided to
copper into anodes and to treat the anodes in the electrolytic

Furnace, in which the precipitate was treated, was a regular
refining furnace previously employed in making wire bar,
l ingot from electrolytic cathodes. The hearth of the fur-
24 by 12 ft., having a capacity of 100,000 lb. of cathodes.
ace was coal fired, and the copper was dipped by means of
dles.

verage composition of the furnace charges was as follows:

| | Per Cent. | Per Cent. Copper. |
|---|---|---|
| Precipitate, | 75.3 | 40 to 80 |
| Purification cathodes, | 1.9 | 92 |
| Refinery slag (oxide slag), | 10.1 | 50 to 60 |
| Electrolytic cathodes, | 2.1 | 99.9 |
| Precipitate anode scrap, | 3.1 | 99.0 |
| Scrap copper, | 7.5 | 99.2 |
| | 100.0 | |

ag from this furnace amounted to 46 per cent. of the total
charged and was of the following average composition:

| | Per Cent. |
|---|---|
| Cu | 16.86 |
| $SiO_2$ | 34.7 |
| FeO | 18.8 |
| $Al_2O_3$ | 13.0 |
| CaO | 9.4 |
| MgO | 2.7 |
| S | 0.47 |
| As | 0.11 |
| Sb | 0.006 |
| Ag, oz per ton | 0.18 |

Table I. shows the weights and assays of the anodes prod

TABLE I.—*Anodes Produced at Furnace No. 1 from Copper P*

| Lot No. | Anodes. | Weight. | Cu. | As. | Sb. |
|---|---|---|---|---|---|
| | | Pounds. | Per Cent. | Per Cent. | Per Ce |
| 1 | 35 | 19,402 | 99.53 | 0.251 | 0.01 |
| 2 | 66 | 37,656 | 99.37 | 0.338 | 0.01 |
| 3 | 71 | 44,102 | 99.26 | 0.479 | 0.01 |
| 4 | 76 | 47,692 | 99.20 | 0.749 | 0.01 |
| 5 | 81 | 52,544 | 98.47 | 0.586 | 0.02 |
| 6 | 102 | 59,290 | 98.85 | 0.600 | 0.05 |
| 7 | 115 | 70,814 | 98.62 | 0.589 | 0.01 |
| 8 | 177 | 105,091 | 99.60 | 0.325 | 0.01 |
| 9 | 142 | 84,790 | 98.92 | 0.368 | 0.02 |
| 1 | 177 | 95,290 | 98.98 | 0.544 | 0.02 |
| 2 | 154 | 88,751 | 99.13 | 0.284 | 0.04 |
| 3 | 317 | 171,469 | 99.38 | 0.240 | 0.02 |
| 4 | 353 | 206,863 | 98.29 | 0.794 | 0.02 |
| 5 | 246 | 150,299 | 98.66 | 0.720 | 0.02 |
| 6 | 165 | 94,390 | 99.56 | 0.253 | 0.01 |
| 7 | 667 | 366,710 | 99.45 | 0.449 | 0.02 |
| 8 | 428 | 231,896 | 99.14 | 0.510 | 0.02 |
| 9 | 450 | 240,103 | 99.12 | 0.580 | 0.03 |
| 10 | 215 | 113,342 | 99.33 | 0.500 | 0.01 |
| 11 | 467 | 237,848 | 99.04 | 0.670 | 0.04 |
| 12 | 46 | 23,228 | 99.20 | 0.860 | 0.04 |
| 13 | 64 | 36,699 | 99.25 | 0.540 | 0.03 |
| 14 | 75 | 44,949 | 99.24 | 0.350 | 0.02 |
| 15 | 70 | 42,252 | 99.34 | 0.430 | 0.02 |
| 16 | 89 | 53,892 | 99.48 | 0.310 | 0.02 |
| Totals, 25 | 4,848 | 2,719,362 | . . . . . | . . . . . | . . . |
| Arithmetical averages, . . . | | . . . . . | 99.14 | 0.493 | 0.02 |

These anodes were refined in 32 of the regular electrolyti tanks. In order that a comparison between cathodes produ these anodes at high-current density and at low-current densi be obtained, 4 of the 32 tanks were operated at one-half the density of the remaining 28. No purification of the electro attempted until the arsenic in solution had reached 55 g. p figure 44 per cent. higher than the copper in solution. done so that the action of the arsenic in solution on the c posited might be observed. Table II. shows the records tanks for the six months that they were treating these an will be noted that the arsenic and antimony content of the produced in the low-density tanks averaged but 0.0031 p notwithstanding the exceedingly high arsenic content of anodes and the electrolyte.

The cathodes produced in the high-density tanks, while h arsenic and antimony than the low-density product, were of grade than was expected.

Average current density, 17.2 amperes per square foot. Average pounds copper per kilowatt-hour=7.24.

| Date Drawn. | Average Amperes for Age of Cu. | Gross Weight of Cathodes Drawn. | Weight of Starting Sheets. | Net Weight Deposited. | Age Cathodes. | Tanks Drawn. | Average Ampere Efficiency. | Cathodes. As and Sb. | Cathodes. Ag. | Sp. Gr. | Solution. Grams per Liter. Acid. | Cu. | As. | Sb. | Fe. | Cl. |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Pounds. | Pounds. | Pounds. | Days. | No. | Per Cent. | Per Cent. | Ounces | | | | | | | |
| January . . . | 3,277 | 21,485 | 1,195 | 20.290 | 9 | 12 | 90.1 | 0.0024 | 0.01 | 1.231 | 146 | 41.8 | 20.73 | 0.41 | 5.85 | 0.0470 |
| February . . . | 4,379 | 33,954 | 1,736 | 32,218 | 8 | 16 | 87.5 | 0.0028 | 0.21 | 1.245 | 148 | 44.4 | 29.63 | 0.49 | 5.40 | 0.0419 |
| March . . . . | 4,642 | 33,884 | 1,882 | 32,002 | 8 | 16 | 86.7 | 0.0036 | 0.14 | 1.253 | 153 | 39.1 | 47.32 | 0.41 | 6.50 | 0.0393 |
| April . . . . . | 4,708 | 26,958 | 1,429 | 25,529 | 8 | 12 | 91.0 | 0.0033 | 0.22 | 1.260 | 151 | 38.3 | 55.09 | 0.35 | 6.65 | 0.0515 |
| May . . . . . | 4,700 | 35,514 | 1,985 | 33,529 | 8 | 16 | 89.8 | 0.0034 | 0.26 | 1.240 | 153 | 41.4 | 43.69 | 0.45 | 5.72 | 0.0490 |
| Averages . . | 4,341 | . . . | . . . . | . . . . | . . . | . . | 89.04 | 0.0031 | a0.17 | 1.246 | 151 | 41.0 | 39.29 | 0.42 | 6.02 | 0.0437 |
| Totals . . . | . . . | 151,795 | 8,227 | 143,568 | 8 & 9 | 72 | . . . | . . . . | . . . | . . . | . . . | . . . | . . . | . . . | . . . | . . . |

a Antimony constant at about 0.0015 per cent.

At High Current Density.

Average current density, 33.8 amperes per square foot. Average pounds copper per kilowatt-hour = 3.64.

| | | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| December . . . | 8,210 | 231,899 | 12,416 | 219,483 | 4 & 5 | 108 | 84.1 | 0.0036 | 0.30 | 1.219 | 145 | 44.0 | 8.74 | 0.32 | 5.75 | 0.0338 |
| January . . . | 6,882 | 333,327 | 17,532 | 315,795 | 4 to 10 | 164 | 86.0 | 0.0043 | 0 12 | 1.230 | 146 | 41.8 | 20.73 | 0.41 | 5.85 | 0.0470 |
| February . . . | 8,049 | 380,754 | 19,972 | 360,782 | 4 to 6 | 184 | 86.2 | 0.0045 | 0.10 | 1.245 | 148 | 44.4 | 29.63 | 0.49 | 5.40 | 0.0419 |
| March . . . . | 9,261 | 446,615 | 25,159 | 421,456 | 4 | 216 | 84.8 | 0.0074 | 0.10 | 1.253 | 153 | 39.1 | 47.32 | 0.41 | 6.50 | 0.0393 |
| April . . . . . | 9,420 | 438,800 | 25,992 | 412,808 | 4 | 212 | 83.0 | 0.0091 | 0.23 | 1.260 | 151 | 38.3 | 55.09 | 0.35 | 6.65 | 0.0515 |
| May . . . . . | 9,350 | 460,024 | 25,353 | 434,671 | 4 | 212 | 86.3 | 0.0070 | 0.24 | 1.240 | 153 | 41.1 | 43.69 | 0.45 | 5.72 | 0.0490 |
| Averages . . | 8,533 | . . . . | . . . . | . . . . | . . . | . . . | 85.07 | 0.0060 | b0.18 | 1.241 | 149 | 41.4 | 34.20 | 0.39 | 5.98 | 0.0438 |
| Totals . . . | . . . | 2,291,419 | 126,424 | 2,164,995 | 4 to 10 | 1,096 | . . . | . . . . | . . . | . . . | . . . | . . . | . . . | . . . | . . . | . . |

b Antimony constant at about 0.0015 per cent.

The anode scrap amounted to 6.3 per cent. of the anode
This was returned to the precipitate furnace.

Table III. shows the conductivity and the arsenic and
content of the furnace products obtained from the treatment
one-half of the cathodes; the remainder of the cathodes we
with other products for furnace treatment.

TABLE III.—*High-Density Cathodes from Copper Precipitate*
*Furnace No. 2.*

| Cathode Dock Lot No. | Weight of Cathodes. | As and Sb. | | Conductivity. | | Di |
|---|---|---|---|---|---|---|
| | | Cathode Sample. | Furnace Sample. | Hard Drawn. | Annealed. | |
| | Pounds. | Per Cent. | Per Cent. | Per Cent. | Per Cent. | |
| 1 | 89,354 | 0.0031 | 0.0041 | 96.90 | 99.50 | Plates a |
| 2 | 88,686 | 0.0037 | 0.0043 | 97.70 | 100.20 | Ingot b |
| 3 | 89,021 | 0.0043 | 0.0036 | 97.50 | 100.20 | Ingot b |
| 4 | 88,144 | 0.0031 | 0.0033 | 97.30 | 100.00 | Ingot b |
| 5 | 92,193 | 0.0046 | 0.0044 | 97.30 | 99.90 | Cake. |
| 6 | 84,793 | 0.0053 | 0.0053 | 96.10 | 98.70 | Ingot b |
| 7 | 92,112 | 0.0050 | 0.0031 | 96.60 | 99.30 | Cake, i |
| 8 | 94,912 | 0.0041 | 0.0034 | 96.50 | 99.20 | Cake, i |
| 9 | 87,175 | 0.0046 | 0.0041 | 97.40 | 100.20 | Cake, i |
| 10 | 90,081 | 0.0043 | 0.0052 | 93.50 | 96.10 | Ingot b |
| 11 | 83,467 | 0.0043 | 0.0072 | 95.50 | 98.20 | Ingot b |
| 12 | 93,291 | 0.0053 | 0.0058 | 94.40 | 97.10 | Ingot b |
| Totals, 12 | 1,073,229 | . . . . . | . . . . . | . . . . . | . . . . . | . . . |
| Averages... . . . . | | 0.0043 | 0.0045 | 96.39 | 99.05 | . . . |

NOTE: The "cathode sample" was obtained by punching three 0.25-in. ho
tenth cathode.

The "furnace sample" is the regular granulated furnace sample and sho
reliable than the cathode sample.

The balance of the cathodes produced were mixed with other cathodes for t

The electrolytic slime obtained from the treatment of the
tate anodes was very voluminous and much remained in su
in the electrolyte. The following analysis is of a sample
slime:

| | Pe |
|---|---|
| Cu, . . . . . | |
| S, . . . . . | |
| As, . . . . . | |
| Sb, . . . . . | |
| Sn, . . . . . | |
| Pb, . . . . . | |
| Fe, . . . . . | |
| Insol., . . . . . | |
| Ag., oz. per ton, . . . . . | 100.90 |
| Au, oz. per ton, . . . . . | 0.56 |

[a] Probably from cans used in the precipitating process.

ime produced was 31,948 lb., equivalent to 23.5 lb. of slime of anodes, as compared with 18 lb. of slime per ton of regular r anodes. This slime was treated in the reverberatory matte .

*Conclusions.*

des of wire-bar grade can be economically produced from precipitate anodes when treated at a current density of 17 to res per square foot.

des, from which cake and ingots averaging 99.85 per cent. of an be made, can be economically produced from copper pre- anodes at a current density of 33 to 35 amperes per square

emoval of the arsenic from the electrolyte is the principal electrolytic refining cost that is in excess of the cost to refine converter anodes.

ld then appear that if anodes can be produced as cheaply in ing furnace as the precipitate can be treated in the smelter, little choice between the two methods, as the copper must ly pass through the electrolytic refinery.

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ON OF THIS PAPER IS INVITED. It should preferably be presented in person at the g, Aug. 18 to 21, 1913, when an abstract of the paper will be read. If this is impossible, on in writing may be sent to the Editor, American Institute of Mining Engineers, 29 eet, New York, N. Y., for presentation by the Secretary or other representative of its es special arrangement is made, the discussion of this paper will close Oct. 1, 1913, when the *Transactions* will go to press. Any discussion offered thereafter should preferably be of a new paper for publication in Vol. XLVII. (with suitable cross references in both

# d of Testing Draeger Oxygen Helmets at the Copper Queen Mine.

BY C. A. MITKE, BISBEE, ARIZ.

(Butte Meeting, August, 1913.)

## *racter of Gases which Caused Helmets to Get Out of Order.*

ɜ September, 1911, the fire area in the Lowell mine continu-
ased and gases resulting from the fire came through the up-
. These gases contained such a large percentage of sulphur
hat the men were unable to go up and down on cages, and,
comparatively short time, repairs in the shaft were found to
ary.

he fire was in sulphide ores and extended into certain tim-
tions, the gases contained varying amounts of $SO_2$, $CO_2$, and
at times, CO. The temperature in the shaft was generally
°, and the relative humidity from 97 to 99.5 per cent. The
action of the fumes became apparent when the cables had
nged more often than usual, the heads of lag screws in the
f the shaft disappeared, and the cages needed frequent
It also became necessary to repair two compartments of the
putting in 150 ft. of new guides between the 600 and 800
d lag screws in nearly all the old guides from the surface to
evel.

## *inary Methods of Testing Helmets which were Unsatisfactory.*

conditions required all work to be done by men wearing
elmets. The Draeger type was tried, and was used until all
ere completed. At first the helmets were tested for air cir-
and air tightness and the injector was tested by water gauge
g to the methods recommended by the Draeger Oxygen Ap-
Co. Six arrangements, covering these different tests, are
d in the catalogue and circulars of the company.

the above tests had been made on the oxygen apparatus, the
complained about leaky helmets. Small quantities of gas
he helmets and could not be detected by these methods. The

tests will not reveal any small leaks in the helmet or bre bag, but test only the strong parts of the apparatus which seld out of order and not the weaker parts, such as the inflation tu handle of the cleaning sponge, and the breathing bag, whi quently needed repairs.

Aside from these objections, it was not practicable to spend amount of time in making a number of tests when the oxygen

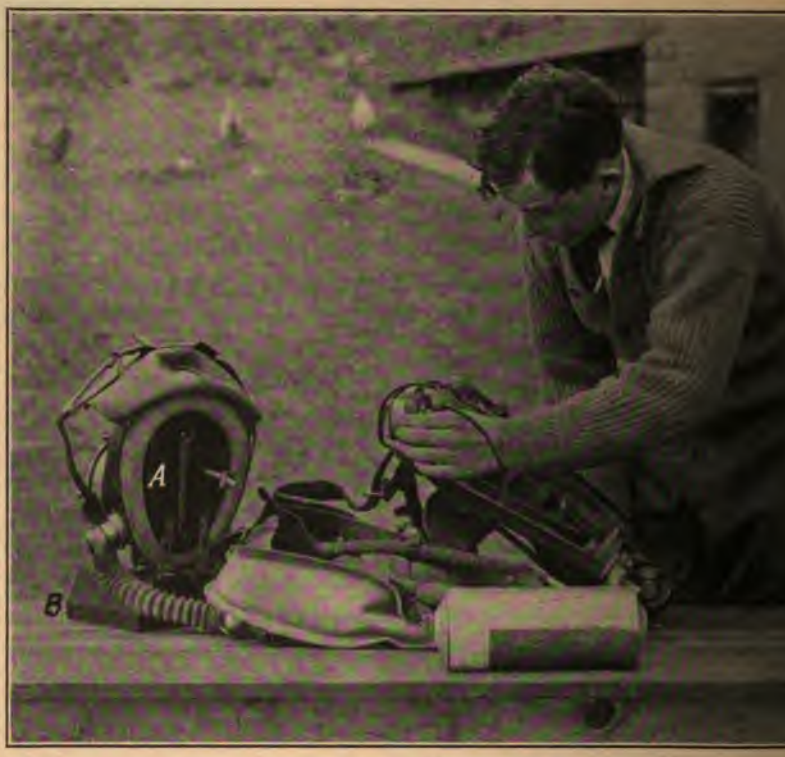


FIG. 1.—PRESSURE TEST OF DRAEGER OXYGEN HELMETS.

apparatus was used by both the day and the night shift men. therefore desirable to find some device which would test b weak and the strong parts of the apparatus within 2 min.

### 3. *The Method Used.*

The following method was used at the Copper Queen mine a found to fulfill the necessary conditions. Fastened to the lid c box containing a complete set of apparatus was a card beari

ned pressure and suction records, similar to the following
e:

| | | | |
|---|---|---|---|
| Box, | No. 1. | Check, | 10.5 cm. |
| Helmet, | No. 5,000. | Pressure tube, | 4⅛ inches water gauge. |
| Apparatus, | No. 5,500. | Suction tube, | 4 inches water gauge. |

esting device consists of a wooden block, *A*, Fig. 1, which fits helmet, taking the place of a man's face, and is provided with gauge on the side next the operator, for recording total pres-

FIG. 2.—SUCTION TEST OF DRAEGER OXYGEN HELMETS.

d suction. The device is mounted on a stand, *B*. There is ustable connection between block *A* and stand *B*, so that the can be fitted accurately to the block.

nake the pressure test, first connect the helmet to the apparatus, it it over block *A* and fit it on, as shown in Fig. 1. Inflate the nside the helmet by pressing the bulb. Remove the potash ge. Open the upper valve by pressing the thumb against eck spring and turn on the oxygen. Within 10 sec. the breath-

ing bag should be filled up and the water gauge should show
sure of 4 in. (reading for helmet in box No. 1). The quanti
determined if the breathing bag is filled in this time and
required pressure.

To make the suction test, put back the potash cartridge, re
tube from the oxygen cylinder, and place the hand firmly on the
of the breathing bag, as shown in Fig. 2. Turn on the oxy
after 10 sec. read the water gauge, which should, in this case,
4 in. If there are small leaks anywhere in the entire app
will be evident from the low pressure or suction reading
water gauge.

Generally the readings found in testing the helmets by the
described above will be much lower than those determined b
ing the water gauge directly to the oxygen tube from the
because of minor leaks or escape through the safety on the b
bag.

The test is conclusive and indicates all leaks in the weaker
the apparatus, such as the inflation tube, the handle of the
sponge, the breathing bag, and all connections. The to
required for making the complete tests is about 2 min. This
of testing the entire apparatus each day was satisfactory
respect.

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[SUBJECT TO REVISION.]

ION OF THIS PAPER IS INVITED. It should preferably be presented in person at the
g, Aug. 18 to 21, 1913, when an abstract of the paper will be read. If this is impossible,
ion in writing may be sent to the Editor, American Institute of Mining Engineers, 29
reet, New York, N. Y., for presentation by the Secretary or other representative of its
ess special arrangement is made, the discussion of this paper will close Oct. 1, 1913, when
f the *Transactions* will go to press. Any discussion offered thereafter should preferably be
of a new paper for publication in Vol. XLVII. (with suitable cross references in both

# e Evolution of the Round Table for the Treatment of Metalliferous Slimes.

BY THEODORE SIMONS, BUTTE, MONT.

(Butte Meeting, August, 1913.)

G the last half century a great amount of ingenuity and energy
devoted to the invention of appliances for the recovery of
minerals from very fine sands and slimes. The reason for
at in almost every dressing plant the greatest losses of values,
ed relatively as well as absolutely, occur in the treatment of
The natural aim of mill operators to minimize these losses
ecent years received another impetus, from the fact that the
y diminishing occurrence of high-grade ores makes a more
ecovery of values from existing resources absolutely necessary
able operation.

esults of such attempts are found in the appearance of a great
of machines and appliances which make it profitable to-day
k old dumps, containing the tailings of older processes. A
ly growing difficulty of obtaining skilled labor and the increase
s made it prohibitory to employ those types of machines
quired much attention, and in the middle of the nineteenth
the various concentrating tables of the percussion type, which
cally discharge their products, began to take the place of
mittent tables, on which, when the bed had reached a certain
s, operation had to be stopped in order to allow the skimming
of the separate layers formed on the table.

course of years, two general types of machines evolved for
ment of fine sands and slimes: The so-called concentrators
ners, on the one hand, and the buddles and round tables on
r.[1] For the profitable treatment of the very finest slimes,
bles have as a last resort proved more satisfactory and eco-
than other machines.

cent years they have again become the subject of close inves-

r type of machines, making use of centrifugal force, is at present being tried
Washoe works in Anaconda, Mont.

tigation on the part of the mill operators, with a view tow
proving the mechanical features of these machines and th
them surer of success in the competition with shaking tables
ners, which during the last decade have to a large extent usu
function of slime-treatment machines.

It is the intention of this paper to outline the evolution c
called round tables from the form in which they first appeare
dressing plants of Germany and Austria to the various forms
they are found to-day in the dressing plants the world over.

## General Principles.

The ultimate success of a machine depends in the first ins
its strict compliance with the laws and principles controlling t
to be performed. No amount of mechanical perfection ca
success when these laws and principles are violated.

To enable the reader to draw correct conclusions as to hov
tables described in this paper have complied with fundamen
ciples, and in order to point out in which direction future i
ments must be looked for, the writer deemed it helpful to
briefly the principles and fundamental laws which underlie a
tion of minerals on so-called round tables as well as c
machines. They are generally known under the name
sizing.

## Film Sizing.

If we spread out, in a very thin film, on a slightly incline
water-sized (classified) pulp containing equal-falling part
mineral, the water running down the incline will not attack
points of each particle with the same force as in a deeper str
the larger particles of lesser density will be exposed at thei
points to a greater force than the smaller, dense particles,
close to the deck of the table the water, due to adhesion, has
velocity than in the upper part of the film.

The result is, that, given a mean velocity of the wate
depends on the inclination of the table, the coarser particle
density will be carried away by its force, while the denser—
the finer—particles will resist the action of the water and rema
ing to the tables. In this manner a separation is made, in w
specific gravity (density) of the particles seems to play the c

An important requirement for a separation of the equ
particles on an inclined plane is a thin film of pulp and w
cause only in a thin film can a difference in the force of attac

ıal points of the particles of different size take place. If the ows over the incline in a deep stream all points of each parti- attacked by the water with nearly the same force, and a sorting vill take place according to the law of equal falling.

efore, to subject a classified pulp to a deep stream of water is lly the same as subjecting it to another process of classification, nay turn out to be more perfect than the preceding one; but effect a separation or concentration of the values.

rther important condition for a successful separation is the velocity of the flow of pulp and water over the table; which s likewise on the inclination of the table. Up to a certain ion the stream of water flowing over the layer of pulp will effect on the particles, because the velocity of the water is ıll to move them. If, on the other hand, the inclination is p, the velocity will be so great that the denser particles will ied away with the less dense ones. Only a mean inclination able, and therefore a mean velocity of the pulp and water, will a separation according to specific gravity.

her requirement for successful separation is the proper con- of the pulp. If too thick, that is, if the percentage of solids liluting water is too great, the water cannot act undisturbed e individual particles contained in the pulp because of inter- of the particles among themselves. If the pulp is too thin, able area is required to do the same amount of work; in other the capacity is unnecessarily diminished.

t the consistency of the pulp should be in any individual case blem that can only be satisfactorily solved by careful experi- ınd trials.

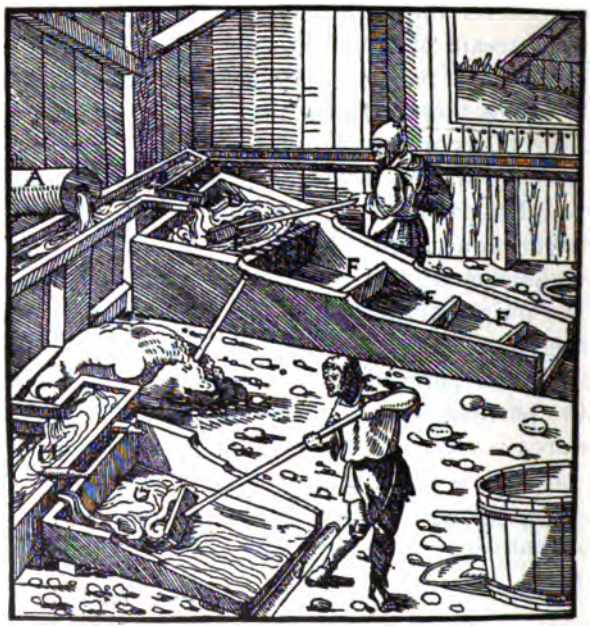
## Film Sizing Appliances in General.

iances that depend partly on separation by falling through a tal current of water and partly upon the differential rate of along the bottom of the stream have been in use from imme- periods. They constitute the most primitive type of a dressing nd illustrations of them are found in Agricola's *De Re Metal-* blished iu 1556 (see Figs. 1 and 2). They are found in almost al, but doubtless independently invented, forms among semi- d races all over the world.

## Buddles.

usually constructed, these consist of a rectangular inclosed ı trough, at the head of which the pulp to be treated is fed in,

an additional stream of water being sometimes run in near of the trough. The inclination of the trough, the qu water, and the resulting velocity of the current are so pro to the size and weight of the particles of mineral that th mineral remains at rest in the trough, while the lighter is ca In order to prevent particles of the lighter material from tangled in and held back by the heavier, it is necessary to material lying on the floor of the trough up against the which is usually done by means of a broom, rake, or hoe.




A—Pipe. B—Cross launder. C—Small troughs. D—Head of the Bu
E—Wooden scrubber. F—Dividing boards. G—Short strake.

FIG. 1.—FROM AGRICOLA'S *De Re Metallica*, PUBLISHED IN 1556.

heads thus obtained are usually shoveled or raked out fron time, while the water runs off continually. Fig. 3 shows as described above.

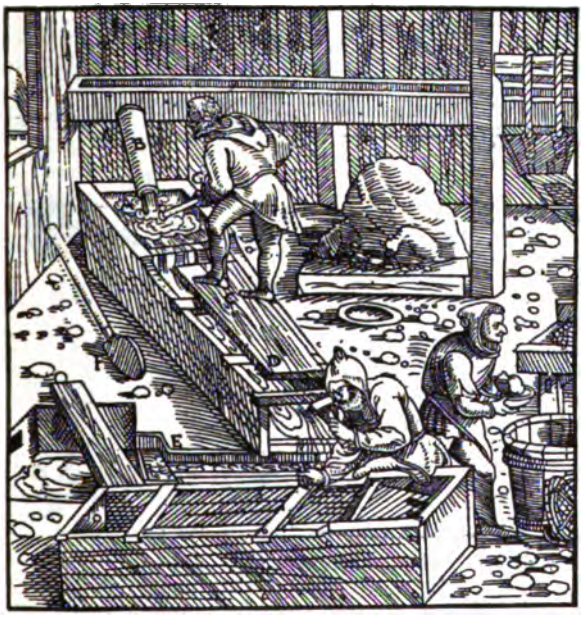
## ROUND TABLES IN GENERAL.

From the rectangular buddle described above, develope natural course of evolution the round table.

A large number of very narrow rectangular buddles around a common center, with their longitudinal dividi removed and the vacant sections between the buddles filled

called round table, either of the form of a cone (convex) or a funnel (concave), according to whether the inclination of the is away from or towards the center.

principle of action of the round table is identical with that of tangular buddle, except for the fact that the velocity of the it flows down the table is not uniform, because on a convex e pulp spreads out over a larger surface, while on a concave it is crowded into a smaller area. In the first case the thick- the film of pulp and therefore its velocity gradually decrease,



A—Head of buddle. B—Pipe. C—Buddle. D—Board.
E—Transverse buddle. F—Shovel. G—Scrubber.

FIG. 2—FROM AGRICOLA'S *De Re Metallica* (A. D. 1556).

he reverse takes place in the concave or funnel-shaped table. s dense particles rushing towards the discharge end of the ill therefore accumulate at this portion of the table in a thicker the case of a convex table and in a thinner mass in the case nnel-shaped table. Inasmuch as these particles are destined the tailings dump or to another machine, this feature does ously affect the concentration of values, which takes place in er portion of the table.

## Round Tables of the Intermittent Type.

The first round tables, like the rectangular buddles, were in tent machines; that is, after having been fed with pulp and wa a certain period, the operation had to be stopped to allow th ming of the various layers of material formed.

## Convex Round Tables—Intermittent Action.

Figs. 4 and 5 illustrate one of the old constructions of a round table. The table *A* forms a ring with an outside dian 20 ft. and an inside diameter of 6 ft., which gives a radial le the table of 7 ft. The deck consists of boards, nailed on the which are arranged star-shaped. The surface is carefully tru a plane. The lower end of the deck is surrounded by a boa

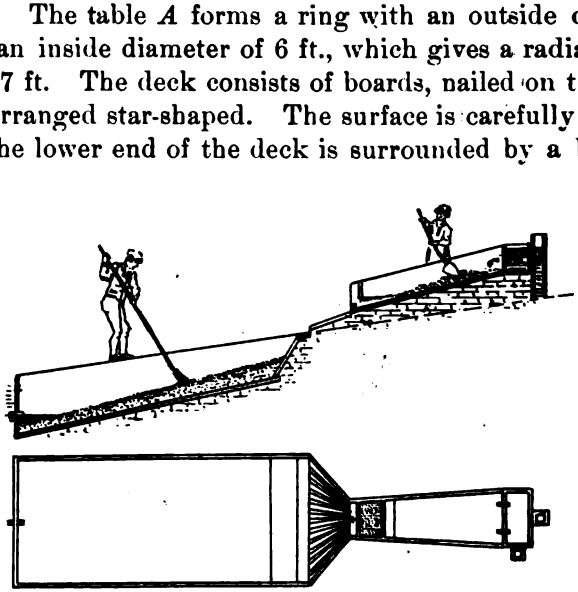


Fig. 3.—Rectangular Buddle.

$A_1$, from 9 to 12 in. high. The openings *O* in this wall per discharge of the tailings into the tailings launder *R*. The inner tric wall $A_2$, which is 12 in. high, is joined by the conical distr apron *H*, which receives the feed through the funnel *C*. The post *K* carries the step bearing for the spindle *S*, which at it end is supported by the bearing $L_1$ and is revolved by means gearing *T* and the pulleys *P*.

The pulp is brought to the table in the launder $R_1$, which de into the funnel *C*, whence it flows on the table via the apron *H* sockets $C_1$, cast integral with the funnel, carry the arms *D* o are fastened the windlasses *N* and $N_1$ by which the brushes *F* can be adjusted. These arms with their brushes make from 1 rev. per min. The object of the brushes is to keep the layer

In proportion as it increases in thickness the brushes are
y means of the windlasses.
drawback of these tables is, that, due to the variable current
ulp, the bed forming upon the deck changes its inclination
t it remains too loose, permitting the formation of furrows,

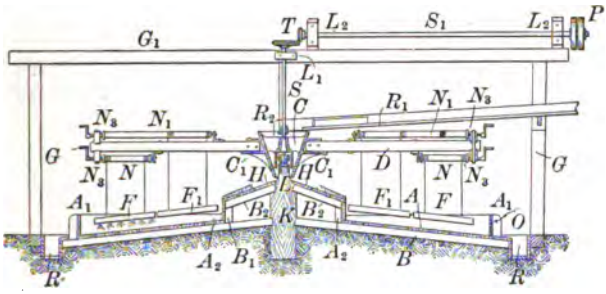



FIG. 4.

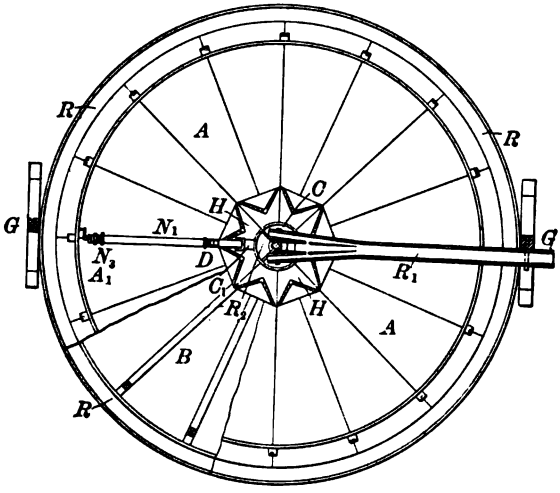



FIG. 5.

FIGS. 4 AND 5.—CONVEX ROUND TABLE.

prevent a perfect separation of the minerals. It is therefore
sible to make a clean product in one operation, but the several
ts must be re-treated on another set of tables.
work is intermittent, due to the necessary stops for skimming.
s from 2 to 3 hr. for the table to form a layer ready for skim-

Convex round tables of the intermittent type similar in de operation to the one shown in Figs. 4 and 5, but making us sonry instead of wood construction, are also built.

### Concave Round Table—Intermittent Type.

The substructure of the concave or funnel-shaped table similar to that of the convex table, as will be seen from the illus

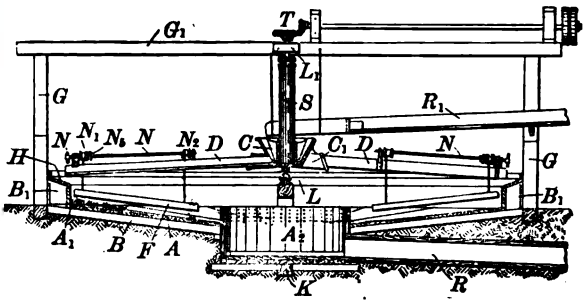



Fig. 6.

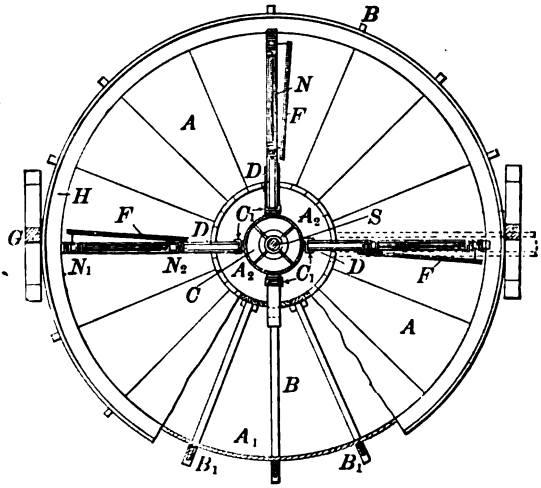



Fig. 7.

Figs. 6 and 7.—Concave Round Table.

Figs. 6 and 7. The deck $A$, which rests on the sills $B$, is sur by a wooden wall $A_1$ from 12 to 15 in. high, supporting the feed apron $H$ which receives the pulp for distribution on the

In the center the table is cut off by a wooden barrel $A_2$ diameter and extending 12 in. above and 18 in. below the d

ne upper part of the cylinder is perforated to allow the pas-
he tailings, which are carried off by the tailings launder *B*.
istribution of the pulp, which is brought to the table by the
$R_1$, is effected via the funnel *C*, from which radiate the four
*D* which spread the pulp on the feed apron *H*. The laun-
lso carry the windlasses *N* by means of which the position
ushes is regulated.
indle revolves 10 times per minute. Owing to the fact that the
lving launders distribute the pulp more uniformly over the
e surface of the bed of minerals remains smoother on a con-
n on a convex table, which results in better work.
rinciples of operation of the concave table are practically the
those of the convex tables. It is likewise intermittent.
reat drawbacks of intermittent action were soon realized, and
ere directed to the construction of tables which discharged
ous products automatically, thereby making the operation a
s one.

## Round Tables of the Continuous Type.

rinciple of continuous action was first applied to the round
making the table itself rotate, and allowing the pulp as it
own the incline to come under the influence of stationary
water, which wash the various products from those portions
ble on which they are formed, and as soon as they are formed,
able collecting launders.
manner a clean surface is ready to receive another layer of
n the table has made a complete revolution; in other words,
n is continuous.
g thus achieved continuous action, capacity was the next
importance. So, rotating tables were made of larger and
ameter. In this process of evolution two obstacles were en-
d. As the tables grew larger, they required more extensive
y foundations and substructures, occupying excessive space
ll. Then they became so heavy as to require an excessive
of power, and, being unwieldy, the motion became less
nd as a consequence the separation less perfect.
obstacles were overcome: the first, by placing several decks
me central shaft one above the other; the second, by making
stationary and allowing the fixtures, consisting of the feed
the spray pipes, and the collecting launders, to revolve.
odern round tables belong to one or the other of these types

as they evolved from the first rotary round table, installe
mills at Clausthal in the Harz mountains in 1853.

In principle, continuous-acting round tables have not u
material changes since their introduction. But numerous
cal improvements have been made, chiefly in the choice of
for the deck covering. Wood, which was used in the con
of the first tables, was subject to the undesirable property of
It was first replaced by cast iron, which for a great number
was used for deck material as well as for other parts of the m
although the turning of the castings to a true surface was exc
expensive then.

In the beginning of the twentieth century a covering o
was tried on a wooden substructure. The first experiments
material were carried on in the Harz mountains at the Hülf
mine. The technical results were altogether satisfactory,
cost of this material proved too high in the long run. Next a
cover was tried with a thickness of 3 cm. (1.25 in.). It was fo
this cement cover would crack through the cold of the win
prevent this a series of nails was driven into the deck b
cement cover was applied, these nails being set 10 cm. (4 i
and protruding slightly above the deck.

This precaution, however, did not altogether prevent c
and new experiments with covering material were made,
result that a concrete foundation with a thin cement surfacin
in the end to be the cheapest and most satisfactory deck c
A 3-cm. (1.25 in.) layer of concrete was placed on the deck,
rough boards, and finished up with cement. This also p
the surface to be made more or less rough, depending on the
ter of the slimes to be treated.

A concrete cover was first employed on the so-called Harz
round table (Fig. 8), the upper deck being funnel shaped, th
cone shaped, both decks being fastened to and revolved by t
main shaft.

This arrangement of decks had the great advantage of
possible the finishing of the product on the lower deck withou
up much mill space, although at the expense of much water.
other hand, the increased weight of the table due to the conc
cement covering made it impracticable to lubricate properly
and step bearing of the main shaft. In the end it became n
to separate the decks, and at first the upper deck was made r
while the lower one remained stationary.

This new method of covering the table with concrete and

other great advantage that it permitted giving the table *in* nclination best adapted to the nature and the consistency of es as found by trial. It also permitted a ready change of nation should a change in the nature of the feed demand it. isadvantage of these tables is, that a small unevenness of the which even with the greatest skill in turning will occur, is ause an accumulation of the heavier material in concentric rticularly when the table is overfed.

cent years iron decks have come into use again, for various First, the technical improvements in the manufacture of pliances have made the manufacture of iron decks much than in previous years. An iron deck is also much lighter

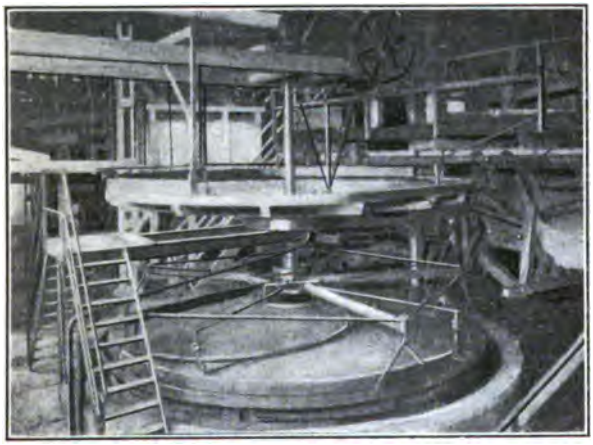


FIG. 8.—HARZ DOUBLE-DECK REVOLVING TABLE.

vooden table with a concrete cover, and hence requires less In round tables of the percussion type, like the Bartsch ooden decks would not offer the necessary resistance, so that machines iron decks are used.

XAMPLES OF ROUND TABLES OF THE CONTINUOUS TYPE.

1. *Revolving Tables with Stationary Fixtures.*

9 and 10 show a revolving round table with stationary dis- and receiving launders, and stationary wash-water pipes. ck *A* has the shape of a slightly inclined cone and, if made , it rests on a cast-iron plate which is supported in the center der *B*. The latter is keyed to the vertical spindle *D*, which ottom revolves in a step bearing *E*, being driven at the top by

a worm wheel $F$, through a worm $G$. The latter forms the horizontal shaft provided with a driving pulley $H$.

The distributing launder $J$ is a ring-shaped vessel divided compartments. The smaller one receives the pulp through

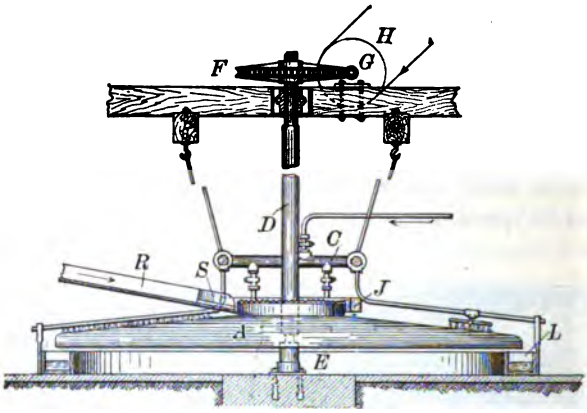



FIG. 9.

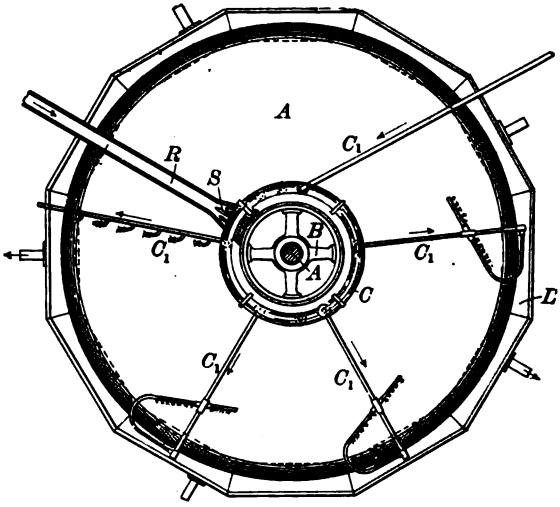



FIG. 10.

FIGS. 9 AND 10.—REVOLVING TABLE WITH STATIONARY FIXTURES.

launder $R$. Into the larger one flows the wash water from th lar tube $C$, which also provides the spray pipes $C_1$. From tributing launder $J$ pulp and water reach the tables throug short pipes in the bottom of the launder.

he table revolves, each part of it passes under the pulp com-
nt and immediately after under the wash-water compartment
distributing launder *J*, remaining under the influence of the
ntil it again passes under the pulp compartment.
operation is continuous. The coarsest, barren mineral particles
nediately washed off the table into their respective section of
eiving launder *L*. A middlings product and the concentrates
settle on the upper portion of the deck are washed off by the
vater into other sections of the receiving launder.
diameter of the table varies with the fineness of the slimes to
ted. A common dimension is from 16 to 24 ft. in diameter.
clination varies from 1 to 10, to 1 to 12. The table makes
5 to 20 rev. per hour, corresponding to one revolution in 3 to

average capacity is 10 gal. per minute of slimes containing
to 10 per cent. of solids. The quantity of wash water is 25
r minute. The power required is 0.25 h-p.
more modern construction of revolving table, the substructure
s of iron beams and corrugated iron, which is followed by a
cover, forming the deck.
*Evans Slime Table.*—In the early seventies there appeared in the
mills of the Lake Superior district revolving round tables
d by Mr. Evans, which differed from the tables theretofore in
having a stationary conical feed apron or head, which extended
y down the incline and was supposed to protect the headings
on the revolving portion of the deck until they were ready
vashed off.
11 gives the main outlines of the table. *A* is a launder to
t the slimes from the catch pit or slime box to the distributor
ch has a partition $B_1$ to separate the clear water from the
d water or slime water. The clear water is supplied by pipe
he distributor, and runs over one-half of the table, while the
vater runs over the other half, being controlled by the division
. The sand and water being on one side of distributor *B* run
n its perforated bottom, and are distributed equally over one-
the stationary head *C*, and run on the rotating table *D* into
cular launder *N*, then through the waste pipes *OO*.
headings remain on the upper part of table *D*, and after con-
ion are shielded from the action of clear water by passing
the spiral-shaped stationary head *C*, until they reappear at the
the revolution from under the widest part of the apron.
gh the action of clear water the proper grades of ore are

washed about half-way down the rotating table *D*. They th
in contact with the diagonal perforated pipe *E*, and are rew
a succession of small jets from perforations of small pipes.
passing between the jets is carried around the rotating table
it comes in contact with a jet of water from pipe *F* and co
board *G*. The jet *F* conducts the ore into hutch *H* throug

The middle or second-grade ore is washed off table *D* by
forated pipe *E*, and is deposited in hutch *J* through pipe *K*
washed.

The speed of the machine is one revolution in 80 sec. T
or incline of the table is 1.25 in. to 1 ft. The pitch of head
in. to 1 ft. The capacity of the machine is from 25 to 35
day of 24 hr.

The same type of table was afterwards used at the Washo
in Anaconda, Mont., for treating copper slimes. To save m

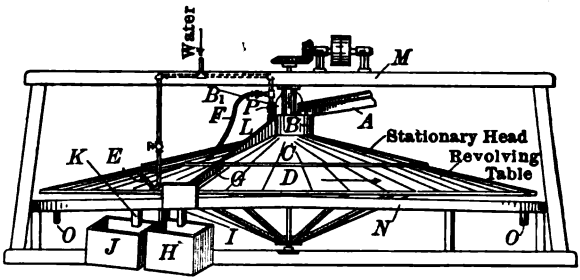



FIG. 11.—EVANS SLIME TABLE.

the tables were built with two decks on the same shaft, as s
Fig. 12.

At these works the construction and operation in the nea
of a 20-deck revolving round table is under contemplatio
subject, I am informed, is treated in a separate paper to be p
at this meeting.

### 2. *Stationary Tables with Revolving Fixtures.*

*The Linkenbach Round Table.* In 1878 Linkenbach desi
the Ems Lead and Silver Works, in Germany, a round tab
is the prototype of all tables belonging in this class.

Figs. 13 and 14 illustrate an early construction of this t
consists of a conical deck *A* with a cement surface on a ro
sonry foundation. Before hardening, the deck is turned
smoothed. Where mill space is abundant this deck can

larger diameter, this being one of the chief advantages of compared with the revolving table.

rtical spindle $A_1$ carrying the fixtures revolves at the bottom bearing $M$, while to the upper end is keyed the worm which is set in motion by the worm $E$. The latter forms f the horizontal shaft $F$ provided with a driving pulley $H$. ed is delivered through pipe $I$ into the revolving distributing $J$, which is suspended from the pipes $K$. Below it, likewise d from $K$, is the wash-water launder $L$. The water is dis- throughout the fixtures by way of the hollow spindle $A_1$.

ded from the revolving beams $B$ and revolving with them llecting launder $G$, the individual sections of which deliver ective products into the stationary sump launders $O$, $O_1$, $O_2$, $O_3$, s of pipes of different lengths.

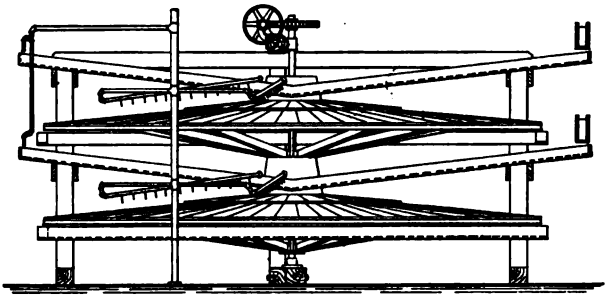


FIG. 12.—OLD STYLE ANACONDA DOUBLE-DECK EVANS TABLE.

paration and automatic discharge of the minerals take place ame manner as in the rotating tables, with this difference: change of the place of delivery where the pulp is spread on is caused by the revolution of the fixtures in the first case e revolution of the table itself in the second case.

sh-water pipes and spray pipes are so arranged that their posi- be changed within certain limits. This permits the pointing ams of water in the direction which they must assume in order e various products formed on the deck into the respective sec- he receiving launder.

e mill space, Linkenbach tables were built with three decks, e the other. Fig. 15 shows an installation of tables of the ch type built by Fried. Krupp in Germany.

re built with diameters of from 19.5 to 32.75 ft. The capac- largest size is from 2,000 to 2,400 lb. of material per hour. ber of revolutions is from 0.25 to 0.43 per min. The power

required is 0.75 h-p. The clear water consumed per minute 48 to 55 gal.

*Principle of Operation of the Linkenbach Table.*—The actio table is continuous; that is, the pulp is spread on the table a

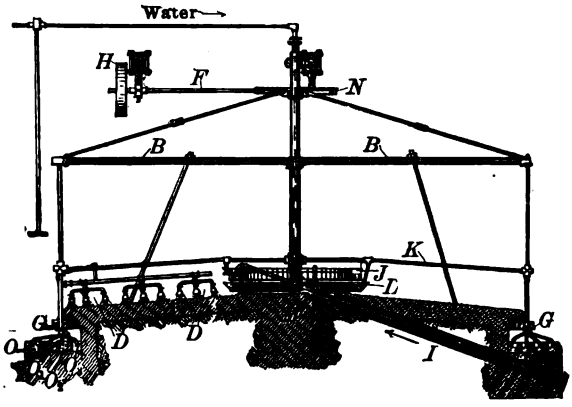



FIG. 13.

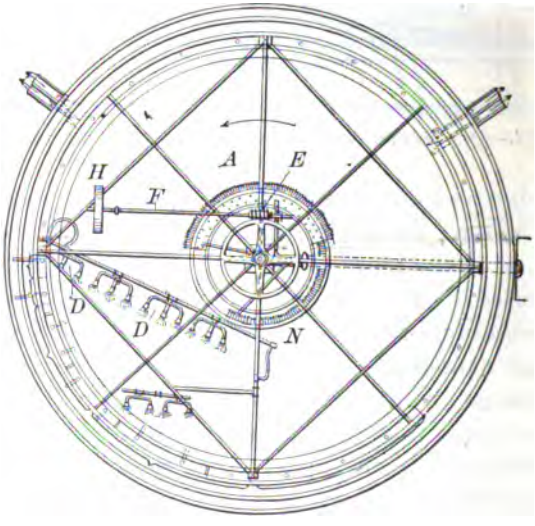
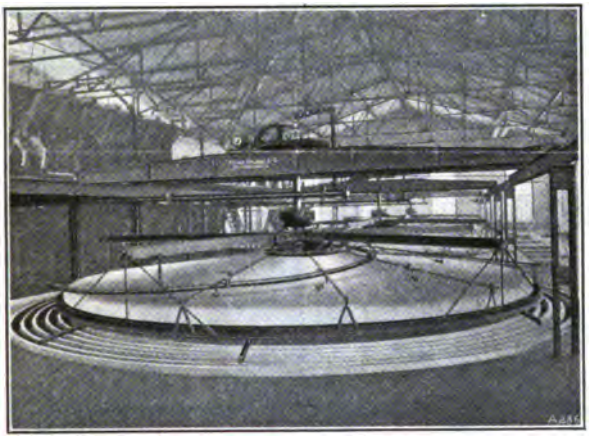



FIG. 14.

FIGS. 13 AND 14.—EARLY FORM OF LINKENBACH ROUND TABL

lowed to separate into various layers, which are washed o sively into respective sections of the receiving launder, which with the fixtures.

observed any particular section of the table during a com-
cle of operation we would see, if conditions were ideal, ap-
ately the following picture: In flowing down the table the
al particles of the pulp will settle on the deck in concentric
cording to their specific gravity, the densest nearest to the top
less dense nearest the circumference. If we have a pulp
ng, for instance, galena and blende as the valuable constitu-
h, say, quartz as a gangue, that section of the table over
the pulp distributor has just passed will immediately be ex-
the action of the wash water, which washes the gangue off
e into its respective section of the collecting launder. There
then on that section of the table near its circumference a ring



15.—TABLES OF THE LINKENBACH TYPE, BUILT BY FRIED. KRUPP, GRUSONWERK, GERMANY.

ext dense particles, consisting of middlings containing blende,
lena, and some quartz. This layer of material is now at-
by the first set of spray pipes that come along in the course
ution and is washed into its section of the receiving launder.
e layer of blende higher up is attacked by a following spray,
ayer of blende mixed with some galena still higher up, and
the pure galena particles nearest to the top of the table, all
ashed into separate compartments of the receiving launder.
he last spray has passed that section of the table, it is com-
cleared to receive a new layer of pulp from the pulp distrib-
under, which follows in the wake of the last water spray.
he proper distribution and direction of the sprays in relation
ulp distributor a more or less perfect separation can be made,
as a number of finished products, middlings, and clear tailings.

### 3. *Percussion Round Tables.*

The success of percussion tables of the rectangular type f
rating valuable minerals from their gangue, particularly wh
eral metals are to be recovered, as, for instance, from an ore co
galena and blende, has led to the application of the percussio
ciple to round tables.

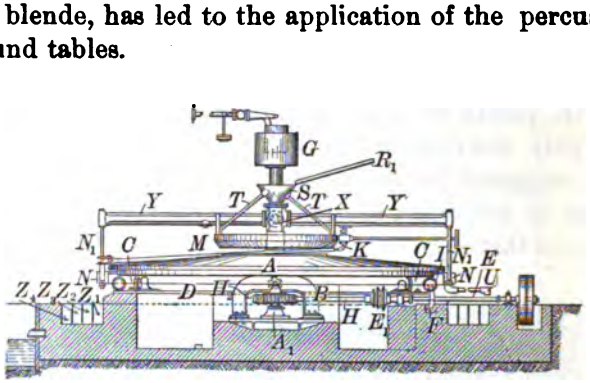



FIG. 16.

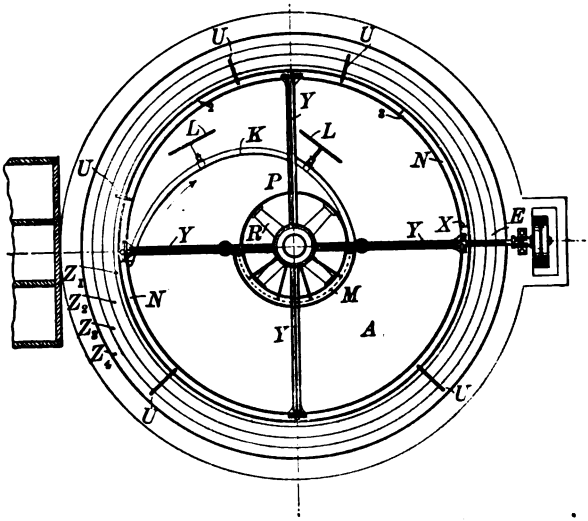



FIG. 17.

FIGS. 16 AND 17.—BARTSCH ROUND TABLE.

A well-known table belonging to this class is the Bartsc
which is operating successfully in various dressing plants i
many. It is manufactured by the Humboldt Engineering Wo
at Kalk, Germany.

*The Bartsch Round Table.*—As illustrated in Figs. 16 to 20, th
consists of a cone-shaped deck *A* made either of two cast-iron

true and covered with a coat of durable paint, or of a rubber tretched over a wooden substructure. The spindle $X$, sup- by a step bearing $A_1$, moves freely within the spider $R$, the orming the central support of the table, which on its circum- rests on the rollers $C$. To the central part of the spindle are

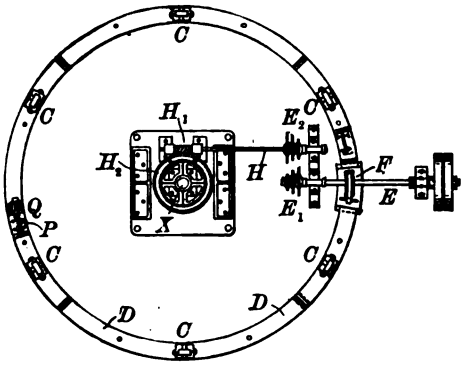



FIG. 18.

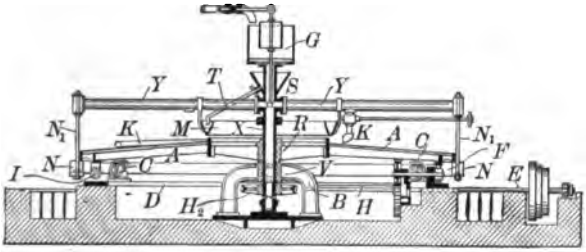



FIG. 19.

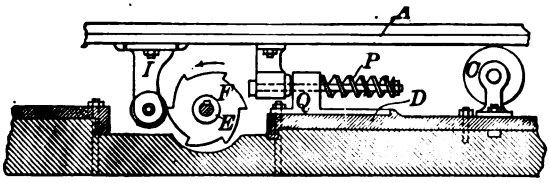



FIG. 20.

FIGS. 18 TO 20.—THE BARTSCH ROUND TABLE.

d the four hollow arms $Y$ on which are suspended the distri- launder $M$, the curved spray pipe $K$, and the rods $N_1$ which the collecting launder $N$.

will be seen, the curved spray pipe $K$ with its auxiliary sprays ts from the end of the distributing launder $M$. All sprays

receive the necessary wash water under constant pressure through the hollow arms $Y$ from the water tank $G$. The latter is provided with a float regulating the inlet valve. The spindle $X$ also carries the funnel $S$ which receives the pulp through the launder $R_1$. From the funnel it flows through the pipes $I$ into the distributing launder $M$, which feeds the table.

The mechanical operation of the table is performed by the spindle $X$, which not only rotates the fixtures, consisting of the distributing launder $M$, the spray pipes $K$ and $I$, the funnel $S$, and the collecting launder $N$, but also imparts to the table the tangential shocks in the following manner:

On the shaft $E$ is keyed a cam $F$, which strikes a roller, carried on a bearing $I$ underneath the table, at the moment when the table, due to the tension of the spring $P$, is in its artificial state of rest against the stop $Q$.

During the forward thrust the spring is further compressed, and at the end of the motion the table is jerked back against the stop $Q$, the direction of the jerk being opposite to that of the rotation of the fittings. By means of sprocket wheels, and chains $E_1$, $E_2$, the motion of shaft $E$ is transmitted to shaft $H$, at the end of which is a worm $H_1$ which rotates the worm wheel $H_2$ at the bottom of the spindle $X$. This arrangement permits of a change in the speed of revolution whenever desired.

Fig. 19 shows a somewhat different arrangement for regulating the speed of revolution by the use of step pulleys in place of sprocket wheels, and the shaft $F$ which carries the cam is set in motion by a spur wheel on shaft $E$.

The pulp is spread on the table through the perforations of the distributing launder $M$, and if the table has the proper inclination the gangue will leave the table, while the valuable (denser) minerals will cling to it until they come under the influence of the spray water.

The individual products are washed into the respective sections of the collecting launder $N$, which rotates with the other fixtures, and they are finally discharged through the pipes $U$, of different length, into the stationary circular launders $Z_1$, $Z_2$, $Z_3$, $Z_4$.

The diameter of the table is 13 ft. It is capable of treating about 0.5 ton of slime per hour. The forward thrust of the table producing the shock is from 10 to 20 mm. During one-half revolution from 100 to 120 shocks are imparted to the table. A complete revolution of the fixtures takes place in about 2 min.

From 0.25 to 0.5 h-p. is required for operation. The consumption

of spray water per hour is from 12 to 18 gal. for light material, and up to 35 gal. for heavy material.

*Principles of Separation of the Bartsch Percussion Round Table.*—While gliding down the incline, the tangential shocks imparted to the table compel the particles of mineral to follow a path which is the resultant of the straight path down the incline and the path at right angles to it, due to the shock.

The distance traveled in the direction of the incline is greater, the less dense and the larger the particles, while the distances traveled in the direction of the tangential force causing the shock are practically the same for all particles. This is due to the fact that the weight and therefore the momentum of these particles is approximately the same.

As a consequence, the resultant paths traveled by the individual particles will form different angles with that generatrix of the cone on which the said particles started on their journey down the table, the magnitude of the angle being a function of density, the largest angles corresponding to the densest, the smallest angles to the least dense particles.

This process is repeated during each time interval between two successive shocks, and each individual particle is advanced not only into another generatrix of the cone, but also into a larger parallel circle of the table; that is, towards the lower circumference of the same.

The distances between two parallel circles measured in the direction of the generatrix, due to acceleration, become greater and greater, while the angle formed between the resultant path of the particles and the generatrix in which this particle happened to be at the beginning of the last time interval, becomes smaller and smaller.

If we connect the positions occupied in succession by one and the same particle during its travel over the whole table surface, we obtain a curve, which in its horizontal projection corresponds to a spiral.

Since the individual particles of a classified feed have different densities, we shall get different curves which will be more or less steep. The least steep curve which intersects the circumference of the table at a point farthest from the generatrix on which the particle started its journey, corresponds to the densest particle, and *vice versa.*

If we treat, for instance, a pulp containing as valuable minerals galena and blende, with a gangue of, say, quartz and graywacke, the table, after being set in operation, will present at any individual moment the picture shown in Fig. 21.

The pulp is spread upon the surface 1–2–3–4, and the gangue is

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e picture shown in Fig. 21.

lp is spread upon the surface 1–2–3–4, and the gangue is

washed into the section $E_1$–$E_2$ of the collecting launder $I$ surface 2–3–5–6 the separation of a second-grade (less de product takes place, which is washed into the section $E_2$ collecting launder; on surface 5–6–7–8– is made a first-c product and washed into section $E_3$–$E_4$ of the launder second-class galena product is separated out on surface 7– first-class galena product on surface 8–9–11–10–8, and wash ively in sections $E_4$–$E_5$ and $E_5$–$E_1$ of the collecting laund

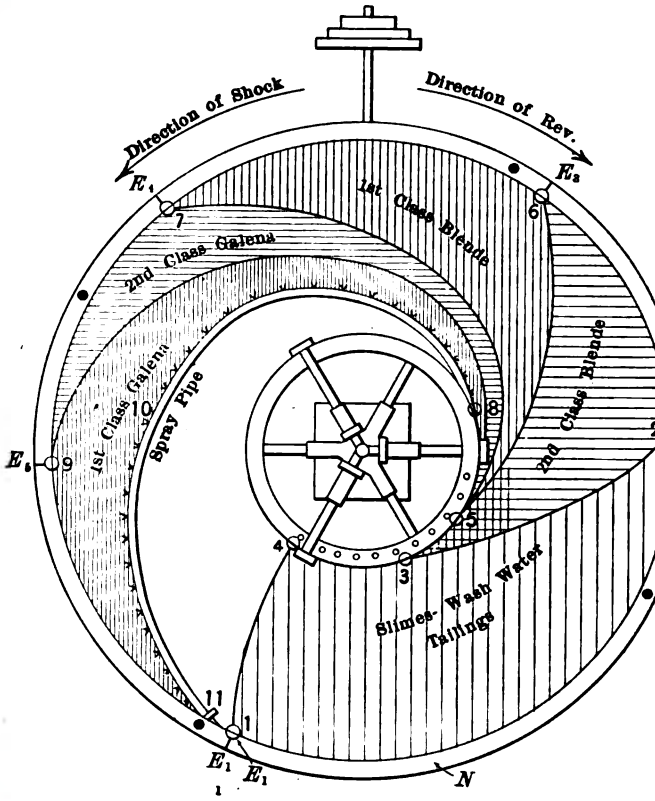



FIG. 21.—DISTRIBUTION OF PRODUCTS ON THE BARTSCH ROUND T

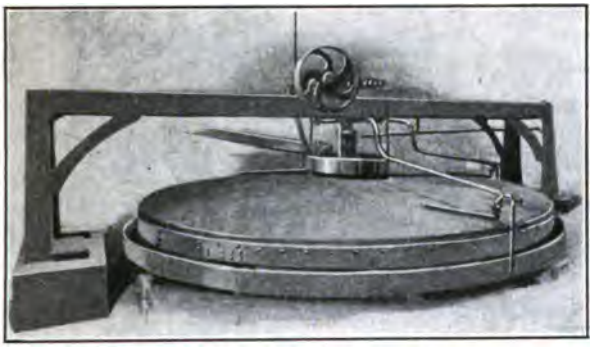
4. *Revolving Round Tables with Adjustable Inclination of* (*System Demuth*).

Various attempts to construct tables with adjustable been made, with more or less success.

Such a table is built by the Firma Goeppel in Bochum, and is illustrated in Fig 22. At this moment the writer disposal no drawing showing the details of construction of

rs, however, that the sections which compose the table over-
other slightly, and are fastened at their upper ends to a
hich can be moved up and down the main vertical shaft,
changing the slope of the deck within certain limits. The
ering, which is either rubber or linoleum, is in one piece.

*Tables with Conoidal-Shaped Decks.*—The generatrix of the
face of all tables described in the preceding pages is a straight
t the Reduction Works of the Boston & Montana Mining Co.
Falls, Mont., tests are being made at present with round



22.—Revolving Table with Adjustable Slope. System Demuth.

aving a deck the generatrix of which is the arc of a circle,
hord has an inclination from the horizon varying with the
of the pulp to be treated.

escription of these tables and the results obtained, I am told,
subject of a separate paper to be presented at this meeting.
he writer's belief that by referring to these tables he has
t the latest stage in the evolution of round tables, and there-
he end of the task which he started out to accomplish in
g this paper.

RANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.
[SUBJECT TO REVISION.]

SION OF THIS PAPER IS INVITED. It should preferably be presented in person at the ng, Aug. 18 to 21, 1913, when an abstract of the paper will be read. If this is impossible, sion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 treet, New York, N. Y., for presentation by the Secretary or other representative of its less special arrangement is made, the discussion of this paper will close Oct. 1, 1913, when of the *Transactions* will go to press. Any discussion offered thereafter should preferably be of a new paper for publication in Vol. XLVII. (with suitable cross references in both

# ıe Gold Placers of Antioquia, Republic of Colombia, South America.

BY M. H. DE HORA, NEW YORK, N. Y.

(Butte Meeting, August, 1913.)

ring my opinion of the importance of Colombia as a field for ent for those interested in mining for the precious metals, I gin by quoting Sir Clement Markham, who, in one of his tial addresses to the Royal Geographical Society of Great called the attention of its members to the fact that "al- Colombia is the least known and least developed of the South an republics, it is far and away the richest mineral country of l." This to American investors is extremely interesting, as ntry is the most accessible of all the South American republics e United States.

st all of the gold shipped by the Spaniards from Madre de ow Colon, came from Colombia, and statistics from the Mint ellin, the capital of the Department of Antioquia, go to show e total production of the precious metals of Colombia, dating e Spanish conquest in the fifteenth century to the present gregates the enormous total of $700,000,000. Of this amount, 70 per cent. represents gold obtained from the placers of the c, more than one-half of which came from the Department of ia, the balance being divided between the placers of the nd the rest of the Republic. The remaining 30 per cent. of 00,000,000 represents the silver and gold obtained from the nines, the silver greatly predominating.

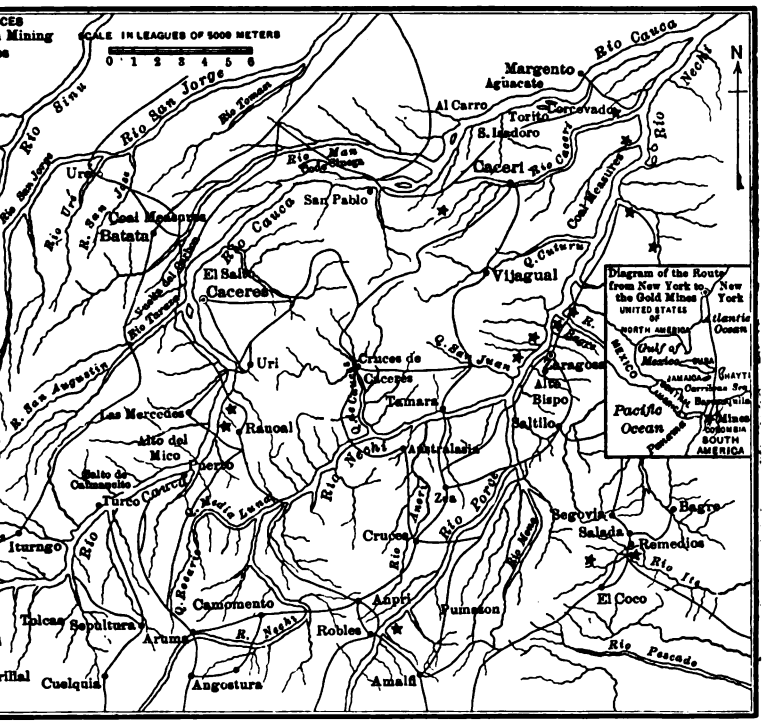
nce at the map of Colombia shows the territory somewhat in pe of an outspread human left hand extended palm downwards paper, with the wrist adjoining the Republic of Ecuador, the of the hand, fingers, and thumb representing the divisions great Andean mountain chain known as the Cordilleras of ia. The first finger represents the Eastern Cordillera, with mb as the extension to the Nevada de Santa Maria, forming

the boundary between the republics of Colombia and
The middle finger represents the Central Cordillera, betw
and the Eastern Cordillera lies the valley of the Magdal
third finger represents the Western Cordillera, between
the Central Cordillera lies the valley of the Cauca, with its
tributaries, the most important from the miner's point of
the Nechi, with its tributary, the Porce. The little fin
hand represents the range of coast hills on the Pacific slop
which and the Western Cordillera is situated the district
the Choco, containing the valleys of the San Juan and At
This district is mainly celebrated for its richness in platin
as gold; but the unhealthiness of its climate, and the d
working the deposits owing to the dry seasons and the
rains during the wet seasons, make it anything but an idea
the investment of capital, although it is very rich in precio

In my opinion, there is no part of the world so fav
investment and so attractive from the gold miner's point
the Department of Antioquia. That portion of Colom
doubtedly the most mountainous and broken portion of the
and while it is almost impossible to find a mother lode—
say, a true fissure vein—the whole country rock is mo
enriched with the precious metal; which as denudation
gradually being concentrated in the innumerable creeks
which intersect the mountain chains in all directions. A
the sketch map which accompanies this paper, Fig. 1, will
meaning plain to any one. Most of the gold recovered h
very short distances, the reason being that during the rainy
which there are two per annum, the rocks of the country,
mainly of granites and schists with occasional sandstone
thoroughly saturated on the surface with moisture; the ra
passes; the tropical sun comes out; the rocks dry up and pul
aerial denudation sets in; the light particles blow away, le
heavy particles, mainly the precious metals and gravels, to
down the hillsides into the streams during the next rainy s

Most of the river beds contain gravels of a depth of fro
ft., carrying gold values from the surface downwards. Th
generally average from 5 to 15 and even 20 cents per
until the last 2 or 3 ft. immediately above the bed rock i
when the values rise from $1 to as high as $5 and even $1
yard. Of course, what I call the pay streak varies in thic
values, but a fair average thickness is from 2 to 3 ft., and a fa
value from $1.50 to $2 per cubic yard.

ulic mining in Colombia is practically a thing of the past, uch because there are any restrictions or laws against it as old Spaniards, with their abundance of slave labor, practi- rked out all the available ground with dump for tailings. It hat there are a few Americans making good money by hy- ing by using elevators, but even ground suitable for elevators carce. The future of the enormous placer deposits of Antio- in dredging. It is a dredgeman's paradise. The gold is




AP OF THE NECHI, PORCE, AND CAUCA GOLD REGIONS, DEPARTMENT OF ANTIOQUIA, COLOMBIA.

usty nor float, but is invariably of the kind known as "shot," th an absence of large nuggets, easily recoverable, and, more t, averaging from 930 to 950 fine. The gravels are not tight; very little clay; they disintegrate easily and are characterized ence of large boulders. Most important of all, owing to the disintegrating nature of the country rock, the bed rock on e gravels rest is so soft and rotten that the lips of the buckets ke cheese; consequently there is no loss attached to the re- f the pay streak. Another point that militates in favor of

the dredgeman is that, owing to the magnificent dist
water courses, there are no sudden rises or wash-outs
rainy season; consequently the dredge is never in peri
an abundance of wood on almost every claim for minin
purposes. Labor is cheap and good when properly hand
who speak the language and know the country.

The Colombian mining law is the most liberal in the wo
simple payment of 20 years' dues in advance on each clai
title in perpetuity equal to a United States patent, but obtai
representation work or any of the irksome preliminaries
the United States Land Office; and all for a sum averagi
$500 and $600 for a claim covering in area a league o
roughly equal to 3 miles.

Lastly, as to the climate; in my opinion, there is no
North Americans or Europeans who are healthy by natu
a clean life. Yellow fever has been unheard of for years.
that there is a certain amount of malaria; for that matter
in New Jersey and of a more malignant form; but any on
precaution to sleep under a mosquito bar runs very little
tion. In my opinion, the reason why people from th
countries do not keep well in the tropics is that they a
leading too easy and too luxurious a life, trusting to
servants to do everything for them.

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# Hardinge Mills vs. Chilean Mills.

BY ROBERT FRANKE, MIAMI, ARIZ.

(Butte Meeting, August, 1913.)

v of the prominence which the conical mill has attained in rushing field within the few years since its introduction, the comparison with its more mature forerunner, the Chilean ed on extensive tests, is submitted in the interest of the rofession.

fter designing the concentrating plant of the Miami Copper 09, the Hardinge conical mill made its appearance in the orum. Its possibility as a suitable crushing device for the s well recognized, but in view of the lack of commercial ation, at the time, as to capacity, efficiency for desired pro- the still more uncertain factors of cost of maintenance and nsumption, it was deemed that the immediate adoption of ine throughout the plant would be a hazardous undertaking. reasons it was decided to equip the majority of the imme- quired units of the plant with Chilean mills, the fine-crush- ciencies of which were better known, and one section with mills, to serve as a test unit for the guidance of future in- s and replacements. Thereby, after 1.5 years' operation with s of mills, a thorough test as to metallurgical efficiency and omy has been obtained.

nical mill used in these tests is the 8-ft. Hardinge pebble ng a cylinder 22 in. in length. The cylindrical portion of s lined with cast-iron liner plates, and the conical extensions bricks bound together by cement. Each liner plate carries ing lifter, the function of which is to increase the height of e lifted material. Danish No. 5 pebbles, obtained from the osits of Denmark, are used for the grinding charge. The nill used is a fast-running, 3-roller, 6-ft. Saturn mill, with f 0.037-in. opening. The feed to these mills is the oversize screens having 0.029-in. openings, which follow rolls crush- in.

ore of this mine, a moderately hard but very fissile schist,

impregnated with finely disseminated granular chalcocite, t
mill has proved itself superior, metallurgically and econom
fine-grinding machine. This superiority it has attained b
nation of commendable characteristics, namely: Smoot
steadiness of operation, delivery of a product enabling bet
tion, more economical water consumption, a lower oper
maintenance cost, and a very low rate of depreciation.

Steadiness in operation, of paramount importance in pla
large capacities, is effected in this type of mill by its si
principle and the consequent simplicity in construction.
screens, dies and mullers are eliminated, and in their p
desirable crushing equivalents are substituted. Thus, th
the Chilean mill is replaced by a perpetual device; the dies
which have long life; and the mullers by flint pebbles
replaceable without interruption to operation.

Delays with these two types of mills in this plant have
to be as follows:

| *Chilean Mills.* | Per Cent. | *Hardinge Mills* |
|---|---|---|
| Screen delays, . . . . . | 0.57 | Relining delays, . . . . |
| Repair delays, . . . . . | 1.54 | Repair delays, . . . . . |
| Total delay, . . . . . | 2.11 | Total delay, . . . . |

From the above it is seen that the delays of the Har
approximate 60 per cent. of those of the Chilean mill, and
ing constitutes more than one-half of the total delay. Thi
due to the fact that when a mill is relined it must be idl
hours, so as to give the cement used for binding time to
delay, however, can be materially reduced by means of sh
so that a newly relined shell will always be ready to be r
an overhead crane when a worn-out shell is to be remove
ing 2 hours for this replacement, the delay from this ca
reduced to about 0.05 per cent. Thus the necessary delay
mill simmer down to those of repairs to bearings and pini
out a shell with worn lining and reinstating a renewed o
occasional replacement of feed scoops, which approxim
delay of about 0.6 per cent.

Less actual attendance is required by the Hardinge
makes it possible to reduce the operating cost in plant
duty of the attendant can be so distributed as to include
vision of other apparatus. Occasional pebble feed, lubric
look-out for obstructed discharge boxes, are the only servic
By their adoption in this plant the operating labor cost of
ing has been reduced about one-half.

he reduction practiced at this plant, the conical mill has proved be a much superior fine-grinding machine. Because of the ly granular character of the chalcocite of this ore, it is the aim uce a product of such size as will liberate a maximum mineral with as small a production of ultra fines as possible. It has und that a product which contains a maximum percentage the sizes of 60 and 200 mesh is best. Below is given a typi- en analysis of feed and product for both types of mills. From seen that the Hardinge mill yields 37 per cent. of the desired material or about 50 per cent. more than the Chilean mill, and smaller production of slime.

| h. | Chilean Mill. | | Hardinge Mill. | |
|---|---|---|---|---|
| | Feed. | Product. | Feed. | Product. |
| | 13.9 | 0 | 12.9 | 0 |
| | 47.5 | 0 | 47.3 | 0 |
| | 22.9 | 2.3 | 26.8 | 0.2 |
| | 5.2 | 11.8 | 5.0 | 3.2 |
| | 0.9 | 6.7 | 0.8 | 4.9 |
| | 1.0 | 11.4 | 0.8 | 13.8 |
| | 0.5 | 6.7 | 0.4 | 10.4 |
| 0 | 0.4 | 5.4 | 0.3 | 8.6 |
| 0 | 0.5 | 6.3 | 0.3 | 8.0 |
| 0 | 0.7 | 7.2 | 0.5 | 10.0 |
| 0 sand | 1.2 | 10.6 | 0.8 | 10.0 |
| 0 slime | 5.3 | 31.6 | 4.1 | 30.9 |

Hardinge mill also consumes less power. At this plant, a 150- luction motor operates three 8-ft. Hardinge mills or two 6-ft. mills, and the power consumption, for the above reduction, ven below. The consumption is given on the basis of both re tonnage and actual feed tonnage, the latter being approxi- at 70 per cent. of the former.

| | Crude-Ore Tonnage. | Actual Feed Tonnage. |
|---|---|---|
| | Horse-Power-Hr. Per Ton. | Horse-Power-Hr. Per Ton. |
| ill, . . . . . | 7.5 | 10.7 |
| mill, . . . . . | 6.7 | 9.6 |

riking feature brought out by the comparative operation of ills, is the difference in the duty exacted of the cone tanks. tions of the plant operated with Hardinge mills have shown age reduction of nearly 75 per cent. in the solid feed and a cent. reduction in the water feed to these tanks, as compared e Chilean mill. This is to be attributed to the combined f the smaller quantity of water fed to the grinding mill and

of the smaller production of slimes. For plants where th
tion of extreme fines is not desired, the opportunity i
offered of lessened outlay for dewatering equipment.

The maintenance costs of Chilean and Hardinge mills a
in the accompanying tables. The cost for each mill is
crude-ore tonnage so that the cost per ton of actual feed
40 per cent. greater than the cost shown, as in the reductio
of this plant about 70 per cent. of the crude-ore tonna
through the fine-grinding mills.

*Chilean Mill.*

Tons milled, 826,000.

| | | Cost Per To |
|---|---|---|
| Driving Mechanism. | | |
| Shafts, pinions and gears, | $0.00230 | |
| Spindles, muller bushings, etc., | 0.00275 | |
| Miscellaneous, | 0.00042 | $0. |
| Crushing Mechanism. | | |
| Dies (12,910 tons per die), | $0.01079 | |
| Tires (7,310 tons per tire), | 0.00987 | |
| Screens (183 tons per screen), | 0.00893 | |
| Miscellaneous, | 0.00176 | $0. |
| Total supplies, | | $0. |
| Repair labor, | | 0. |
| Shop expense, | | 0. |
| Total maintenance cost, | | $0. |

*Hardinge Mill.*

Tons milled, 450,000.

| | | Cost Per To |
|---|---|---|
| Shafts, pinions and gears, | | $0. |
| Lining. | | |
| Liner plates and lifters, | $0.00403 | |
| Silex, | 0.00300 | |
| Cement, | 0.00050 | 0. |
| Pebbles (2.51 lb. @ $0.013), | | 0. |
| Miscellaneous, | | 0. |
| Total supplies, | | $0. |
| Repair labor, | | 0. |
| Shop expense, | | 0. |
| Total maintenance cost, | | $0. |

From this it is seen that the Hardinge mill, for the practi
plant, shows a maintenance cost of about 0.5 c. per ton less
Chilean mill. It is to be noted, however, that pebble con
constitutes 70 per cent. of this cost, and the freight on pebl
prises approximately 50 per cent. of their expense in this
The item of "shafts, gears and pinions" is probably somev
in that these mills have not been operated sufficiently long

verage. Nevertheless, this part of the cost is small since ars have a long life, and, constituting but a small percentage otal, is inconsequential.

ver, the decisive factor of the lower cost of the Hardinge ts low rate of depreciation. The life of its shell, if proper aken that the lining is not allowed to wear through to it, omparatively infinite. Six Chilean mills have shown an effi- e of 825,000 tons, making the rate of depreciation, inclusive portation and installation costs, about 3 c. per ton. Allow- fe of 10 years for the Hardinge mill, its depreciation cost e less than 0.5 c. per ton.

narizing these factors, the net gain in cost by operating with dinge mill, for the practice of this plant, shows as follows:

| erating. | Cost Per Ton. Cents. |
|---|---|
| or, | 0.50 |
| ver—0.6 kw-hr., | 0.75 |
| | 1.25 |
| ntenance, | 0.50 |
| reciation, | 2.50 |
| ing, | 4.25 |

e above are to be added other advantages, the more conspicu- which are: greater capacity by reason of lower power con- n and lower delays; superior product enabling a better on to be made; smaller water consumption; and for minimum actice requires less dewatering equipment.

ermore, this mill is not yet out of the experimental stage, and e possibilities of still better performances. For instance, by ning the cylinder of a 6-ft. middlings recrushing mill from 22 ., it was found that the capacity of the mill was doubled, the onsumption lessened, and the pebble cost decreased to nearly . It would seem, however, that this idea can be carried too the more the cylinder of this mill is lengthened, the more it approach a tube mill, and so become a slimer. For regrind- dlings, however, this variation in dimension is a step in the rection, in that the liberation of occluded mineral necessi- ine product. Also large percentage variation in the sizes of ems to have a considerable influence on the consumption of Thus it was found in a test in which all the feed was sized 2.5 mm., that the pebble consumption was 1.85 lb. per ton l feed as against a consumption of 3.60 lb. with the over- d shown in the table. Experimental variations in speed, in of the feed, and in size of pebble charge, may lead to further ies.

An interesting comparison, which, while based on rath
sumptions, is so decisive in result as to be given credit, i
chanical crushing efficiency of these machines determin
method of calculation discussed by Algernon del Mar in
on Mechanical Efficiency of Crushing.[1] These calcu
shown in the accompanying tables, are based on Rittinger
"the work done in crushing is proportional to the re
diameter." This assumes that all surfaces exposed give
unit resistance to crushing, whereas it is to be inferred tha
some surfaces which, by reason of inherent fissility of the
lower unit resistance than surfaces not so favored. However
the large number of surfaces produced, it would seem rea
assume that an average unit resistance to crushing will p
not unduly long test. Furthermore, since in these calcula
machines are treated equitably with regard to the practical
which do not enter into the law, the comparative results ca
sidered fairly reliable.

Table I. shows the crushing efficiency without regard to
product made, from which it is seen that the units of redu
formed by the Hardinge mill exceed those of the Chilean m
18 to 23 per cent., depending upon the degree of accura
in the assumptions made in the calculations, and conside
cent. as a safe limit. This evaluation proves that the Har
converts more of the power consumed into reduction of
than does the Chilean mill.

Table II. shows the comparative crushing efficiency with
size of product made. From this it is seen that in amount o
formed on the various sizes of the feed, the Hardinge m
the Chilean mill in all cases, again showing that this mil
more of the power taken by it into actual work done.
more, the excess work done is mostly expended on the gra
duct desired, thereby proving that this mill more efficie
the duties assigned. It is here that the cone comes into p
geometrical device serves the function of adjusting the cr
ergy expended so as to be proportional to the force require
the particles to a given size. This is effected by two prin
are inherent with the operation of the mill. First, throug
tinual displacement of the larger particles of the charg
smaller, there takes place a segregation of the particles in
according to size—the larger assuming positions at the gr
eter and the smaller receding toward the smaller end o

[1] *Engineering and Mining Journal*, vol. xciv., No. 24, p. 1129 (Dec. 1

mill; second, through the combined action of this segregation diminishing action of centrifugal force toward the apex of ne, varying intensities of energy are imparted to the pebbles arger receiving greater inertia by reason of greater mass and lift and the smaller less and less inertia by reason of the smaller nd lower lift. Thus there exist within the mill an orderly ar-ent of zones of ore particles, each requiring a certain amount ct to be reduced to a given size, and a series of zones of so arranged as to impart impacts that tend to be proportional crushing energy required by the ore particles upon which these re exerted. For these reasons the production of slimes is zed and the accumulated forces are utilized to best advantage, s in the Chilean mill the crushing forces are uniform and dis-ageously expended upon a mixed aggregation of coarse and rticles.

Hardinge ball mill has also been tested in this plant as a sub-for rolls, for intermediate crushing on 0.5-in. material. This owever, was soon discarded, since it was found that the desired could only be obtained at too low a capacity, and the con-on of steel balls was too great to be economical.

ks are due J. Parke Channing, Consulting Engineer and Vice-nt of the Miami Copper Co., for permission to publish the lata. I am also indebted to B. Britton Gottsberger, General r, for his kindness in placing at my disposal the metallurgical these tests.

I.—*Mechanical Crushing Efficiency—Hardinge vs. Chilean Mills.*

s of law, that "the work done in crushing is proportional to the surface exposed g" and therefore "nearly proportional to the reduction in diameter" or "nearly al to the reciprocals of the diameters crushed to."

| Hardinge Mill. | | | | | Chilean Mill. | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| Reciprocal of Average Size. | Feed. Per Cent. | Relative Surface in Feed. | Product. Per Cent. | Relative Surface in Product. | Mesh. | Reciprocal of Average Size. | Feed. Per Cent. | Relative Surface in Feed. | Product. Per Cent. | Relative Surface in Product. |
| 4.1 | 12.9 | 53 | 0 | . . . . | + 4 | 4.1 | 13.9 | 57 | 0 | . . . |
| 7.2 | 47.3 | 341 | 0 | . . . . | + 10 | 7.2 | 47.5 | 342 | 0 | . . . |
| 8.3 | 26.8 | 490 | 0.2 | 4 | + 20 | 18.3 | 22.9 | 419 | 2.3 | 42 |
| 7.7 | 5.0 | 189 | 3.2 | 121 | + 30 | 37.7 | 5.2 | 196 | 11.8 | 445 |
| 8.4 | 0.8 | 47 | 4.9 | 286 | + 40 | 58.4 | 0.9 | 53 | 6.7 | 391 |
| 3.6 | 0.8 | 67 | 13.8 | 1,154 | + 60 | 83.6 | 1.0 | 84 | 11.4 | 953 |
| 8 | 0.4 | 55 | 10.4 | 1,435 | + 80 | 138 | 0.5 | 69 | 6.7 | 925 |
| 3 | 0.3 | 49 | 8.6 | 1,402 | +100 | 163 | 0.4 | 65 | 5.4 | 880 |
| 0 | 0.3 | 66 | 8.0 | 1,760 | +150 | 220 | 0.5 | 110 | 6.3 | 1,386 |
| 3 | 0.5 | 151 | 10.0 | 3,030 | +200 | 303 | 0.7 | 212 | 7.2 | 2,182 |
| 0 | 4.9 | 1,960 | 40.9 | 16,350 | —200 | 400 | 6.5 | 2,600 | 42.2 | 16,880 |
| | 100.0 | 3,468 | 100.0 | 25,552 | | | 100.0 | 4.207 | 100.0 | 24,084 |

SUMMARY.

| | Hardinge. |
|---|---|
| Units of work in product | 25,552 |
| Units of work in feed | 3,468 |
| Units of work done by mill uncorrected for capacity | 22,084 |
| Units of capacities of 2.50 tons and 2.25 tons per h-p. day, respectively | 55,210 |
| Excess units of work done by Hardinge mill | 10 |
| Excess efficiency, assuming method of calculation correct | 23 |
| Excess efficiency, assuming 5 per cent. as the limit of error | 18. |

TABLE II.—*Screen Size Crushing Efficiency.*

*Hardinge Mill.*

| Mesh. | Reciprocal of Aperture. | Feed. (Cumulative Per Cent.) | Relative Surface in Feed. | Product. (Cumulative Per Cent.) | Relative Surface in Product. |
|---|---|---|---|---|---|
| 4 | 4.9 | 87.1 | 427 | 100.0 | 490 |
| 10 | 13.3 | 39.8 | 529 | 100.0 | 1,330 |
| 20 | 29.4 | 13.0 | 382 | 99.8 | 2,934 |
| 30 | 50.5 | 8.0 | 404 | 96.6 | 4,878 |
| 40 | 66.7 | 7.2 | 480 | 91.7 | 6,116 |
| 60 | 115 | 6.4 | 736 | 76.9 | 8,844 |
| 80 | 147 | 6.0 | 882 | 66.5 | 9,776 |
| 100 | 182 | 5.7 | 1,037 | 57.9 | 10,538 |
| 150 | 272.5 | 5.4 | 1,472 | 49.9 | 13,598 |
| 200 | 333 | 4.9 | 1,632 | 40.9 | 13,620 |

*Chilean Mill.*

| | | | | | |
|---|---|---|---|---|---|
| 4 | 4.9 | 86.1 | 422 | 100.0 | 490 |
| 10 | 13.3 | 38.6 | 513 | 100.0 | 1,330 |
| 20 | 29.4 | 15.7 | 462 | 97.7 | 2,872 |
| 30 | 50.5 | 10.5 | 530 | 85.9 | 4,338 |
| 40 | 66.7 | 9.6 | 640 | 79.2 | 5,283 |
| 60 | 115 | 8.6 | 989 | 67.8 | 7,797 |
| 80 | 147 | 8.1 | 1,191 | 61.1 | 8,982 |
| 100 | 182 | 7.7 | 1,401 | 55.7 | 10,137 |
| 150 | 272.5 | 7.2 | 1,962 | 49.4 | 13,462 |
| 200 | 333 | 6.5 | 2,165 | 42.2 | 14,053 |

*Comparison of Efficiency.*

With Units Corrected for Capacity.

| Size. | Hardinge Mill. Energy Units. | Chilean Mill. Energy Units. | Difference in Favor of Hardinge. |
|---|---|---|---|
| At 4 mesh | 158 | 153 | 5 |
| At 10 mesh | 2,002 | 1,838 | 164 |
| At 20 mesh | 6,380 | 5,423 | 957 |
| At 30 mesh | 11,185 | 8,568 | 2,617 |
| At 40 mesh | 14,090 | 10,447 | 3,643 |
| At 60 mesh | 20,270 | 15,318 | 4,952 |
| At 80 mesh | 22,235 | 17,530 | 4,705 |
| At 100 mesh | 23,752 | 19,636 | 4,116 |
| At 150 mesh | 30,315 | 25,875 | 4,440 |
| At 200 mesh | 29,970 | 26,748 | 3,222 |

ANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.
[SUBJECT TO REVISION.]

ION OF THIS PAPER IS INVITED. It should preferably be presented in person at the g, Aug. 18 to 21, 1913, when an abstract of the paper will be read. If this is impossible, on in writing may be sent to the Editor, American Institute of Mining Engineers, 29 reet, New York, N. Y., for presentation by the Secretary or other representative of its ess special arrangement is made, the discussion of this paper will close Oct. 1, 1913, when the *Transactions* will go to press. Any discussion offered thereafter should preferably be of a new paper for publication in Vol. XLVII. (with suitable cross references in both

# oele Plant of the International Smelting & Refining Co.

BY H. N. THOMSON AND L. T. SICKA, TOOELE, UTAH.

(Butte Meeting, August, 1913.)

## GENERAL.

Tooele plant of the International Smelting & Refining Co. is at the mouth of Pine canyon, Tooele county, Utah. It is d with the main line of the San Pedro, Los Angeles & Salt lroad by the Tooele Valley railway, and is about 7 miles from unction. Fig. 1 is a general view of the plant.

1.—TOOELE PLANT OF THE INTERNATIONAL SMELTING & REFINING CO.

opper plant has been in operation since July, 1910, and the st furnace at the lead plant was blown in on Feb. 29, 1912. ocation of the plant was selected on sloping ground with the using gravity to the fullest extent and thus keeping labor at a m. Practically the entire plant is constructed of steel, rein- oncrete, and brick.

am ores, both lead and copper, are received over the Utah dated Aerial Tramway. The tram buckets, each containing

about 1,100 lb. of ore, are dumped into wooden terminal bins of 800 tons capacity. Ore is trammed from the terminal to receiving bins in a 50-ton electrically propelled steel bottom-dump car. All other ores, coal, and fluxes are received in railway cars, and delivered directly to the sample mill receiving bins, or to the lead plant charge bins, when material is already of suitable size for shovel sampling and direct sintering or smelting.

Coke is received in railway cars and unloaded from an overhead trestle into a storage pit arranged with concrete tunnel underneath. This tunnel contains a belt conveyor, equipped with a series of receiving hoppers and adjustable feed chutes, by means of which a uniform stream of coke is maintained on the conveyor. This conveyor discharges into a 750-ton circular bin, from which coke for blast-furnace charge is drawn direct. This is equipped with weighing hoppers similar to those under the main bin system.

### *Receiving Bins.*

The capacity of the sample mill receiving bins is about 4,000 tons of ore. These are divided into compartments for the purpose of classification. A portion of the same series of bins is devoted to coal service, and has a capacity of 2,500 tons. The ore is delivered to the sample mill by belt conveyors equipped with automatic feeders. The coal is delivered to the reverberatory furnaces and boiler plant by hopper-bottom tram cars.

### *Sample Mill and Crushing Plant.*

This is situated adjacent to the receiving bins, and consists of two complete independent sections for crushing and sampling. The method of sampling is the standard Brunton system, consisting of four Brunton oscillating time sampling machines in each section. The final sample weighs 3.2 lb. for each ton of ore sampled. The crushing sections are arranged to supply either the copper or the lead plant, and are equipped with necessary crushers, rolls, elevators, trommels, and belt conveyors, to deliver ore of suitable size as required by the different plants.

Adjacent to the mill is a complete pulping and sampling room equipped with necessary fine-grinding machinery and electrical drier. A room is provided for mine representatives from which sampling operations can be watched.

### *Resample and Converter Ore Bins.*

These consist of a series of side-discharge bins, having a capacity of 3,000 tons, divided into compartments for classification and stor-

age of small lots pending acceptance of first sampling. The contents of the bins are drawn into railway or local tram cars and delivered direct to the converter plant, or switched back to the sample mill receiving bins for resample.

*Roaster Bins.*

These consist of a series of side-discharge bins with a total capacity of 5,500 tons of crushed ore. Each side of the bins discharges on to a belt conveyor by automatic feeder, thence by successive series of belt conveyors to the roaster charge hoppers. The distribution of material to these hoppers is by mechanically propelled discharge trippers.

## Copper Plant.

*Roaster Plant.*

This consists of 32 16-ft. MacDougall type roasters, arranged in four rows. Gases from each two rows combine in one elevated brick flue with steel hopper bottom arranged with dust pipes. These flues discharge the gases directly into a dust chamber, 140 by 120 ft., and 30 ft. high above the hopper level. The dust is withdrawn from the chamber into tram cars, through three tunnels arranged with hoppers and discharge gates for this purpose.

*Reverberatory Plant.*

This plant consists of one 19 by 90 ft. and four 19 by 102 ft. Anaconda type coal-fired reverberatory furnaces. Each furnace is equipped with a 700 h-p. Stirling waste-heat boiler. One boiler is arranged with underfeed stoker and conveyor system for fuel supply, thus permitting power generation in case of furnace repairs or when any other reason necessitates shutting down the smelting furnace.

Slag is skimmed into 22-ton pots arranged with electrical tipping apparatus, Fig. 2. Ashes are discharged into 7-ton automatic side-tipping cars. Matte is conveyed by clay-lined launders, slope $\frac{7}{8}$ in. per foot, to the ladle, which is poured direct into the converters by crane.

*Converting and Casting Plant.*

This consists of five stands of horizontal cylindrical type shells, 96 by 150 in. These are electrically operated by individual motors. The converters are all lined with magnesite brick. Air pressure of from 10 to 11 lb. is found to be most satisfactory. The converters are served by a 60-ton electric traveling crane. Siliceous ore is handled in 2-ton capacity boats. Converter slag is cast in beds, loaded

into cars with crane, and shipped to the crushing plant to be s
and prepared for blast-furnace use. Copper is cast into sta
steel molds, being poured by a 30-ton crane. The converte
are delivered to the intake of a No. 20 Sirocco fan of 180,00
minimum capacity. This discharges directly into a brick flue
to the bag house chambers. In case of sudden excessive temp
the gases may be by-passed and discharged directly to the sta

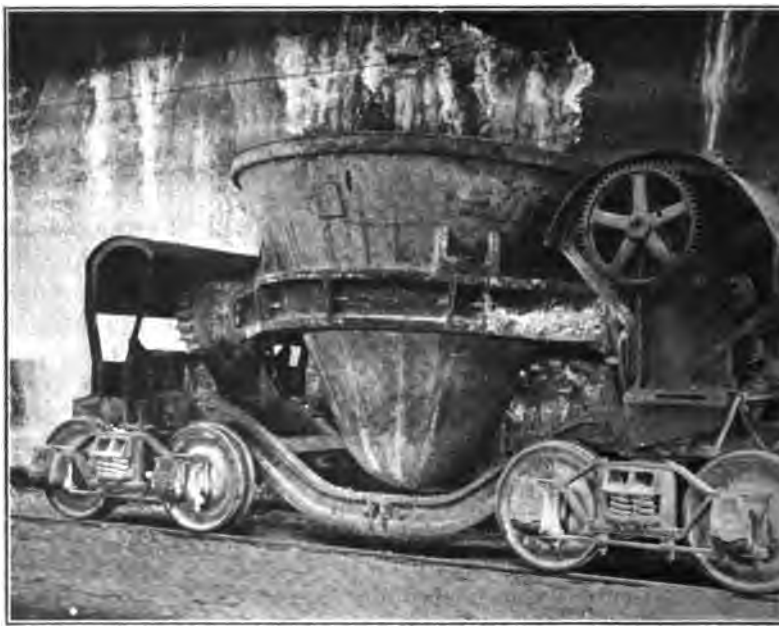

FIG. 2.—22-TON ELECTRICITY TIPPED SLAG POT.

*Converter Bag House.*

This building is divided into eight lower chambers, or cella
ered with a steel thimble floor. The building above the
floor consists of one large chamber containing 960 woolen fabr
18 in. in diameter by 30 ft. high. The flow of the gases to th
chambers is controlled by large externally operated disk valv
ated in the main flue. Shaking the bags is accomplished by
current system. This comprises a small auxiliary flue situa
rectly over the main flue, and arranged with openings leading
into the lower chambers, and controlled with disk valves. O
of this auxiliary flue is connected with the intake of a No. 4½
fan, the discharge from which passes directly into the main flu
bag-shaking system is shown in Fig. 3.

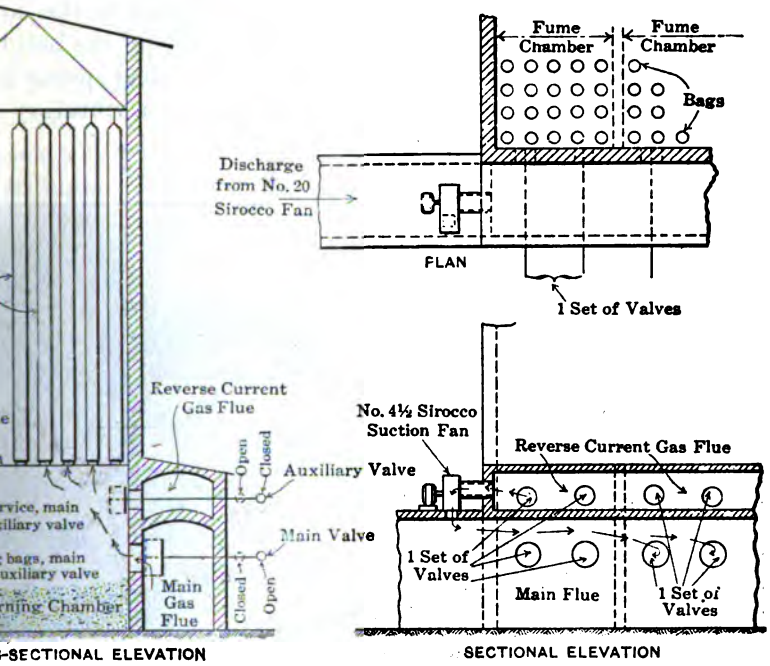


FIG. 3.—REVERSE CURRENT BAG-SHAKING SYSTEM FOR BAG HOUSES.

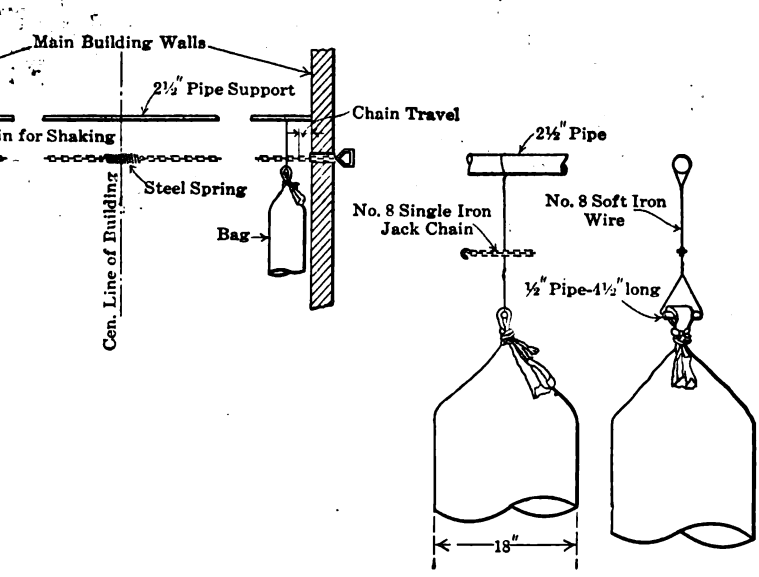


FIG. 4.—AUXILIARY BAG-SHAKING DEVICE FOR BAG HOUSES.

The bags can also be shaken by hand in case of necessity. accomplished by means of a small chain fastened to the u connection, and extending through the side walls of the buil handle is provided at each end. A long spiral steel spring centrally on each of the above chains which gives a vibratory The device is shown in Fig. 4.

FIG. 5.—VIEW SHOWING FLUE SYSTEM AT THE TOOELE PLANT.

Dust accumulation in the lower chambers is removed by a of screw and belt conveyors discharging directly into railw Gases from the bag house pass through a brick downtake into stack, 15 by 150 ft.

*Stack and Flues.*

s from the reverberatory plant pass to the main copper plant
/ a 20 by 18 ft. brick flue, 1,360 ft. long, with flat arched roof,
es from the roaster dust chamber through a flue 16 by 16 ft.
ft. long.
stack is of wire cut brick construction, 25 ft. in diameter inside,
) ft. high. The stack and the flues are shown in Fig. 5.

## Lead Plant.

*Sinter Plant and Blast-Furnace Charge Bins.*

e bins are arranged in two adjacent rows; are 260 ft. long,
ide, and divided into 52 compartments. Each compartment
long and 16 ft. wide, with a maximum depth of 24 ft. The
e hopper bottomed and arranged with independent chutes and
type hand-operated gates. Each compartment has a capacity
tons of lead ore. The adjacent compartments converge to
discharge chutes.
series of compartments are supplied directly by railway cars
head conveyors with automatic traveling trippers. Thus, ore
entrates already fine enough for treatment can be unloaded
rom railroad cars, while all other material comes direct from
shing plant by a system of 20-in. belt conveyors, 820 ft. long.
ntire structure is inclosed. Each pair of compartments is
d with a suspended weighing hopper in conjunction with
Fairbanks suspended-type scales having independent beams,
in Fig. 6. This arrangement permits material from each com-
nt to be weighed separately. All scale hoppers have circular
nd-operated gates by which the hopper contents are discharged
into sinter or blast-furnace charge cars. One end of the
tion is reserved for material for sinter-plant charge, while the
reserved for blast-furnace charge. This makes an extremely
system, as material in all 52 compartments is available on
otice. All material being available for either sinter or blast-
charge car permits of immediate charge alterations and elimi-
ossible trouble caused by irregularities in bedded ore. All
al for the lead plant can be classified in the 52 compartments,
e material is uniformly spread in the charge cars by running
slowly forward and back while ore or coke is being discharged
ne weighing hopper.
charge cars are hopper-bottom type with two drop doors. They
ivoted four-wheel trucks at each end, arranged with railway

type motor, geared direct to axle. Each car is thus equipp
two motors, which are operated through a street railway type
ler at one end of the car. The drop doors and brake mechan
be operated from either end of the car. The sinter-plant ch
body is 12 ft. long and 4 ft. wide, and holds from 3 to 5 tons,
ing on the material. The blast-furnace charge-car body is 14
and 5 ft. wide, and holds from 5 to 8 tons.

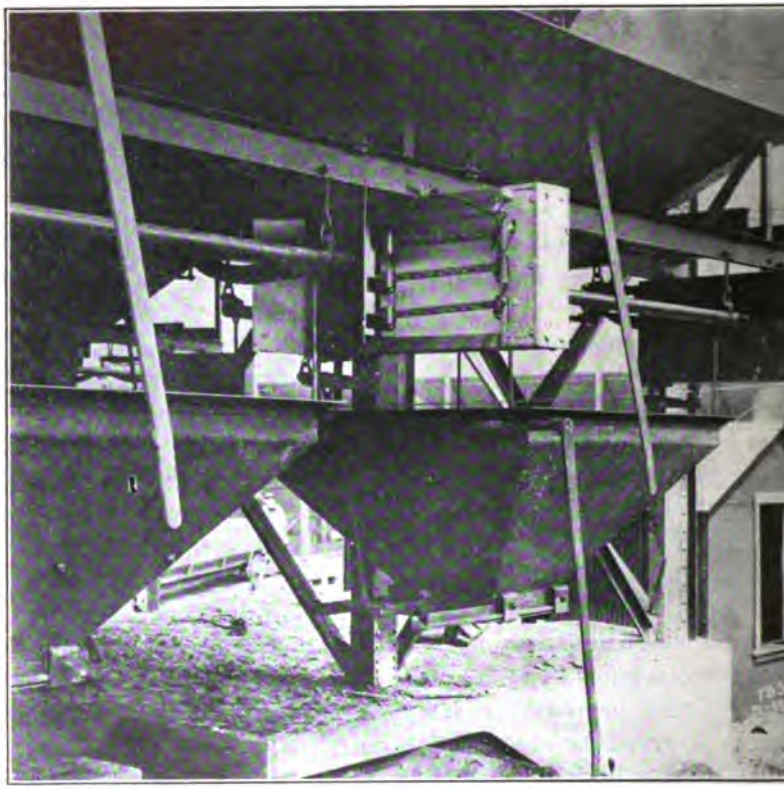

FIG. 6.—SUSPENDED WEIGHING HOPPER AND SCALES.

*Sintering Plant.*

This is housed in a building 55 ft. wide, 160 ft. long, an
high, arranged with ground floor, two main working floors, an
distributing floor. The plant consists of 10 standard 42 by
Dwight-Lloyd machines.

The weighed sinter charge is dropped from the car into a ste
per located at the extreme end of the main charge bins. This
is divided into two sections, each fitted with a small belt fee

ng to one central chute, thence to the main inclined 16-in. belt or. This delivers to a system of cross conveyors aranged with stations located directly opposite each machine feeder hopper. machine is arranged with a special adjustable feed and dis- ng apparatus. The feed hopper consists of a cylinder with bottom holding from 4 to 5 tons of charge. The discharge is fitted with an adjustable sleeve for regulating the flow of terial on to a horizontal rotating feeder table 3 ft. in diameter. aterial is discharged from this table by an adjustable plow and irectly into a segment of an adjustable inverted cone. This the feed to the apex of a raised conical formation in an inclined ble chute. The material by this device is distributed uni- across the entire width of the sintering machine, and falls into ing hopper, which is directly over the traveling grates of the ng machine, as shown in Fig. 7.

rotating feed table is operated by bevel gears and sprocket drive from the sinter-machine drive gear. The charge is ig- y oil burners inclosed in a steel housing. Oil is supplied from 00-gal. service tanks, which in turn are intermittently supplied vity from a 14,000-gal. cylindrical storage tank. This is lo- n an underground concrete building.

product of the machines discharges into inclined chutes ar- with movable gates, thence into railway cars. Each machine is by an individual 3-h-p. variable speed back-geared motor, is controlled on the main operating floor. Each machine has ft. wind boxes. The suction draft system of the plant consists special fans of modified Sirocco type, single intake, with im- 5 ft. 6 in. in diameter, mounted on a shaft and direct coupled 5-h-p. motor running at a speed of 850 rev. per min. The ca- is from 8,000 to 16,000 cu. ft. per minute. The draft varies 5 to 12 in., depending on the porosity of the material on the e grates. Each fan is connected to the wind boxes of two es by a 30-in. steel pipe. The gases are discharged from the o a modified form of balloon-type steel flue having a continu- pper bottom fitted with small dust-discharging gates. This 440 ft. long, and leads directly to the chimney.

*Blast-Furnace Plant.*

blast-furnace building is 66 ft. wide, 138 ft. long, and 44 ft. The plant consists of four furnaces, with a fifth under con- on. Two of the furnaces are 45 by 180 in. at the tuyeres, and e 60 by 180 in. All furnaces are the same height. From the

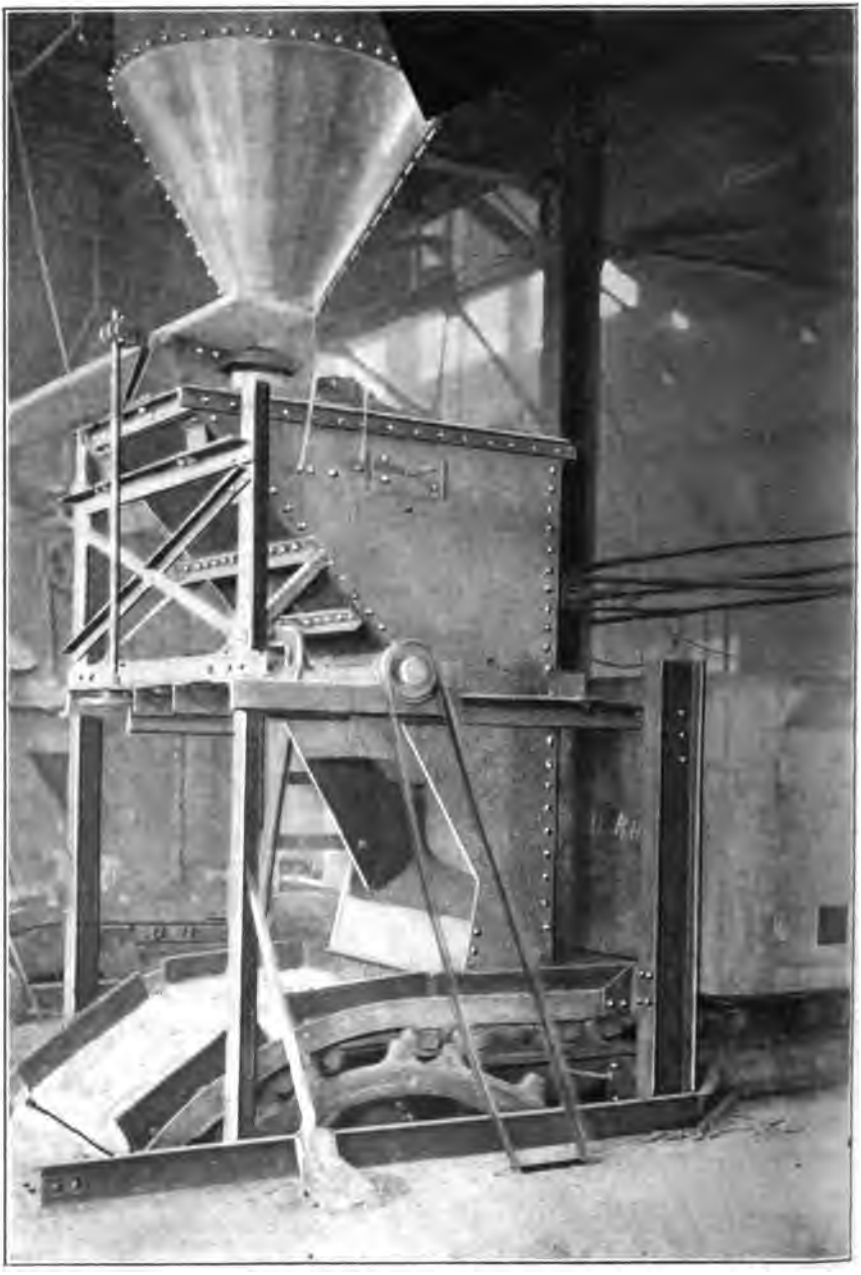

FIG. 7.—FEED DISTRIBUTOR FOR SINTERING MACHINES.

base of the crucible to the charge floor level is 30 ft., and from the center of the tuyeres to the charge floor level is 24 ft. 8 in. All the crucibles are elliptical in shape and are constructed of heavy steel plate and angle shapes, the lower portion being imbedded in 18 in. of concrete. The interior is lined with special fire brick and arranged with the crucible 32 in. deep and the full length and width of the furnace. The water jackets are of heavy steel plate, all joints welded on the outside. They are 6 ft. high, three on each side and one on each end. Each side jacket has four tuyere openings tapered from 4.5 in. outside to 4 in. inside the face of the jacket. The jackets are held in place by bolted angle connections and binder beams with adjustable tie rods. Furnaces Nos. 1 and 2 have brick shafts from the top of the jackets to the feed floor. This is supported on cast-iron columns with necessary binder rods. Furnaces Nos. 3 and 4 have a tier of upper water jackets 9 ft. in height. From the top of the upper jackets the furnace shafts of Nos. 3 and 4 are constructed of brick, supported on structural steel frame and columns. These shafts are lined with cast-iron plates. The tops of the furnaces are equipped with hinged drop doors operated by geared winches. The blast-furnace charge is made up in the car from several different weighing hoppers which contain the predetermined amount of each material. The charges can be weighed in the hoppers and thus be ready for immediate use on the arrival of the charge car. The charge car is run directly over the furnace and dumped by simple release of a latch.

The gases from each furnace pass into an elevated steel dust catcher through a downtake under the charge floor. This dust catcher is 20 ft. in diameter by 18 ft. high, with self-supporting conical roof. The gases enter at a tangent and are drawn into the main flue through an uptake 6 ft. in diameter. The main flue is similar in construction to the sinter-plant flue, and is about 500 ft. in length from the furnaces to the bag-house fan intake. The dust collected in the dust catchers is withdrawn directly on to a belt conveyor system, which delivers into railway cars.

Lead is tapped into conical pots of 3 cu. ft. capacity, and trammed direct to the drossing plant. Slag and matte are tapped either into extra large movable settlers, or into 7-ton pots to be transferred to the settling furnace. In a case of a shut down of the settling furnace the matte is tapped from the settler into large pans of about 1.5 tons capacity. On cooling, it is removed from the pan by a crane and subsequently shipped to the converter plant for treatment.

*Settling Furnace.*

This is a coal-fired reverberatory furnace 12 by 36 ft. wi area of 64 sq. ft. Its total capacity is 125 tons of slag a The entire bottom of the furnace is inclosed in a steel plat box to prevent the escape of lead.

The stack for this furnace is of self-supporting steel typ diameter by 110 ft. high, and lined for 40 ft. with fire-cl The flue and stack are so designed as to permit the installa waste-heat boiler, should this prove advisable.

Slag and matte are tapped into the furnace from pots, two in the roof being provided for this purpose so that two po handled at the same time. Slag is skimmed from the front a 22-ton pot, thence to the dump. Matte is tapped at the s furnace into 8-ton cast steel ladles mounted on transfer car cars are trammed to the converter building, where the picked up and poured direct into the converters by the crane.

*Drossing Plant.*

This is inclosed in a building 40 ft. wide, 80 ft. long, a high. The equipment consists of four coal-fired cast-iro having a content capacity of 33 tons of lead. The kettle rounded by necessary steel working floors. The lead is d means of a Howard press operated entirely by compressed dross is returned to the charge floor of the blast-furnace p mono-rail tram and elevator system. The drossing an operations are controlled by an electric pyrometer, whic regularity of operation. The bullion is siphoned into cast- vidual molds mounted on a radial supporting frame. The b approximately 100 lb. each. They are transferred to truck consisting of 30 bars. This is trammed directly over bull into railway cars for direct shipment to the International finery at East Chicago, Indiana.

*Bag House.*

This is of similar design and construction to the conv house, with the following exceptions:

There are 10 compartments with 144 bags each. The and reverse current fans are of the same capacity as those a verter bag house. Seven compartments are equipped wit cotton fabric bags and three with woolen bags. The burni have small cooling flues underneath, which are connecte

on flue through a small chimney. The gases from the bag pass through a downtake to the base of the lead-plant chimney. pass flue is provided so that the gases may by discharged direct chimney in case of too high temperature or a shut down of the ouse for repairs, etc.

*Chimney.*

is of 12 ft. inside diameter at the top, and 18.5 ft. inside diam- the base, and 200 ft. high. It is constructed of pressed brick, andles both the sinter-plant and blast-furnace gases.

POWER PLANT.

power used by both the copper and the lead plants is generated main power house,which contains the following units:

750-kv-a. electric A. C. generators, 2,200 volt, direct coupled ical triple-expansion marine type steam engines.

750-kv-a. electric A. C. generator, 2,200 volt, direct coupled to s type steam turbine.

50-kw. electric D. C. exciter generator, 110 volt, direct cou- A. C. motor.

50-kw. electric D. C. exciter generator, 110 volt, direct cou- horizontal high-speed steam engine.

250-kw. electric D. C. exciter generators, 500 volt, direct cou- horizontal tandem-compound Corliss steam engines.

blast-furnace blowers, rotary type, top charge, nominal capac- ,000 cu. ft. free air per minute, compressed to 38 oz. gauge- re. Impellers are single-end gear driven, and direct coupled izontal tandem-compound Corliss engines.

converter blowing engine, duplex air cylinders, direct driven izontal cross-compound Corliss steam engine. Capacity 9,500 free air compressed to 12 lb. gauge pressure.

converter blowing engine, duplex air cylinders, direct driven izontal cross-compound Corliss steam engine. Capacity 13,500 free air per minute compressed to 12 lb. gauge pressure.

90-lb. compressor, two-stage air cylinders, direct driven by ntal cross-compound Corliss steam engine. Capacity 4,000 cu. air per minute compressed to 90 lb. gauge pressure.

90-lb. compressor, two-stage air cylinders, belt driven by a 175- ectric motor. Capacity 2,000 cu. ft. free air per minute com- d to 90 lb. gauge pressure.

Leblanc type condenser, direct coupled to A. C. electric motor, e accommodation of A. C. generator, converter blowing and compressor engines.

One Leblanc type condenser, direct coupled to non-conden turbine, for direct accommodation of steam turbine of 750 erator set. Condenser turbine exhausts into open filter water heater.

One direct-acting jet condenser, steam driven, for the ac tion of steam engines of the D. C. generators and bl blowers.

All condenser circulating water is delivered to natural cooling towers.

Three direct-acting, duplex, tandem-compound boiler-fe outside-packed plungers. Steam cylinders exhaust into heater.

One direct-acting duplex steam fire pump.

Ashes from the boilers are handled by reverberatory f cars.

All A. C. electric power generated is distributed to t departments of both copper and lead plants at 2,200 volts. units of 75 h-p. and over use power at 2,200 volts; units and less at 440 volts; lighting power at 110 volts. In electric motor drives are used to advantage on belt conv pumps, etc.

All D. C. power generated is used for electric locomot and blast-furnace charge cars, 50-ton tram cars, converter traveling cranes in converter plant, power house and variable speed motor drives in sintering plant, conveyor f some belt conveyors.

*Steam Boiler Plant.*

Steam for the power plant and general plant service i by the following Stirling water-tube boilers:

Four 700 boiler horse-power reverberatory waste-heat b

One 700 boiler horse-power underfeed stoker fired b slack coal fuel.

Three 350 boiler horse-power hand fired boilers, using fuel.

Feed water is supplied to the boilers at a temperature mately 200° F. Gases from the boilers pass directly into reverberatory furnace flue.

MISCELLANEOUS.

*Water Supply.*

This is obtained from springs in Pine canyon and stor ervoir of about 2,500,000 gal. capacity, from which the w

nd distributed to the various departments of the copper and ants by gravity pipe lines. The water is of excellent quality culinary and boiler-feed purposes. Scale in boiler tubes is t, muddy nature and can be readily cleaned out.

cooling water for the blast-furnace jackets is supplied from an ndent natural-draft type cooling tower and is kept in circula- means of turbine pumps direct coupled to electric motors.

ity supply fire tanks are located at an elevation considerably he plants in general and connected to an independent fire line having hydrant and hose stations with hose and car equipment ted about the plants.

*Industrial System.*

entire industrial tramming systems about the plant consist of miles of standard 4 ft. 8.5 in. gauge tracks, with 60-lb. steel nd substantial fastenings. All tracks on which electric loco- s and charge cars operate are equipped with overhead trolley ystem. D. C. power is used at 500 volts. The locomotives nge from 8 to 18 tons in weight, and are equipped with ade- notor power and controllers.

*Laboratory, Shops, and Offices.*

laboratory consists of complete equipment necessary for all cal and assay work required for both plants, and all control n smelter settlements. Electricity is used exclusively for hot and water stills.

shops consist of machine, boiler, blacksmith, electric, and car- repair shops, adequately equipped with machine tools and elec- ne service to accommodate general machinery and locomotive and fabrication of improved plant equipment.

plete warehouse and office accommodations are maintained, ces accommodating superintendent and assistants, engineering ment, time keepers, and clerical force.

METALLURGY.

llurgy at the copper plant presents a few novel features. Sul- the charge to the MacDougall roasters averages only 22 per This low sulphur necessitates the use of from 1 to 2 per cent. to produce a satisfactory calcine. Several of the roasters are ed with pipe hoppers from the charge floor, which pass through e wall and discharge on to the fifth hearth of the furnace. us ore necessary to adjust the charge to give a 41 per cent.

$SiO_2$ slag at the reverberatories is fed through these pipes. becomes mixed with the hot calcine coming down throug nace, as it crosses the fifth and sixth hearths before droppin calcine hoppers.

The charge to the reverberatories is low in copper, aver than 2.5 per cent. The matte produced runs about 20 p copper and is converted direct. No particular difficulties h experienced in this operation, and two tons of ore runni cent. of $SiO_2$ are smelted in the converter for every ton produced.

At the lead plant some departures from accepted standar have been made. No ore-bedding system, either manual or cal, is in use, the material being handled as already descri roasting equipment other than Dwight-Lloyd machines is No matte crushing or roasting equipment is required.

In case of a shut down of the settling furnace the cold taken to the converter plant and melted direct in the conve about 3 per cent. of coke and 1 per cent. of coal. About 50 of the lead in the matte is recovered in the bag-house fum per cent. in the converter slag.

The sinter charge will average 16 per cent. of S, and th per cent. The blast-furnace slag will run from 32 to 36 p $SiO_2$, this being varied as the zinc percentage on the charg or low.

The lead plant is still in the throes of construction, and th metallurgy has not as yet reached a state sufficiently settled a detailed description possible at this time.

RANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.
[SUBJECT TO REVISION.]

SION OF THIS PAPER IS INVITED. It should preferably be presented in person at the ng. Aug. 18 to 21, 1913, when an abstract of the paper will be read. If this is impossible, sion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 treet, New York, N. Y., for presentation by the Secretary or other representative of its less special arrangement is made, the discussion of this paper will close Oct. 1, 1913, when of the *Transactions* will go to press. Any discussion offered thereafter should preferably be of a new paper for publication in Vol. XLVII. (with suitable cross references in both

# e New International Diamond Carat of 200 Milligrams.

BY GEORGE FREDERICK KUNZ, NEW YORK, N. Y.

(Butte Meeting, August, 1913.)

nanifold inconveniences resulting from the absence of a uni-
ndard of mass for determining the weight of precious stones
ıg been obvious. This lack has been keenly felt in commer-
nsactions, and those who have devoted time and research to
y of historic diamonds and precious stones have had frequent
to deplore the absence of such a standard in the past.

paper read in Chicago in 1893, before the International Con-
Weights and Measures, held in connection with the World's
ian Exposition, the writer suggested dividing the carat into
s, and constituting a standard international carat of 200 mg.;
5 diamond carats or 20 pearl grains to a French gram, mak-
0 carats or 20,000 pearl grains to a kilogram. He also called
ı to the fact that while this would depreciate the present dia-
arat or pearl grain only about 2.5 per cent., it would abolish
blesome discrepancies between the various carat-weights now
and could be easily explained and understood everywhere.[1]

ubject of the various diamond carats, their incongruity, and
ılting confusion as to the correct weights of historic gems
efinite records of them are searched for, has been fully treated
n an extensive study of this subject in *The Book of the Pearl.*[2]
e earnest and unremitting efforts of C. E. Guillaume, Director
Bureau Internationale des Poids et Mesures at Sèvres, is largely
eventual success of this eminently desirable reform, which
constantly urged both by articles on the subject and by ad-
delivered in Paris before the International Committee for
s and Measures.[3]

general adoption of a uniform standard for dealings in pre-

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*he Book of the Pearl*, by George Frederick Kunz and Charles H. Stevenson, p.
York, 1903).
pp. 321–329.
E. Guillaume, Les récents progrès du système métrique; reprint from vol. xv. of
*Mémoires du Bureau Internationale des Poids et Mesures*, pp. 62–66 (Paris, 1907).

cious stones, based upon a carat of 200 mg., was early rec
be a result much to be desired, and great progress has rec
made in this direction. The carat has heretofore varied
in the different countries, with more or less resulting con
inconvenience to business. The metric carat has now bee
adopted in most of the countries of Europe, and its use
pulsory. Our own country has at last taken action,
Britain will do so before long. Many of our dealers a
ers recognize the theoretical advantage and the convenie
change, but they apparently fail to take any active interest
practical reform. There can be little question, howeve
change must come ere long. A single standard for all cou
the substitution of decimal for common fractions in the
advantages so plain that they must surely soon be realized

A striking illustration of the defectiveness of the syste
in use, as compared with the proposed new standard, is
recent article by L. J. Spencer, of the British Museum
Larger Diamonds of South Africa.[4] In this article an effo
to clear up certain published errors and misstatements as t
diamonds obtained in recent years from the African mine
proved impossible to ascertain definitely the precise weigh
of these notable stones, especially those of the earlier disc
account of the uncertainty as to which carat-weight had
ployed in determining them. The metric equivalent of
"Board of Trade carat" is 205.304 mg.; while that of th
lish carat, in use prior to 1888, was 205.409 mg. Hence,
of the stones, only an approximate statement of their origi
is now possible.[5]

A first step in the direction of simplifying the carat-
taken in 1871, when the syndicate of Paris jewelers, golds
others dealing in precious stones proposed the adoption of
205 mg. (3.1636 grains) to take the place of the older F
of 205.5 mg. (3.17135 grains); this action was confirme
On Oct. 17, 1890, the Association of Diamond Cutters o
fixed the value of the carat at 1 kg. = 4,875 carats, whi
carat of 205.183 mg., or 3.16561 grains troy.

The more radical and effective change to the carat of 20
the subject of a resolution passed in January, 1906, by the
Syndicale de la Bijouterie, Joaillerie et Orfèvrerie de Pari

[4] *Mineralogical Magazine*, vol. xvi., No. 74, pp. 140 to 148 (London, Oct.,
[5] G. F. Kunz, Precious Stones during 1911; reprint from *The Mineral*
xx., pp. 624, 625.

e Syndicale des Négociants en Diamants, Perles, Pierres Pré-
et des Lapidaires de Paris, and the reform was strongly ad-
by M. Guillaume, in 1906, before the Commission des Instru-
t Travaux. Shortly before, in the early part of 1905, the
Federation of Jewelers had petitioned the Imperial govern-
legalize the carat then in common use as a standard weight,
was refused as in violation of the law prescribing the exclu-
of the metric system. In the course of the discussion aroused
refusal, M. Guillaume advocated the new international carat
mg. as a possible solution of the difficulty, and as early as
905, this proposition was taken into consideration by the In-
nal Committee. In August, 1906, on the motion of Ludwig
r, the following resolution was adopted by the German Fed-
: "Considering that it is both necessary and advantageous to
the old carat by the metric carat of 200 milligrams, the Fed-
authorizes its president to approach the Imperial government
foreign associations in order that the metric carat may be
ced as soon as possible in all countries."

wing the action of the International Committee, on Dec. 7,
e Chamber of Commerce of Antwerp promised to rescind the
on of Apr. 29, 1895, approving the adoption of a carat of 205.3
soon as the new international carat of 200 mg. should come
versal use in the markets. About the same time the Associa-
Jewelers and Goldsmiths of Prague formally authorized the
Federation to propose in its name the reform of the old carat
as possible by international agreement, and the Association
smiths of Copenhagen declared its willingness to support such
ure. The Belgian Committee of Weights and Measures, in
907, declared its willingness to petition the government to
the new carat on its adoption by the more important
es.

ngland and its colonies the proposed change was favorably
d. In September, 1907, a resolution indorsing the new carat
ssed by the Association of Manufacturing Jewelers of Mel-
Australia; on Oct. 16, 1907, the Association of Societies
Protection of Commerce in the United Kingdom passed a
on urging its adoption in all countries, and on Jan. 23,
e Birmingham Jewelers' and Silversmiths' Association gave
ion to the hope that all nations would speedily accept an
tional carat of 200 mg.

new carat has now been legalized in a number of countries;
s, laws favoring it are in course of preparation, or of adoption.

Definite information has reached us concerning the legal status of the new international carat in the following lands:

*Germany.*—In the German Empire the question of the carat has been very simply settled, without any new legislation. The fifth article of the law of May 17, 1856, abrogated the carat in common use, which was not referred to in the laws of Aug. 17, 1868, and of May 30, 1908. Now while in most countries not only is the use of non-metric unities prohibited, but also any nomenclature foreign to that of the metric system, the laws of the German Empire are silent in regard to this latter particular. Hence those interested can give the name "carat" to the unity they commonly use without coming in conflict with the laws, on the sole condition that this unity be represented by a standard figuring in the table of standards of mass subject to verification.

In conformity with the legal stipulation officially communicated by the Secretary of the Interior to the General Federation of German gem-dealers and jewelers, the Secretary has decided to adopt the new carat.

*Belgium.*—The draft of a law, signed by the King, was submitted to the Chambers at the beginning of July, 1909. This law has not yet been promulgated.

*Bulgaria.*—The new carat was included in the provisions of a law published Apr. 10, 1910.

*Denmark.*—The first article of a law voted Apr. 1, 1910, and which became operative on that day, is couched in the following terms:

"In the application of the metric system to the commerce in precious stones, pearls, etc., the metric carat, equivalent to 200 milligrams, shall henceforth be used."

*Spain.*—A Royal Order of Mar. 11, 1908, has prescribed the use of the new carat of 200 mg.

*France.*—The law of June 22, 1909, comprises a single article, as follows:

"In the transactions relating to diamonds, pearls, and precious stones, the designation 'metric carat' may, in violation of the first article of the law of July 4, 1837, be given to the double decigram.

"The use of the word 'carat' to designate any other weight is hereby prohibited."

Thus the new French law formally prohibits the use of the old carat, but admits that of the new international carat of 200 mg., considered to be an authorized violation of the fundamental law on the application of the metric system. The prohibition regarding the old carat was to become effective from Jan. 1, 1911, but owing to the

difficulty of preparing the proper weights at so short notice, the request of the jewelers that a postponement should be granted until Jan. 1, 1912, was accorded; since that time the law has been operative.

This law was elaborated in two decrees, of July 7 and of Dec. 13, 1910. The first of these decrees stipulates that "the form of the carat-weight shall be that of a quadrangular, truncated pyramid, or of a cylinder surmounted by a knob. However, the carat-weights of less than one gram are to be in the form of square-cut metal plates. The dimensions of the cylindrical weight shall differ from those established for such weights by the fifth appendix to the ordinance of June 16, 1839.

"The various weights are to be inscribed in intaglio and in legible characters, the number of grams on the lower face; that of the metric carats, followed by the abbreviation C. M., on the upper face."

The same decree enumerates the carat-weights constituting the minimum complete series with which the dealers interested must provide themselves; this series is in conformity with the metric system, between 2 mg. and 100 g.

*Holland.*—The law defining and legalizing the carat of 200 mg. was laid before the Second Chamber on June 9, 1910. In the exposition of the motives for its preparation reference is made to the desire expressed by the International Committee of Weights and Measures in its session of 1905, and to the decision of the Fourth General Conference. This law was promulgated Apr. 7, 1911.

*Italy.*—Parliament has already legislated, in principle, the new international carat (July 7, 1910); a Royal Decree will fix the date on which it shall come into use, after consultation with the National Commission of Weights and Measures.

*Japan.*—An ordinance of Nov. 11, 1909, specifies that "when the weights of precious stones are expressed in carats, the word 'carat' should designate the mass of 200 milligrams."

*Mexico.*—The government, considering that the designation "metric carat" merely constitutes an exception to the fundamental law, does not see any objection to tolerate the authorization of its use; it considers that this permission should be of a temporary character.

*Norway.*—The law authorizing the new carat bears date of May 27, 1910; the decree putting it in force was promulgated June 17 of the same year. The text of the law is as follows:

"The name 'metric carat' designates a special metric unity of mass, amounting to 200 milligrams, exclusively destined for the estimation of the price of diamonds, pearls, and other precious stones, and for their sale or purchase.

"The decimal multiples and subdivisions of the metric ca
be authorized in so far as they may be necessary.

"The word 'carat' shall be in the future exclusively rese
the designation of the mass above defined."

The decree establishes the series (similar to the metric
the multiple and subdivided weights of the carat. It also
that the carat-weights shall have the form of an equilateral
one side of which shall be turned up.

*Portugal.*—The "metric carat" (*quilate metrico*) is include
table annexed to the decree of Apr. 19, 1911.

*Roumania.*—A Royal Decree of Mar. 3, 1910, prescribes, fr
1, 1911, the use of the new international carat: the verificati
carat-weights is to be effected in conformity with the gene
comprised in the law concerning weights and measures.

*Russia.*—The reform of the carat is comprised in the gen
in course of revision.

*Servia.*—The same conditions obtain here as in Russia.

*Sweden.*—The law establishing the new carat was pron
June 10, 1910; the obligation to observe it was fixed for Jan.

*Switzerland.*—The carat of 200 mg. is comprised in the
weights and measures promulgated June 24, 1909.

As may be seen, the new international carat of 200 mg.,
or in course of adoption in 17 countries, is prescribed by law
slightly different texts, according to the sense in which the
viewed; in general, the prohibition implicitly contained in th
weights and measures touching the use of any non-metric
expressly affirmed as regards the carat in common use, but
ployment of the word "carat" to designate a mass of 2 deci
admitted as a toleration necessitated by the state of things, a
fied by the special conditions of the commerce in precious st

In those countries in which no legislation has fixed the
the carat, but where the metric system is in use, the toleratio
has been enjoyed by the ill-defined unity by which the w
precious stones has been computed, should *a fortiori* be appl
unity strictly determined by its simple relation to the gra
may therefore say that the reform is in reality more genera
indicated by the legal status of the carat.[6]

As a preliminary to the presentment of the law legalizing
of the new carat in Holland, the measure was submitted by
ister of Agriculture, Industry, and Commerce to the Amster

[6] Comité International des Poids et Mesures; *Precès-Verbaux des Séances*, P
2d ser., vol. vi., Session of 1911, pp, 202 to 205.

Vereinigung and to the Diamond Exchange of Amsterdam; ies concurred in advocating the change.

ormity with a resolution passed by the Fédération Interna- es Bourses du Commerce de Diamants, Perles et Pierres s of Antwerp, at the convention of 1912, Adolph Adler pre- the Minister of Industry and to Senator Braun petitions for zation of the new carat, and received from the latter the that he would urge the passage of a law to that effect in an Chamber of Deputies.

ng since some fifty jewelers met in the rooms of the National Board of Trade, in New York, to consider the advisability ng the new standard. Prior to the meeting, M. D. Röths- o was elected Chairman, had sent out 4,000 postal cards to welers in the United States, requesting their opinion on the the introduction of a new international carat of 200 mg. ut 360 replies were received; however, there was some con- n the fact that but nine of these were negative.

hat the new carat is likely to be soon in universal use, it well to consider whether jewelers should not substitute the e gram for that of the pennyweight. The grain is equiva- out 64.8 mg. (more exactly 64.798918), and the pennyweight 555 g., the troy ounce being 31.1 g., and the troy pound There can be little doubt that when the great convenience tation resulting from the adoption of the new international 00 mg. shall have been fully realized, the advantage of using instead of the pennyweight will become clearly apparent, as y the troublesome multiplications by 24, 20, and 12 will be y with. As five of the new international carats make exactly alculations regarding the weights of diamonds, pearls, and stones will then be in accordance with the metric system.

portant step in the direction of the official recognition of the was the sending of a communication by Secretary Nagel, of rtment of Commerce and Labor, to the Secretary of the recommending its adoption as the standard weight in com- e value of precious stones imported into the United States. of this letter was prepared by Director S. W. Stratton, of au of Standards, Oct. 29, 1912. After drawing attention to ion of the new carat by so many European countries, the excellent reasons are adduced for favorable action in the the Treasury Department:

arat is not the most important element in estimating the precious stones, particularly of the diamond, but it is never-

theless important that its value be fixed. The change fron now used by the Treasury Department to the one propo probably be of no significance in so far as the amount of d cious stones collected by the Department is concerned, bu be very important in its effect upon the unification of sta therefore have the honor to suggest that the Department use the international carat of 200 milligrams and thus set a which I feel sure will be followed by the jewelry trade in States."

Official steps have been taken by our State Department, gestion of the Treasury Department, tending to secure com by the governments of Great Britain, Holland, and Belgiu own in the employment of the new international carat of determine the weight of all precious stones exported or imp these countries, and the favorable action of the Treasury D now realized was foreshadowed by S. W. Stratton, Direc Bureau of Standards in Washington, D. C., in a letter of Jan

A number of representative dealers in precious stones m don, on Feb. 7 of this year, under the auspices of the Lond sale Jewelers and Allied Trades Association, Ltd., to co question of the new carat-weight. One of the speakers the meeting that the Board of Trade had been approach subject, and it appeared that its members thought favora chances of the adoption of the new system in England af more propaganda work had been done there. The Boar does not at present exercise any official control over the weights in use, but a legalization of the new carat would r proper testing and stamping of the new series of weights.

A strong point made in favor of the adoption of the m in England is that, according to the English Weights and Act of 1878, the old carat is an illegal weight, this act stipulating that gold, silver, and precious stones should o by the ounce troy, or decimal part thereof. Hence it has that, strictly speaking, a contract to buy or sell so ma weight of diamonds would be an illegal contract and not e The new metric carat, on the other hand, would be fully the Weights and Measures (Metric System) Act of 1897, therefore constitute a perfectly legal unit of mass.[7]

[7] See W. J. Lewis Abbott and Leonard J. Spencer, in *The Watchmaker, smith and Optician*, pp. 1447, 1449, 1451 (Dec. 2, 1912).

The carat-weight equaled essentially the Roman *siliqua*, $\frac{1}{1728}$ of a Roman pound, 2.91 grains troy, or 188.5 mg. It is interesting to note that at the present time this is the weight of the Bologna carat. The name is derived from the Greek *keration*, "little horn," and refers to the shape of the seed pods of *Ceratonia siliqua*, the carob-tree (St. John's Bread), the seeds of this tree having been used to weigh the precious material because their weight is fairly constant. The word carat has come to us through the Arabic *qîrât*, which became in Old Portuguese *quirate*, appearing in modern Portuguese and Spanish as *quilate*. A fourteenth century instance of the use of this word, under the form *garat*, to denote a pearl-weight is given in the *Nuremberg Chronicle*.[8]

The relation of the carat to the *siliqua* does not appear to have been constant, for Isidore of Seville, writing in the seventh century A. D., states that in his time a *cerates* equaled one and a half *siliqua*.[9]

The weight of the seeds of *Ceratonia siliqua*, as averaged from 50 specimens, has been given as 197 mg. (3.04 grains); the orange-red reniform seeds of *Erythrina corallodendron* had the same weight, while the lenticular seeds of *Adenanthara pavonina* gave a much higher average, namely 274 mg.[10]

The variations in the weight of the old carat are shown by the statement of Jeffries in 1751 that in his time there were *about* 150 carats in the ounce troy.[11] This would have given (if exactly 150) a carat value of 3.2 grains, or 207.357 mg.; later, John Mawe (in 1823) reckons 151.25 carats to the ounce, making a carat of 3.174 grains. The present English carat, having a value of 3.1683 grains, was already given by P. Kelly in 1831,[12] although it was not officially accepted by the English Board of Trade until 1888. Even now the English carat does not come within the scope of the Weights and Measures Act of 1878, while there is good reason to believe that the metric carat would have a legal status under the provisions of the Weights and Measures Act of 1897, touching the use of the metric system.

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[8] Grimm's *Deutsches Worterbuch*, vol. v., p. 73, art. Karat (Leipzig, 1873).

[9] Du Cange, *Glossarium mediæ et infimæ Latinitatis*, vol. ii., p. 286; s. v. Cerates (Parisiis, 1842).

[10] According to Leonard J. Spencer, a higher average weight for the seeds of *Ceratonia siliqua* has been given, namely, 205.2 mg. or 3.1667 grains, almost exactly that of the English carat; see G. F. Herbert Smith, *Gem Stones*, p. 84 (London, 1912). The average of 3.04 grains approaches very closely to the Florentine carat, 3.03245 grains; this was probably the carat-weight used by Tavernier.

[11] David Jeffries, *A Treatise on Diamonds and Pearls*, pp. 2, 3 (London, 1751).

[12] Patrick Kelly, *Universal Cambist* (London, 1831), vol. i., p. 220, where he writes: "The Ounce troy weighs 151½ Diamond carats, the carat is therefore 3⅛ grains troy or 205¼ French Milligrammes."

The following table of carat-weights heretofore in use in
ous countries will give the general reader some idea of th
conditions with which gem-dealers have been forced to cont

*Diamond Carats.*[13]

| | Milligrams. | Gra |
|---|---|---|
| Turin | 213.5 | 3. |
| Persia | 209.5 | 2. |
| Venice | 207.1 | 3. |
| Austro-Hungary | 206.1 | 3. |
| France (old) | 205.9 | 3. |
| France (later) | 205.5 | 3. |
| France (modern) | 205.0 | 3. |
| Portugal | 205.8 | 3. |
| Frankfort and Hamburg | 205.8 | 3. |
| Germany | 205.5 | 3. |
| East Indies | 205.5 | 3. |
| England and British India | 205.3 | 3. |
| Belgium (Antwerp) | 205.3 | 3. |
| Russia | 205.1 | 3. |
| Holland | 205.1 | 3. |
| Turkey | 200.5 | 3. |
| Spain | 199.9 | 3. |
| Java and Borneo | 196.9 | 3. |
| Florence | 196.5 | 3. |
| Arabia | 194.4 | 3. |
| Brazil | 192.2 | 2. |
| Egypt | 191.7 | 2. |
| Bologna | 188.6 | 2. |
| International carat of the year 1877 | 205.0 | 3. |
| New international carat | 200.0 | 3. |

We must note in this table the very wide discrepancy bet
heaviest carat-weight, that of Turin, equivalent to 3.2948 g
(213.5 mg.), and the lightest, that of Bologna, represen
2.91054 grains troy (188.6 mg.). Hence the Turin carat
more than 13 per cent. heavier than that of Bologna, an
difference when we have to deal with such costly commoditi
cious stones.

The impossibility of carrying on a diamond business syste
with such an appalling variation in the weight of the diamo
and with no possible means of an effective check to deter
accuracy of the weights employed, must be clear to all. I
that a great number of jewelers use a set of weights for a
20 years, and in the meantime these weights will either we
if they are handled and lifted with the fingers, as is often
they may become heavier.

[13] *The Book of the Pearl*, by George F. Kunz and Charles H. Stevenson, p
York, 1908).

rnment official informed me at Washington that a number weights used in various establishments were tried, and, to shment of every one concerned, a wide variation was found ue to their long use and to the varying standards of differalities. If we had a definite standard, a special set could or testing and checking up, once a month, or every three e weights in use, discarding those that varied. Thus, there much greater accuracy in the books of an establishment if nternational carat of 200 mg. were used.

e here a simple and correct rule for calculating the weight of diamonds under the new and the old standards. As a ry operation the number of sixty-fourths is to be expressed al fraction in the usual way, by dividing the numerator by ninator 64; for example, $\frac{1}{4}$, $\frac{1}{8}$, $\frac{1}{32}$, $\frac{1}{64} = \frac{27}{64}$; $27 \div 64 =$

international carat = 205 mg.
v international carat = 200 mg.
ntly:
carat is 2.5 per cent. heavier than the new carat, and the 2.44 per cent. lighter than the old carat.
carat weighing 2.5 per cent. more than the new carat, 2.5 should be added to price of new carat to obtain price of old

v carat weighing 2.44 per cent. less than the old carat, 2.44 should be deducted from price of old carat to obtain price rat.

grain being one-quarter of old carat and the new interna- n one-quarter of new carat, the same rules apply to grains. vert old carat-weights into new carat-weights, add 2.5 per eight of old carat; deduct 2.44 per cent. from price of old

vert new carats into old carats, deduct 2.44 per cent. from new carat; add 2.5 per cent. to price of new carat.

:

any new international carats and what would be the price of 97, $\frac{1}{4}$, $\frac{1}{8}$, $\frac{1}{32}$, $\frac{1}{64}$ old carats at \$100 = \$9,742.19?

| | | |
|---|---|---|
| decimals | 97.422 | |
| dd 2.5 per cent | 2.435 | |
| New carats | 99.857 | |
| rice per old carat | | \$100.00 |
| educt 2.44 per cent | | 2.44 |
| Price per new carat | | \$97.56 |

99.857 carats would give \$9,742.05.

How many old carats and what would be the price p
99.857 new carats at $97.56 = $9,742.05?

| | | |
|---|---|---|
| New carats | 99.857 | |
| Deduct 2.44 per cent | 2.436 | |
| Old carats | 97.421 | |
| Price of new carat | | $9 |
| Add 2.5 per cent. to price of new carat | | |
| Price of old carat | | $10 |

97.421 carats at $100 per carat = $9,742.10.

A useful publication in French to aid in making the
tions is that issued in 1912 by G. Ymonnet, and entitled
*relations entre le carat métrique à 200 mm. et le carat ancien*
The exact ratio of the English Board of Trade carat, th
recognized in Great Britain, to the new international ca
to 1.0265.

The fact that the general adoption of the new interna
owes so much to the activities of the Director of the I
Committee of Weights and Measures, the directing bod
ternational Bureau of Weights and Measures, suggests a
in regard to this Committee. Organized under a treat
Paris, in 1875, by 20 of the leading nations of the world
trusted with the important task of supplying the various
nations with standard platinum-iridium meters and kilo
forming strictly to their prototypes, the original meter an
recognized as standards by the French government in 17
in 1889, this task had been successfully accomplished, one
meters and one of the kilograms were chosen as internat
types, and each of the different countries represented in
received an allotment of one or more of these standard
and mass. Two meters and two kilograms fell to the
United States. These are now kept at the Bureau of St
Washington and constitute the fundamental metric standar
and mass for our land. The international prototypes i
carefully preserved in a specially constructed underground
are only accessible to the International Committee. To
maintenance of strict conformity between the different m
ards, the Committee contemplates the introduction of c
between them at such times as may seem expedient. T
tional Bureau has the joint support of the following cou
United States, Great Britain, Germany, Russia, Franc

Belgium, Argentine Confederation, Spain, Italy, Mexico, tugal, Rumania, Servia, Sweden, Norway, Switzerland, , Japan, and Denmark.[14]

s, in the rooms of the Chambre Syndicale in Paris, a stand-ce, which is verified once every month by the Maison Exu-cess can be had to this balance by any member at any time, ds materially in securing standard weights of absolute accu-ere are two great sources of error: one is the accretion of perspiration; the other is the wearing away of the weights on the pan of the balance, in the weight box, and by the n lifting them up.

iter had a set of the new international weights made in ese new weights, which will be required as soon as the national carat of 200 mg. shall be generally recognized oyed by American gem-dealers as the standard weight for tones, have already been made by a number of balance-The price charged for a set of these weights, 18 in number, om 100 carats to $\frac{1}{100}$ carat, is $3.80; of course, a fuller set t a proportionately higher price. It has been stated that ally no such weight as a "pearl grain;" yet while this is, eaking, a fact, the so-called "pearl grain" being merely a rat, the term has so generally been used and is so well d by all familiar with precious stones and pearls, that it id to be "consecrated by usage." In any case, however, ent that, with the introduction of the new carat, pearls can ed with greater accuracy, is a perfectly just one, for the of having a fractional weight as small as $\frac{1}{25}$ of a "pearl stead of being confined as at present to the larger fraction carat), is clearly manifest.

ute diplomatist, Talleyrand, is credited with the basic idea tric system, namely, the acceptance of the length of a pen-ting seconds on latitude 45°, as a standard unit of length. osition, made by him in 1790, was accepted by the French Assembly and the sovereign, Louis XVI. The details of n were worked out by the great French mathematicians grange, Laplace, Condorcet, and Monge, who constituted ee appointed by the French Academy of Sciences for this

l notation, without which the metric system would have been ibility, is said to be of Hindu origin, although it reached

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*rnational Metric System of Weights and Measures*, pp. 3, 4, Department of Com-bor, Bureau of Standards, S. W. Stratton, Director (Washington, 1906).

Europe through the Arabs. Abu Ja'far Mohammed b
named Al-Khowârazmi from his birthplace, Khwârazm (K
who flourished in the ninth century A. D., is considered
originator, or introducer of decimal notation among the
the introduction of the system into Europe has been t
translation of one of his works made in 1202 by Leonardo

The nine numerals with the zero were introduced among
about 773 A. D. and are by some believed to have been b
an Indian ambassador to Bagdad at that time. Of their u
at an earlier period we have monumental evidence in ro
tions, and, while the earliest date expressed in these Hindu
is 738 A. D., there is sufficient proof of their employment
the sixth century.

Introduced to Europeans at the beginning of the thirteen
the so-called Arabic numerals very gradually came into g
For example, in the calculation of the tides in London, i
teenth century. A work of Petrarch, printed in Cologn
has the pages numbered in this way.

The earliest example of the decimal point is said to be fo
Arithmetic of Frances Pellos, written in the dialect of
published at Turin in 1492.[15]

The first appearance of the symbols + and — in a pri
is in an arithmetic of Johann Widmann printed in Leipzig
and the sign denoting equality, =, is first found in print
Recorde's (1510?–1558) *The Whetstone of Witte, or the sec
Arithmetike*, London, 1557. He is believed to have adapte
similar sign used in medieval manuscripts as an abbreviati
Latin *est*, "is."[17] Rahn is credited with the first use of t
of division, ÷, and Thomas Harriot is said to have been
print the signs > and <, denoting respectively excess and
his *Artis analyticae praxis*, London, 1631.

A simple method of making a rough mental calculation of
tric weights and measures into equivalents of the English s
given by Albert A. Cary.[18] This may be in many cases ac

[15] *Sen segue de la art de arithmeticha, et semblatment de ieumetrica dich ho nom
de lo abaco (compilada es ea opera per Fr. Pellos), Impresso in Thaurino lo prese
abaco per meistro Nicolo benedeti he meistro Jacobino luigo de sancto germano-Ne
Di 28 de Septembrio.*

[16] Widmann, J., *Behede und hubsche Rechenung allen Kauffmanschafft*, K
Kacheloffen, 1489. 8°, 236 pp.

[17] The title "Whetstone of Witte" is the translation of *cos ingenii*, a punn
*cosa* (thing), early used to signify our algebraic *x*, or unknown quantity.

[18] Simple Approximate Metric Conversions, *Power*, vol. xxxvii, No. 7,
(Feb. 18, 1913).

by merely adding 10 per cent. to the figures expressing the metric quantity. For instance, 1 m. equals 1.094 yd. nearly; the error in turning meters into yards in this way amounts to a trifle less than $\frac{6}{1000}$ yd. in each meter. Thus 67 m. would equal 67 + 6.7 yd., or 73.7 yd., or, multiplying this by three, 221.1 ft.; the more exact equivalent being 73.27 and 219.82, respectively. With the kilogram the metric figures must be doubled and then 10 per cent. added to the product to get an equivalent number of avoirdupois pounds, as the kilogram equals 2.204 lb. avoirdupois. Hence 76 kg. can be thus roughly turned into pounds: $76 \times 2 = 152 + 15.2 = 167.2$ lb., the exact equivalent being 167.504 lb., a comparatively slight error when only an approximation is sought. An addition of 5 per cent. to the figures expressing a given number of liters would also offer a fairly close approximation to the number of quarts. If we take, for example, 64 liters, we have: $64 + 3.2 = 67.2$; here the exact equivalent is 67.629 quarts.

In a communication to the Société Française de Physique in Paris, Edouard Guillaume calls attention to a slight difference which exists between the liter and the kilogram or cubic decimeter of water. Accurate modern measurements have shown that the difference amounts to $\frac{27}{1,000,000}$ or a little over $\frac{1}{40,000}$. This very slight, but still appreciable, error is due to the fact that the liter represents the volume of the kilogram of water at its maximum of density under atmospheric pressure.

The metric system was legalized in the United States by the act of July 28, 1866, and although its progress towards popular and general recognition and employment has been unfortunately slow, at intervals since that date certain special official enactments have prescribed its use in particular cases, as in the postal service, where the post offices exchanging mails with foreign countries are provided with balances denominated in metric grams, under the terms of Section 3,880 of the Revised Statutes. A much more important case, however, because it concerns an exclusively national use of the metric system, was the enactment that the weight of the half-dollar, the quarter-dollar, and the dime should be computed in grams, the half-dollar to weigh 12.5 g. and the quarter-dollar and dime respectively one-half and one-fifth of this weight. In 1894 a further step was taken by the enactment that the international units based on the metric system should be "the legal units of electrical measure in the United States."

Already, by an order approved Apr. 15, 1878, the Secretary of the Navy had directed that the metric system should be used in the

Medical Department of the Navy, and later, Apr. 13, 1
ordained that all requisitions and accounting for medical
the War Department should be made in conformity with
standard. The scope of these regulations was broadened
tive orders of Nov. 21, 1902, providing that "for all officia
and pharmacal purposes, officers shall make use of the me
of weights and measures."

In our colonies, Porto Rico and the Philippines, the me
had been in use long before the date of our occupation, a
tinued use, for Porto Rico, was made obligatory by a procl
the military governor, dated Mar. 18, 1899, while in the
Philippine Islands the continuance of its employment wa
under Sections 3,569 and 3,570 of the Revised Statutes.

It is not only in the form of the new international car
metric system has found application in determining the
of precious stones. Certain classes of these stones are dis
and sorted by passing them through sieves graded accor
diameters of their apertures expressed in millimeters. The
used consists of a brass ring grooved on the inner side so
may be set within it a series of sieves. A sieve of a cer
having been duly adjusted, the stones to be measured are p
it, and shaken to and fro until all smaller than the apertu
particular sieve have passed through it. These stones
gathered up and placed upon a sieve with smaller ape
those which do not pass through this latter sieve are cla
serial number. Pearls are also sometimes classified in the

Here, however, as in the case of the carat, a fixed st
been lacking, there being Paris stone-sieves with a differen
$\frac{1}{5}$ mm., Paris and Idar "pearl-sieves" with a progressive
amounting to from $\frac{1}{25}$ to $\frac{1}{5}$ mm., and Bohemian garnet-
approximately the same progressive difference as the Pa
But this theoretical uniformity does not exist practical
sieves are not standardized, and those used by one dealer
usually differ from those used by another. Moreover, t
difference between sieve and sieve is not constant. In
some numbers, or grades, it amounts to $\frac{1}{8}$ mm., while in
others it is but $\frac{1}{10}$ mm., so that a great quantity of sto
placed in certain grades and very few in certain other g
obviate all these difficulties an international series of sieve
proposed. The No. 1 sieve is to have apertures of $\frac{1}{10}$ mm
ceeding number to have apertures $\frac{1}{10}$ mm. greater in d
that in the case of the No. 10 sieve this would measure 1

mm., and so on. By this means the number of the sieve would diately and invariably denote the diameter of the holes; in the of No. 17, for instance, the diameter would be $1\frac{7}{10}$ mm.; for 6, $3\frac{6}{10}$ mm. As another improvement, it is suggested that the be made as thin as possible, and that thin steel plates be sub- ed for brass as a material.

*Countries Which Have Definitely Accepted and Legalized the Metric Carat.*

| | |
|---|---|
| Spain | 1908 |
| Japan | 1909 |
| Switzerland | 1909 |
| Italy | 1910 |
| Bulgaria | 1910 |
| Denmark | 1910 |
| Norway | 1910 |
| Holland, law promulgated Apr. 7 | 1911 |
| Portugal | 1911 |
| Roumania | 1911 |
| Sweden | 1911 |
| France, adopted June 22, 1909; legalized Jan. 1 | 1912 |
| Germany | 1912 |
| Belgium | —— |

e following resolution was adopted at the Eighth Annual Con- ce on the Weights and Measures of the United States, held at ureau of Standards, May 14 to 17, 1913:

*esolved*, that this Conference is in favor of the metric carat t of two hundred (200) milligrams being adopted as the stand- weight for precious stones."

as been suggested that as every State having a special depart- of weights and measures should be properly equipped for the f testing the accuracy of weights and balances used for pre- stones, provision should be made for officially verifying such ts and balances, the owners merely needing to defray the costs nsportation, which would be inconsiderable.[19]

a letter sent in the latter part of May, 1913, to G. E. M. John- ecretary of the Decimal Association of London, some important nation is conveyed by P. A. MacMahon, deputy warden of the ards of the English Board of Trade, promising the general adop- f the new metric carat in England in the near future. He :[20]

A. Dazney, The Application of the Weights and Measures Question to the Jewelry *Jewelers' Circular Weekly* (Apr. 24, 1912).

*welers' Circular Weekly* (June 11, 1913).

"In reply to your letter of the 21st of May respecting the
of the metric carat in the sale of diamonds and other precio
I have to acquaint you that the Department has now decide
steps to make the metric carat and its necessary multiples
multiples standard weights in the United Kingdom, and
Order in Council giving effect to this decision will probably
this year."

On July 1, 1913, the provision goes into effect in Belgium
the metric carat the standard of weight for precious stone
country, and it is confidently expected that in Holland, w
extensive diamond interests are so nearly allied to those of
a similar provision will be very shortly enacted. It is thus
that by 1914 the six countries chiefly interested in the diamo
the United States, Great Britain, Germany, France, Holl
Belgium, will all be freed from the use of the antiquated
plicated carat-weights of the past.

In an interesting and instructive address delivered by
A. Fischer, of the United States Bureau of Standards, b
Retail Jewelers' Association of the District of Columbia, the
took occasion to assure his hearers that the jewelers of this
could count upon the assistance and support of the Bureau
ards if they followed the example set by their European con

For jewelers and gem-dealers doing business in New Yo
of considerable interest is that the Mayor's Bureau of Wei
Measures is ready, after July 1, 1913, to place its services a
posal of the general public in correctly determining the w
precious stones according to the carat of 200 mg. Com
John L. Walsh states that the mechanical department of the
at 224 West 49th Street, will then be supplied with faci
weighing gems from $\frac{1}{100}$ carat to 500 carats, in accordance
new standard. While no other city department is as yet ab
nish this service, the New York State Bureau of Weights a
ures, at Albany, and the National Bureau of Standards, at
ton, D. C., can verify sets of the new weights by mean
standard weights deposited there.

In conclusion, it is a pleasure to chronicle the definite o
ceptance of the new carat by the United States Treasury De
On June 17 instructions were issued by the Department to
of customs prescribing the metric carat of 200 mg. as the
unit of weight for imported diamonds and other precious st
for pearls; these instructions to take effect July 1, 1913.

[21] In a personal letter, Mar., 1913.

use of the new carat-weights has received the approval of the al Jewelers' Board of Trade, A. Henius, President; the Na- Jobbers' Association, and the American National Jewelers' ation; and each of these bodies has passed a resolution com- ng the metric carat and recommending that members should y it as a standard weight after July 1, 1913.

resolutions of the National Jewelers' Board of Trade, passed neeting of the Board of Directors, held Nov. 14, 1912, read as s:

er discussion it was moved, seconded, and carried:

hat the National Jewelers' Board of Trade most heartily endorse ovement of the Committee that was formed for the purpose of option of the decimal metric carat, and recommend to its full ership the adoption thereof, in accordance with the Resolution at the meeting held October 29th, and that the President be ized to appoint ten members of this Board to co-operate with mmittee to be appointed by Mr. Rothschild, and that the Sec- be instructed to notify every member of the Board of the taken here to-day; that the President or the Secretary com- ate the action of the Board to the proper officials of the National Jewelers' Association and National Jobbers' Association, re- ng them also to use their influence for the decimal metric carat ."

following letters and telegrams register the final success of the ational metric carat in our country:

(*Copy.*)

DEPARTMENT OF COMMERCE AND LABOR,
BUREAU OF STANDARDS.
Washington.

June 13, 1913.

eorge F. Kunz,
05 Fifth Avenue,
New York City.

AR DR. KUNZ:

g to acknowledge receipt of your letter of June 12, regarding loption of the metric carat. In reply, I would state that the the Treasury Department intends to issue regarding the metric was submitted to the Department of Commerce for comment riticism and has been passed upon by this Bureau. It is our

understanding that the Treasury Department will issue this
take effect on and after July 1st, 1913.

With kindest regards, I remain,

Sincerely yours,

(Signed) S. W. STRATTO

Di

(*Copy of Telegram.*)

WASHINGTON, D. C., June 18th,

Dr. George F. Kunz,
405 Fifth Avenue, New York.

Notice of the Treasury Department concerning adoption c
Tariff will be promulgated July first, according to word
from Treasury Department to-day. You are therefore at
publish Bureau letter.

S. W. STR

(*Copy.*)

TREASURY DEPARTMENT,
Washington.

June 16

Mr. George F. Kunz,
405 Fifth Avenue,
New York.

DEAR SIR:

Replying to your letter of July 12, asking if the Treasur
ment will adopt or reject the new international carat of 20
milligrams, on July 1, I am very glad to inform you that th
ment has adopted the new international carat, beginning Ju
collectors of customs have been so informed.

Respectfully,

JAMES F. CURTIS,
Assistant Sec

(*Copy.*)

French Telegraph Cable.

PARIS, June 20

George F. Kunz,
409 Fifth Avenue, New York.

Merci telegram communique a confreres accueillez fel
cordiales pour consecration ocurre a laquelle avez donne to
devouement.

LEON R

alue of pearls is computed by squaring the number of "pearl (quarter-carats) and multiplying the product by a figure de- l upon as the base value of the grain. To ascertain the exact a pearl having a certain weight in metric carats, we should ltiply the figures by four, which will give us the number of pearl grains," and then multiply this product by itself, the eing in turn multiplied by the figure representing the value s of a "base-grain." For example, in the case of a pearl g 3.64 metric carats and having a base-grain value of $6, we as follows:

$$3.64 \times 4 = 14.56 \text{ (number of "pearl grains").}$$
$$14.56 \times 14.56 = 211.9936.$$
$$211.9936 \times 6 = 1271.9616 \text{ (value of pearl in dollars).}$$

is to say, such a pearl would be worth $1,271.96. Several ave been published containing the squares and cubes, and oots and cube-roots, of a long series of numbers, and any one would prove useful in these operations.[22]

dition to the jewelers already noted, no one has been more gable in the introduction of the international carat of 200 mg. ns. Ch. Ed. Guillaume, Director of the Bureau Internationale ds et Mesures of Sèvres; Mons. Leon Rheims, President e Syndicale Négociants Pierres Précieuses of Paris; and in ntry, Dr. S. W. Stratton, Director of the Bureau of Stan- Department of Commerce and Labor, and Hon. James F. Assistant Secretary of the United States Treasury.

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*'s Tables of Squares, Cubes, Square-Roots, Cube-Roots, Reciprocals of all integral to 10,000* (London, 1882).

L. Crelle's *Rechentafeln* (Berlin, 1889).

n Elderton, *Tables of Powers of Natural Numbers; in Biometrika*, vol. ii. and iv., 0 (Cambridge, Nov., 1903).

ANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.
[SUBJECT TO REVISION.]

ION OF THIS PAPER IS INVITED. It should preferably be presented in person at the eeting, October, 1913, when an abstract of the paper will be read. If this is impossible, on in writing may be sent to the Editor, American Institute of Mining Engineers, 29 reet, New York, N. Y., for presentation by the Secretary or other representative of its ess special arrangement is made, the discussion of this paper will close Oct. 1, 1913, when f the *Transactions* will go to press. Any discussion offered thereafter should preferably n of a new paper for publication in Vol. XLVII. (with suitable cross references in both

# Shock Tests of Cast Steel.

BY JOHN H. HALL, NEW YORK, N. Y.

(New York Meeting, October, 1913.)

remont test for measuring the energy consumed in breaking d bar of steel is not so well known in this country as it to be. The test specimen used in this test is about $\frac{5}{8}$ by $\frac{1}{4}$ $1\frac{1}{4}$ in., notched with a hack saw on the lower side to cause to break at a fixed point. A weight of 10 kg. falling 4 m., vided with a knife or chisel which strikes the specimen, ne test piece by impact. The residual force in the falling fter it has broken the specimen is recorded by a suitable nd the difference between the total energy and the residual f the weight (in kilogrammeters) is the force consumed in the specimen.

perience of several years with the Fremont testing machine n its great value in readily detecting brittleness not revealed e. testing or slow bending testing. In this paper it is pro- discuss (1) the supersensitiveness of the test as applied to and (2) its value in revealing the effect of heat treatment rtain conditions.

ing coupons cut from castings heat treated by quenching and g, attention was drawn to the wide variations in the value of ont test of specimens cut from different parts of the coupon. er was of the form and dimensions shown in Fig. 1, and test pieces were cut from two parts of this coupon, marked in that figure. The results of the tests are shown below, nd *B* in each instance being from the same coupon:

| | *A.* Kilogram-meters. | *B.* Kilogram-meters. | Coupon. | *A.* Kilogram-meters. | *B.* Kilogram-meters. |
|---|---|---|---|---|---|
| . . . . . | 12.0 | 12.5 | 9, . . . . . | 27.5 | 32.5 |
| . . . . . | 11.0 | 32.5 | 10, . . . . . | 17.5 | 31.0 |
| . . . . . | 11.5 | 30.0 | 11, . . . . . | 14.5 | 31.0 |
| . . . . . | 20.0 | 33.0 | 12, . . . . . | 15.0 | 27.5 |
| . . . . . | 17.5 | 33.0 | 13, . . . . . | 19.0 | 30.0 |
| . . . . . | 17.5 | 20.0 | 14, . . . . . | 17.5 | 25.5 |
| . . . . . | 17.5 | 13.0 | 15, . . . . . | 15.0 | 30.0 |
| . . . . . | 10.0 | 30.0 | | | |

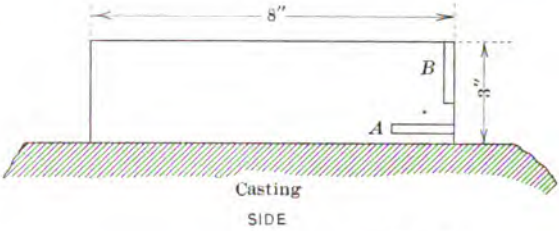


FIG. 1.

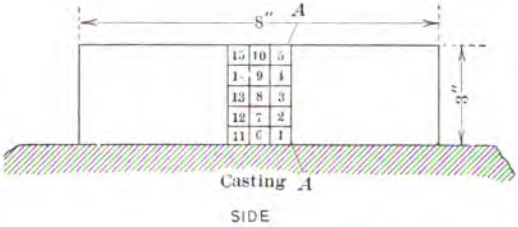


FIG. 2.

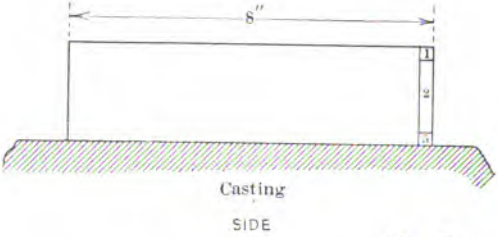


FIG. 3.

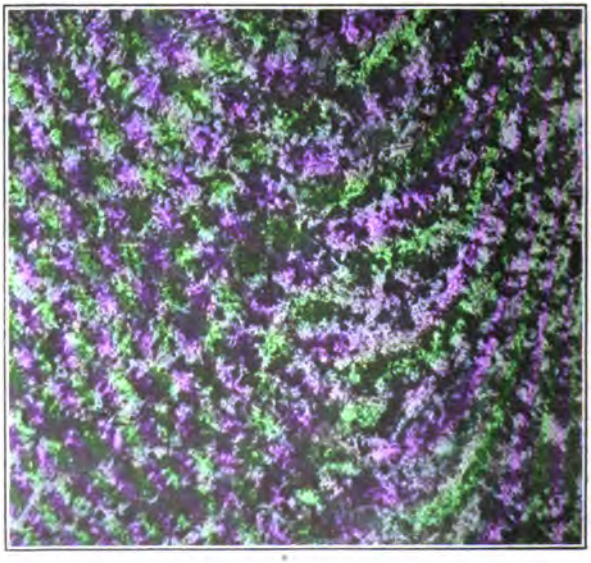

FIG. 4. × 60.

ently the toughness of the steel at the outside of the coupon
n greater than that of the steel from the part next the casting.
r to test this point, test pieces were cut from a single coupon,
vn in Fig. 2, and tested, with the following results:

| | Kilogram-meters. | Test. | Kilogram-meters. | Test. | Kilogram-meters. |
|---|---|---|---|---|---|
| . . . . . | 18.0 | 6, . . . . . | 10.0 | 11, . . . . . | 19.0 |
| . . . . . | 18.0 | 7, . . . . . | 17.0 | 12, . . . . . | 19.0 |
| . . . . . | 19.0 | 8, . . . . . | 19.0 | 13, . . . . . | 24.0 |
| . . . . . | 31.0 | 9, . . . . . | 33.0 | 14, . . . . . | 24.0 |
| . . . . . | 33.0 | 10, . . . . . | 31.0 | 15, . . . . . | 32.0 |

pieces cut from the end of the same coupon, as shown in
were tested, with the following results:

| | Kilogrammeters. | Test. | Kilogrammeters. |
|---|---|---|---|
| . . . . . | 32.0 | 4, . . . . . | 32.0 |
| . . . . . | 30.0 | 5, . . . . . | 10.0 |

evident from these figures that the Fremont test is capable of
ng variations in the toughness of steel due to very slight varia-
the rate of cooling in the first quench, inasmuch as, though
rence in microstructure can be detected in specimens from the
nd outside of the coupons, yet a marked superiority in tough-
shown by the outer ½ in. or so of the metal. For this reason,
as applied to cast steel in heavy sections is rather too delicate.
ite of this drawback, the shock test has a great value in re-
the brittleness inherent in cast steels which have a coarse
ructure. Two small castings, the microstructures of which
wn in Figs. 4 and 5, were analyzed, with results as follows:

| C. Per Cent. | Si. Per Cent. | Mn. Per Cent. | S. Per Cent. | P. Per Cent. |
|---|---|---|---|---|
| 18 | 0.41 | 0.93 | 0.065 | 0.052 |
| 20 | 0.45 | 1.02 | 0.057 | 0.049 |

ont tests of these castings gave:

| Test. | Kilogrammeters. | Test. | Kilogrammeters. |
|---|---|---|---|
| 4, . . . . . | 5.0 | 5, . . . . . | 7.0 |

how the properties of similar steel quenched and annealed
to 900° C. for 5 hr. and quenched, reheated to 680° C. for
and air cooled), the following test is given of a coupon from a
of similar analysis (C, 0.21; Mn, 1.17 per cent.) whose micro-
re is shown in Fig. 6:

| le th. | Elastic Limit. | Extension in 2 in. | Contraction. | Fracture. |
|---|---|---|---|---|
| n. | Pounds per sq. in. | Per Cent. | Per Cent. | |
| 00 | 71,650 | 20.37 | 50.25 | Silky. |

| Bend ¼ by 1 in. on 1-in. Mandrel. | Fremont Kgm. |
|---|---|
| 180° | 21.0 |

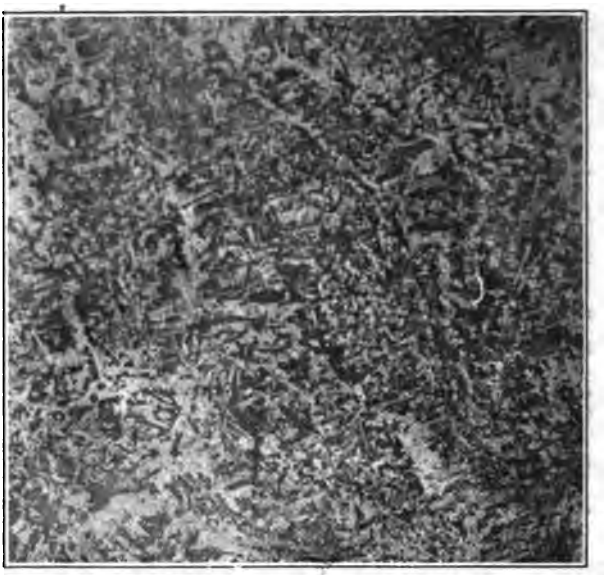

FIG. 5. × 60.

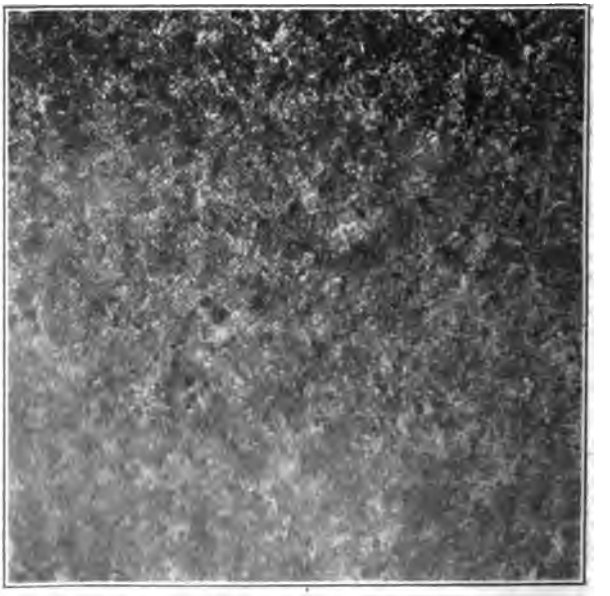

FIG. 6. × 60.

e casting whose microstructure is shown in Fig. 5 was afterward into two approximately equal parts, one-half set aside for test, her half heat treated in the same manner as the coupon which he tests just quoted (heated to 900° C. for 4 hr. and quenched, eheated to 680° C. for 8 hr. and air cooled). The pieces were

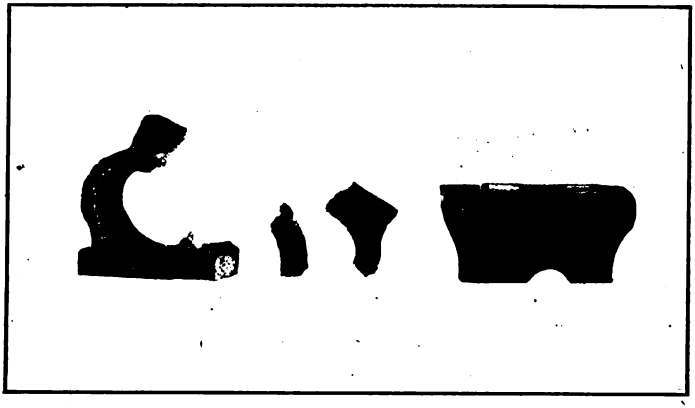

Fig. 7.

Fig. 8.

ubjected to the blows of a 500-lb. drop. The half in the al condition broke in three pieces with one blow of the drop a height of 4 ft., while the heat-treated half endured without ng one blow from 4 ft., one blow from 5 ft., one blow from 6 ft., ow from 7 ft., and two blows from 8 ft., six blows in all. Fig. 7 the two halves, and Fig. 8 a back view of the heat-treated half, esting.

FIG. 9. × 60.

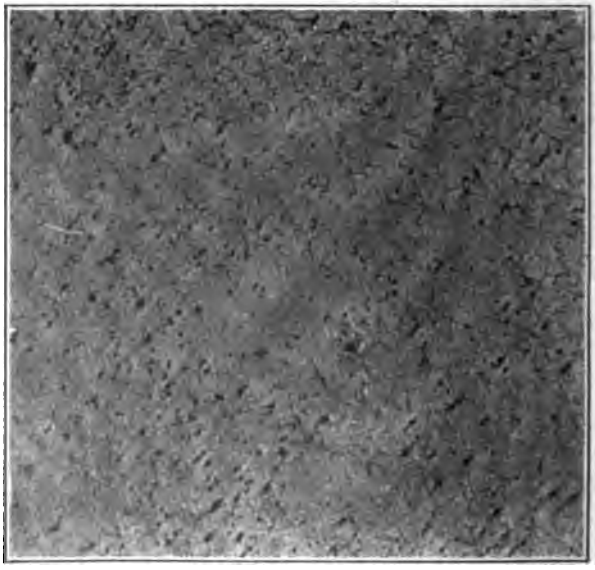

FIG. 10. × 60.

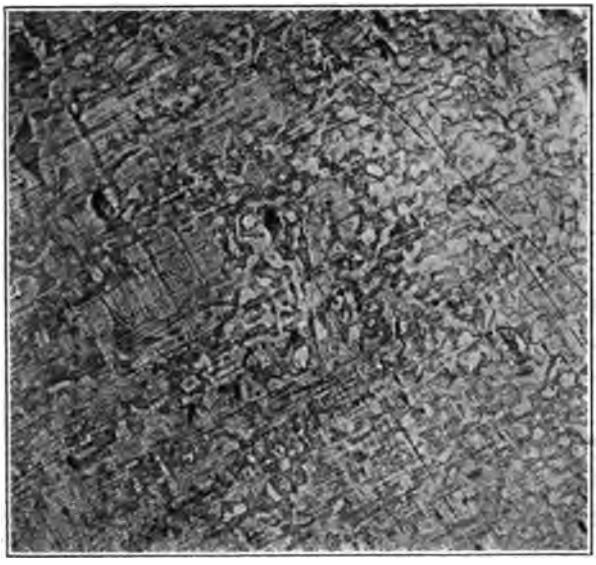

FIG. 11. × 60.

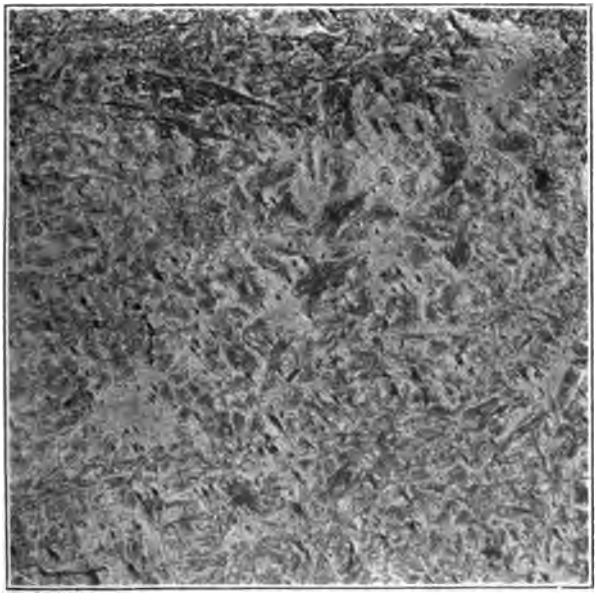

FIG. 12. × 60.

FIG. 13. ×

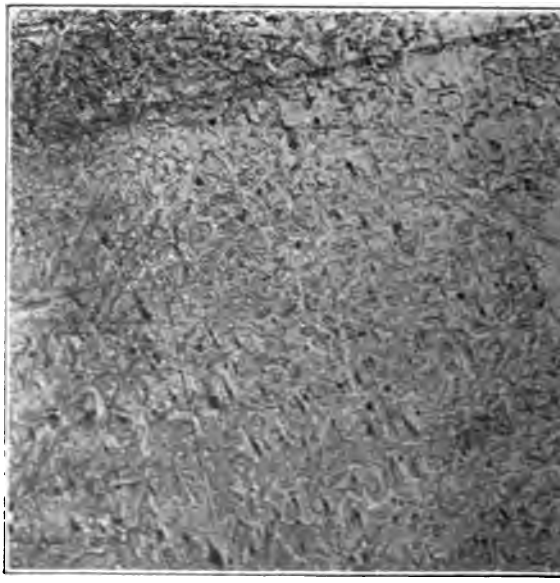

FIG. 14. ×

casting had no section over 1 in. thick, and averaged ½ in. all over. The results of the drop test thus confirm the Fremont which show that the steel annealed by heating and slow cooling, gives the coarse microstructures shown in Figs. 4 and 5, is brittle under sudden shock.

following tests, from coupons 2 by 3 by 8 in. cut from a single g, and heat treated in several different ways, show strikingly the of sudden cooling upon the shock toughness of even very mild The analysis of this steel was: C, 0.10; Si, 0.19; Mn, 0.23 nt.

ortunately, two of the coupons were not sound enough to give test bars of any value, and one bending bar contained a blow- The microstructures of these coupons are shown in Figs. 9 to d the heat treatments and physical properties in the accom- g table:

| Treatment. | Tensile Strength. Pounds per sq. in. | Elastic Limit. Pounds per sq. in. | Extension in 2 in. Per Cent. | Contraction. Per Cent. | Fracture. |
|---|---|---|---|---|---|
| Ione, . . . . . | 50,020 | 23,930 | 35.2 | 47.85 | Coarse cryst. |
| 00° 3 hr.; cooled in 40 min., | 55,390 | 28,750 | 38.5 | 67.0 | Silky cup. |
| 00° 3 hr.; cooled slowly, . | . . . | . . . | . . . | . . . | . . . |
| 00°; quenched in water, . . | . . . | . . . | . . . | . . . | . . . |
| 00° 3 hr.; cooled in air. 710° 6 h.; cooled in air, . . . | 56,450 | 32,950 | 39.25 | 65.2 | Silky cup. |
| 00° 3 hr.; quenched in water. 680° 8 hr.; cooled in air, . | 53,200 | 27,100 | 34.95 | 60.4 | Silky cup. |

| No. | Bend ½ by 1 in. on 1-in. Mandrel. Degrees. | Fremont Kgm. |
|---|---|---|
| 9, . . . . . | 50 | 2.5 |
| 10, . . . . . | 180 | 20.0 |
| 11, . . . . . | 45 (blow-hole) | 10.5 |
| 12, . . . . . | 180 | 27.5 |
| 13, . . . . . | 180 | 17.5 |
| 14, . . . . . | 180 | 25.0 |

s steel in the cast condition, if judged only by tensile test would ed fairly tough. The bending test gives a low value, and the ont test shows that under suddenly applied shock the steel is nely brittle, and offers almost no resistance to fracture. The s heat treatments are of about equal value as judged by the e and bending tests. The Fremont values, however, are lowest slowly-cooled steel, considerably higher in the air-cooled and led bar and the bar cooled at an accelerated rate, and highest in the quenched bars, more than twice as high as in the slowly- l bar. That a steel so low in carbon, silicon, and manganese as

this should be so greatly improved in resistance to shock
ing points to the great advantages of such treatment for ca
may be subjected to sudden severe stresses in service, su
chinery and automobile castings; and brings out strikingl
of testing under sudden severe shock in revealing degree
ness in even a "dead soft" cast steel.

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SION OF THIS PAPER IS INVITED. It should preferably be presented in person at the meeting, October, 1913, when an abstract of the paper will be read. If this is impossible, sion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 Street, New York, N. Y., for presentation by the Secretary or other representative of its nless special arrangement is made, the discussion of this paper will close Oct. 1, 1913, when of the *Transactions* will go to press. Any discussion offered thereafter should preferably rm of a new paper for publication in Vol. XLVII. (with suitable cross references in both

# coria Process for the Manufacture of Fine-Ore Briquettes, e-dust Briquettes, and Slag Brick for Building Purposes.

BY ERNEST STÜTZ, NEW YORK, N. Y.

(New York Meeting, October, 1913.)

problem of increasing blast-furnace efficiency through diminu- flue-dust production while operating with burdens consisting of fine ores has of recent years attracted the attention of rgists, whether directly or indirectly interested in the econo- iron smelting, and the various methods evolved from theory tice have been submitted to wide discussion. After the very h and, at that time, exhaustive view of the whole subject, vas afforded to the industry by the meeting of this Institute uary, 1912, the subject was again taken up at the meeting of ein deutscher Eisenhüttenleute at Düsseldorf in December, While American and German conditions are not entirely l, it may be of interest to give here the words of the gentle- mming up the discussion at the Düsseldorf meeting:

chief advantage offered by the use of briquettes lies in the t disturbances in the furnace, caused mostly by the presence ores in the burden, are avoided, and that blast pressure can be rably reduced. The resulting economies cannot be overrated, lower blast consumption per ton of product, the installation able to take care of increased production. In consequence of ressure and speed of blast, heat is more efficiently utilized, temperature is lowered, and coke particles are less liable to ied along with the flue dust, of which considerably less is d. Finally, by increased regularity of operation and higher ture in the lower parts of the furnace, the quality of the pro- improved. The question of briquetting iron ores has grown ne of the most important in the iron industry, and it is not any question of what is the cost of briquetting a ton of ore, but what, ore briquettes, is the cost of production of a ton of iron."

. Weiskopf, of Hannover, admirably condenses in these words

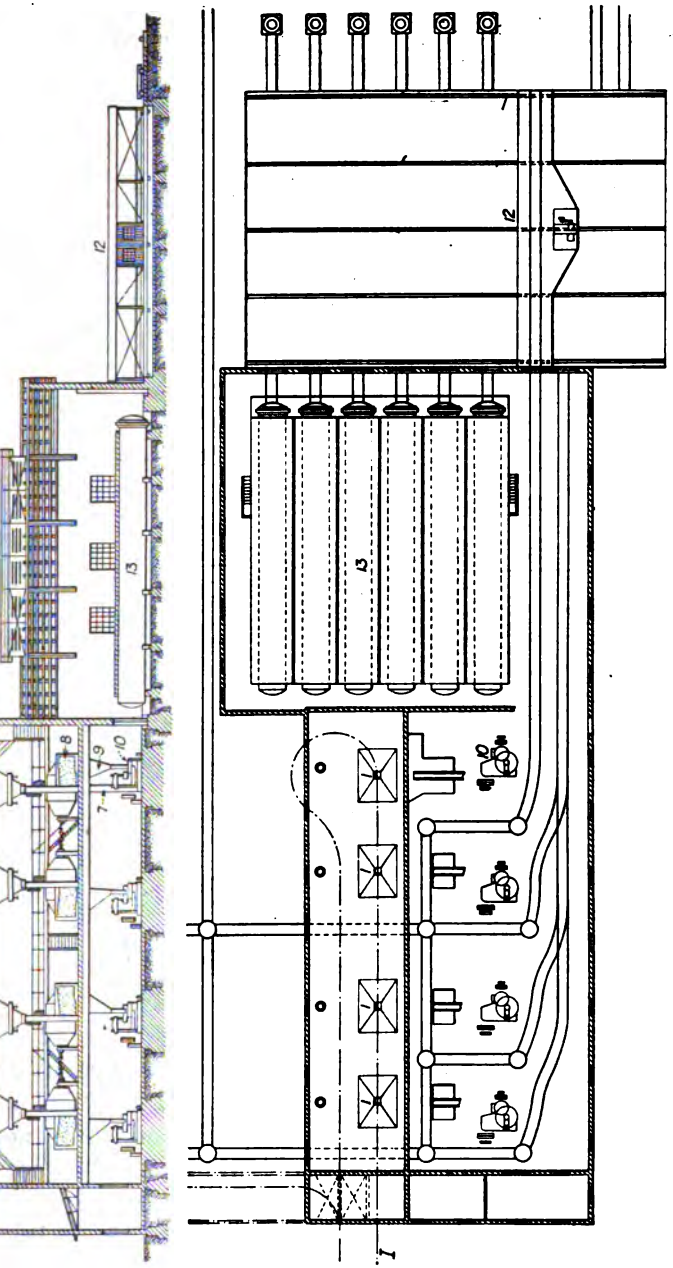
the reasons for the interest shown by the industry in th
of the most efficient means for turning the wasteful exp
dling and recharging flue dust into a productive factor in
improvement of blast-furnace practice.

The methods which have been developed for that purp
numerous; in Table I. their characteristics and comparat
reproduced in American values from a table published
*Eisen* of Feb. 13, 1913. Most of them have been men
scribed by Felix A. Vogel in a paper read before this In
was mistaken, however, in stating in the subsequent dis
the Scoria process "had not been in use for some time
more than two years the Scoria process had then be
operation on a scale of a daily capacity of 200 tons a
hausen works of Friedrich Krupp A. G., which, on the
the results then obtained, has now doubled its installatio

This makes it appropriate to give here a short acc
simple method, which, as its name implies, uses slag
preparing the valuable materials for smelting in the b
In other words, it uses a binder, but with the distinct
tween manufacture and use, the material so employed u
essential transformation, being hydraulic until it passes
nace and being there turned automatically into a fu
The metallic oxides will therefore be free to react duri
sage through the region of the top gases, and the bind
briquettes the necessary consistency to support the w
stock pile.

Figs. 1 and 2 show a plan and longitudinal and cros
an installation of four briquetting presses, each capable
out from 1,800 to 2,000 briquettes per hour, or about
to 12,000,000 in 300 working days of 20 hr. Taking
per briquette at 8.8 lb., this gives about 144 to 160 ton
working day, or about 43,200 to 48,000 tons of briquet
per year.

To summarize the process of manufacture, the granula
the lime are mixed with the valuable materials and ma
in rotating steaming drums, where they are exposed t
tension steam. When thoroughly mixed, sifted, and
powdery mass is conveyed to the presses to be formed
which are carried in trainloads of trolleys to cylindri
which they are hardened by being exposed to the act
tension steam for from 8 to 10 hours. The trainload
bricks goes straight from the kiln to the blast furnace




*I*, Conveyor from outside storage; 1, Storage bin; 2, Conveyor-hopper; 3, Steaming drum for moderate-tension steam to develop hydraulic qualities; 4, Storage bin; 5, Scraper plate to remove larger particles; 6, Grinding and mixing trough (*Kollergang*); 7, Bucket conveyor to 8, Sieve drum where foreign bodies, iron, etc., are thrown aside; 9, Storage bin for 10, Briquetting presses; 11, Trolleys, which are formed into trains and by means of 12, Traveling platform, are run into 13, Hardening kilns, where they remain for from 8 to 10 hr. exposed to high-tension steam of about 120 to 150 lb. pressure.

FIG. 1.—PLAN AND LONGITUDINAL SECTION OF BRIQUETTING PLANT USING THE SCORIA PROCESS.

dumped, unless required for immediate use. In the
illustrated, particular attention has been paid to an arran
which the switching operations, by means of a travelin
can be carried out in such a manner that the trolley c
moved only in entire trains.

It will be noticed that the manufacture is almost automa
necessary amount of steam is provided for, there is noth
process itself that requires supervision. The transportatio
cally all mechanical, the actual handling of the briquettes
essary only once: namely, in transferring them from the
trolleys before they are hardened. The labor required fo
lation of two presses consists of a foreman and 14 men
labor for steaming drums, grinding and conveying machine
hardening kilns, and trolley transportation.

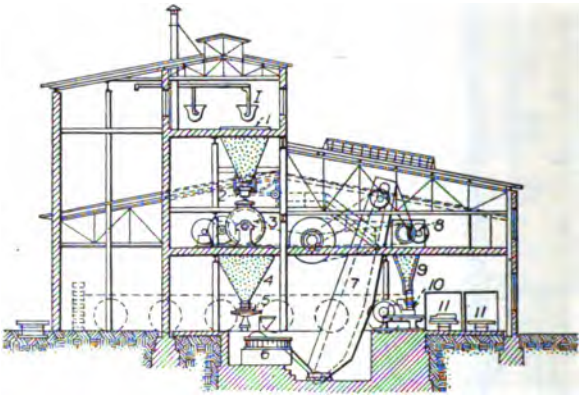


FIG. 2.—CROSS-SECTION OF BRIQUETTING PLANT.

The fact that in the Scoria process no air drying is n
serves particular attention, as it is thereby distinguished fro
briquetting processes; and the saving in space and handli
siderable factor in its economical advantages.

The running of the plant requires nothing in the way o
perience or skill, and the work is such that the labor regu
able at the plant can readily undertake it without special
Raw materials are always at hand at the plant and have
supplied from outside, and the success of briquetting does
on special conditions of the flue dust.

When the American Institute of Mining Engineers at
in February, 1912, devoted considerable time to the discu
merits of briquetting and sintering processes, the Scoria

nately not represented. I may be permitted, therefore, in this
add in its behalf something to that discussion and to inquire
me of the reasons which may have induced a works so highly
ed as Krupp's to discard another already installed process in
r. Of course, low cost counts for something, and the accom-
g table shows that in this respect the Scoria process leaves
to be desired in comparison to others. The administrative
ges have already been pointed out. But both these combined
hardly have given the Scoria a marked advantage over others,
esult of using Scoria briquettes on blast-furnace operation as a
had not been signally salutary. For instance, in a trial run of
days' duration, a proportion of 43 per cent. of the charge (bur-
us limestone) was added in the shape of Scoria briquettes
in any way interfering with the regular running of the furnace.
fact is, the hydraulic binder gives the body the necessary con-
at a minimum consumption of binding material, and, being
nature of a hydrosilicate, leaves the oxides freely accessible to
gases. It is idle to pretend that in the case of the Scoria
it would be necessary "to have enormous pressure on one
the briquette to produce an actual flow of gas through the
re." The force of the human lung is sufficient for that pur-
The objection may be all right with binders of a clayey nature,
h the Scoria binder it is theoretically unsound and practically
ed. On the other hand, the temperature necessary for the de-
n of the hydrosilicates is above that where sintering com-
, so that there is no possibility of the briquettes crumbling
rematurely.

her objection that has been brought against briquetting pro-
s that the coke that has been wasted through the top of the
becomes of no value in the briquetting process while it is util-
one or two of the sintering processes. True, the sintering
does use the coke contained in the flue dust, in order, as it
to predigest the food for the furnace. But is the furnace
ly a dyspeptic, that it should pay for the service of "predigest-
when it can get its food in its natural form? The Scoria pro-
turns all the coke blown out with the flue dust in undimin-
uantity and in the most suitable condition for assimilation in
nster's giant stomach. In that it is more economical than any
ompetitors.

herefore, the use of this binder is not a disadvantage as com-
o sintering processes, the only remaining criticism would bear
he question whether the remelting of the slight quantity of

slag in the furnace is a wasteful proceeding. Even were t
cost of melting, say, 4 or 5 per cent. of the weight of th
cannot be compared with the cost of coke wasted th
tendency of fine material to scaffold, quite apart from t
there are many occasions when the slag is positively bene
the utilization of ferro-manganese slag in order to increas
ganese content of basic pig iron.

Actually a very important coke economy will result fr
of the Scoria process. Working with a homogeneous an
piled stock column, the gases will circulate and pass up u
pressure, the combustion will be far more complete than i
possible, the production of carbon monoxide will be les
under those circumstances it has been proved by actual o
Germany that a saving of from 15 to 20 per cent. of cok
effected. This could be still further increased if the burd
consist of only lump ore more than 1.25 in. in size and
Under such conditions blast-furnace operation would be
ated that the output of the furnace could be enormously
The inventor of the process feels confident that it is well
means of possibility to so perfect operating conditions as
one ton of iron in a day for every cubic meter (35 cu. ft.) o
nace volume.

The Scoria process, based on sound theoretical consider
been carefully worked out scientifically, and it has stood
actual operations on a large scale. It has come to fru
slowly than some others, but for that reason it comes fo
with more matured experience. It may appear more c
than the use of sintered material, but when it is remembe
success does not depend on the "maintenance of absolute
conditions of adjustment as to mixture, . . . size o
compression, moisture, air blast, and heat treatment,"[1] bu
more of a humdrum workaday nature, it is evident that a
in that respect are misleading.

There yet remains to be mentioned, however, an advan
Scoria process which makes it particularly valuable to sm
furnace plants or those where the disposal of the slag i
problem.

The Scoria plant may be used to produce excellent sla
brick, and can alternate their manufacture with the pro
flue-dust or fine-ore briquettes, according to requirement

[1] *Metallurgical and Chemical Engineering*, vol. x., No. 9A, p. 596 (Sept. 1

notice. The operations are almost identically the same, except e flue dust or fine ore is left out of the mixture. Such have shown ample strength for all purposes, not only under laboratory tests, but also in actual use for building purposes a number of years. The cost of production, of course not g the slag as having a value, in Germany, on a two-press in- on capable of furnishing 24,000,000 brick per year is 5.30 or about $1.25, per thousand.

bricks were tested at the Royal Testing Laboratory, Gross- elde, by steeping in water and exposing to frost at 0° F (—17° 25 successive operations, without showing an appreciable loss gth.

further proof of the growing favor this process meets with I may mention the completion in April of a plant of a capac- 00 tons per 20-hr. day for the Aktien Gesellschalt für Hütten- Duisburg-Meiderich, the trial operation of which has turned atisfactory that a doubling of the installation is contemplated near future.

TABLE I.—*Manufacturing Cost for Various Bri*
*Sub*

(German v

| Name of Process. | 1. Material. | 2. Daily or Yearly Production | 3. Cost of Installation. | 4. Additions. Quantity and Kind. | Value Per Ton of Product. | Depre A. | B. Pe Ton Pro duc |
|---|---|---|---|---|---|---|---|
| | | Tons. | | | Cents. | Per Cent. | Cent |
| Ronay. | Flue dust. | 100<br>30,000 | $28,500 | ..... | ..... | 15 mach.<br>10 bldg. | 13. |
| Schumacher, ground quartz lime. | Flue dust and fine ores. | 200<br>60,000 | 50,000 | 10 per cent. lime, 5 per cent. quartz flour. | 27 | 10 mach.<br>2 bldg. | 7. |
| Schumacher, magnesium chloride. | Flue dust. | 100–120<br>30,000–36,000 | 16,000 | Magnesium chloride. | 12.5 | 10 | 4. |
| Scoria. | Fine ore and flue dust. | 200<br>60,000 | 25,000 | 4 per cent. blast furnace slag.<br>4 per cent. lime. | 10 | 10 | |
| German Briquetting Gesellsch. | Fine ore and flue dust. | 160<br>48,000 | 11,000 | 10 per cent. lime and cement. | 25 | 10 | 2. |
| Dahl. | Fine ore and flue dust. | 500<br>150,000 | ..... | 8 per cent. lime.<br>1 per cent. blast furnace slag. | ..... | 10 | 12. |
| Gröndal. | Magnetite. | 1 furnace<br>43<br>12,900 | 17,500 | ..... | ..... | 10 | 13. |
| Agglomeration, Fellner & Ziegler. | Fine ore. | 150<br>45,000 | 38,000 | ..... | ..... | 10 | 8 |
| Dellwick-Fleischer water-gas. | Fine ore. | 100<br>30,000 | 36,000 | ..... | ..... | 10 | 7. |
| Converter sintering. | Purple ore. | 1 Converter<br>10–30<br>3,000–9,000 | 4,800 | ..... | ..... | 10 | 6. |
| Zellpech. | Flue dust and ore. | 400<br>120,000 | 36,000 | 4 per cent. organic binder | 43 | 10 | 2. |
| Crusius. | Fine ore and flue dust. | 600<br>a18,000 | ..... | 4 per cent prepared bitumen. | ..... | 10 | 5 |

a So given in *Stahl und Eisen.*

The different columns give the follow g information :

Column 2 gives daily and yearly capacity (a year reckoned at 300 workin figures in the following columns are based on this.

Column 5 : A and B give depreciation according to figures furnished by co panies ; C and D, their reduction to a uniform standard of 10 per cen comparison.

*eration Processes Tabulated According to Cost Estimates*
*ucting Companies.*

ars and cents.)

| 6. sumption Ton. oduct. | | 7. Wages Per Ton. of Product. | | 8. Power, Steam, Repairs, etc. | 9. Remarks. | 10. Total Manufacturing Cost. | |
|---|---|---|---|---|---|---|---|
| B. | C. | A. | B. | | | A. | B. |
| nts. | Cents. | Cents. | Cents. | Cents. | | Cents. | Cents. |
| ..... | ..... | 10 | 11.5 | 18 | ..... | 41 | 38 |
| ..... | ..... | 15 | 18 | 18 | See *Stahl und Eisen*, Mar. 4, 1908, p. 322. | 67 | 72 |
| ..... | ..... | 16 | 16 | 8.5 | See *Stahl und Eisen*, June 22, 1910, p. 1061. | 41 | ..... |
| ..... | ..... | 10 | 11 | 9 | See *Stahl und Eisen*, Mar. 4, 1908, pp. 324–5. | 32 | 34 |
| ..... | ..... | 4 | 13 | 6.5 | Wages for 11 workmen per shift are given; installation cost is given too low, the company itself giving cost of product at 55 to 80 cents. | 37 | 55 |
| ..... | ..... | 16.5 | | 11.5 | ..... ..... | 63 | ..... |
| 43 | 25 | 30 | 40 | 25 | In Sweden cost of coal — $5,30 per ton; wages, 80 cents per shift. | $1.10 | $1.00 |
| 27 | 27 | 6.5. | 7 | 18 | See *Stahl und Eisen*, May 4, 1910, p. 759. | 60 | 60.5 |
| 29 | 29 | 16.5 | | | ..... | 52.2 | 52.5 |
| .5 .5 | ..... | 15 to 23 15.5 | 16.5 | + Depret'n. — Power. 5, 6, 7 | See *Stahl und Eisen*, Feb. 9, 1911, p. 245. | 30 to 50 | ..... |
| ..... | ..... | 2.5 | 2.5 | 6.5 | See *Stahl und Eisen*, Aug. 19, 1908, pp. 1196–9. | 53 | ..... |
| ..... | ..... | 16.5 | | 23 | ..... | 38 | ..... |

[b] 6 per cent. of coal for ignition.

[c] 6 per cent. of coke breeze or 10 per cent. of low-grade fuel.

n 6: Coal consumption in this column refers only to that of sintering processes. sumption for steam and power for other processes are included in Column 8. es under C are reduced to a common price of $2.85 per ton.

n 7 gives wages: under A, according to constructing works figures; under B, e assumption of an average daily wage per man of $1.38.

n 10 gives cost of production: under A, according to constructing works figures; comparative figures of values reduced to a common basis.

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# Precipitation of Copper from the Mine Waters of the Butte District.

BY J. C. FEBLES, BUTTE, MONT.

(Butte Meeting, August, 1913.)

## History.

use of iron for the precipitation of copper was known at least y as the fifteenth century. Both Paracelsus and Basil Valen- fer to it in their writings, as early as 1500 A. D. It was used n for this purpose prior to 1637, and at Rio Tinto, Spain, as s the sixteenth century.

statement has been made repeatedly that in the early history mining industry at Butte, the practicability of this method of ring copper from solution was overlooked, and that enormous ies of copper-bearing water were allowed to go to waste, the in which might easily have been recovered at small cost; and is oversight on the part of the mine operators was due to nce of this method of recovering copper. As a matter of fact, ttle copper-bearing mineral was encountered in the mines of until a depth of 400 or 500 ft. had been reached, and in many nuch greater depths were necessary to reach the copper-bearing

water encountered in the upper levels (although in most cases ighly mineralized) contained little or no copper, and it was only onsiderable areas of copper-bearing ore had been uncovered veloped that copper sulphate began to appear in the mine n any considerable quantity. In fact, the first water that was ed to contain copper carried it in such small amounts as to be e or no importance. It was only after years of underground had opened extensive areas of stopes, drifts, and other under- l workings, these being subsequently filled with waste, nearly all ch contained small copper values, which became oxidized by re to the air and were acted upon by the slowly percolating l water, that the recovery of copper from mine water became rcially important.

Below is given the analysis of a sample of mine wate
the Gagnon mine, analyzed by W. F. Hillebrand.[1] Fig
per million.

| | | | |
|---|---|---|---|
| $SiO_2$ | 29.5 | $Fe_2$ | |
| Ca | 512.1 | Mn | |
| Mg | 102.6 | Cu | |
| K | 11.4 | $PO_4$ | |
| Na | 82.8 | $SO_4$ | |
| $Fe_2$ | 0.4 | Cl | |

Although this water is not copper bearing, it is not h
it could easily become so if brought in contact with s
minerals.

At this time the Gagnon mine was pumping about 15
a minute, and some time subsequently a careful assay
showed 0.0016 per cent. of copper, which would be
about 28 lb. per day in the water pumped. An attem
copper from such water would not have paid the cost o
of the precipitating plant.

When mining operations had reached such a stage as
water to come in contact with copper salts, which, fr
had become soluble; then, and not until then, mine wa
carry enough copper in solution to make its recovery pr

The first production of copper precipitate from the
Butte was from the St. Lawrence mine. In November
broke out on the 400 level of the St. Lawrence, and
made to extinguish it by filling the mine with water, b
connected workings of other properties, it was found to
to make the water rise high enough to submerge the fire
this water was subsequently pumped out of the mine it
be very rich in copper sulphate. It has been state
pumping first began this water contained as much as 0
of copper. The statement cannot be verified, however,

When pumping operations were started for the pu
watering the St. Lawrence the water was allowed to fl
hillside and find its way to the gulch below. In doing
through the yard of a man named Miller, who was
living in East Butte. This Mr. Miller enlarged and cl
ditch which carried this rich copper-bearing water throu
placed scrap iron and tin cans in the ditch, and thus be

[1] W. H. Weed: Geology and Ore Deposits of the Butte District, Mon
*Paper No.* 74, *U. S. Geological Survey* (1912).

on of cement copper. Subsequently he improved his little by building some boxes and flumes of old lumber. He sold tate, in small lots, to the Colorado Smelter. His operations f short duration, however, and were carried on only upon a nall scale.

890 William Ledford secured from the Anaconda Copper Co. a lease on the water pumped from the St. Lawrence mine. gan operations by digging holes in the ground, which were filled crap iron and tin cans, and allowing the water to flow through oles and over the iron. After operating in this way for a time lt a flume of lumber, about 40 ft. in length, which he used in ction with the ground holes. The results obtained with the were evidently satisfactory, for he continued adding to this until he had a plant of sufficient size to enable him to abandon tation in the ground holes.

ford's operations were profitably carried on during the continu- f his lease. He sold his precipitate to both the Colorado and rrot Smelters.

he expiration of Ledford's lease, the Anaconda Co. decided to perate the precipitating plant, which it has continued to do up time.

he development of other mines became more extensive, furnish- r passages for the oxidation of low-grade copper sulphides in filling and other places, percolating waters were enriched in sulphate to such an extent that it became profitable to erect perate precipitating plants for its recovery. Consequently, the next ten years, most of the larger mining companies were bly operating such plants.

## Principles of the Process.

s well known, the principal reaction which occurs is the pre- ion of metallic copper at the expense of metallic iron, by gal- action, and may be expressed by the equation,

$$Fe + CuSO_4 = FeSO_4 + Cu.$$

resulting ferrous sulphate becomes oxidized by constant aera- the flumes and towers to ferric sulphate, which, being sub- to still further oxidation, is thrown out of the solution, both as erric sulphate and as ferric hydrate, with the liberation of free which reacts upon either copper or iron salts, again forming the ponding sulphates.

The quality as well as the quantity of the cement copp
in a given time depends upon a number of conditions, wl
briefly enumerated as follows:

1. The amount of copper in solution.
2. The temperature of the solution.
3. The freedom of the solution from salts other tha
4. The rate of flow.
5. The kind of precipitating apparatus used.
6. The kind of iron used.
7. The size and shape of the iron, and the way it i
8. The proper handling and cleaning of the iron.

1. It will require no argument to make plain that the
percentage of copper sulphate the solution contains, the
will be the rate of precipitation, and also the better the qu
precipitate.

2. The greater the temperature of the solution, the mor
deposition.

3. The freer the solution is from iron and other sulphate
rapid the deposition and the better the quality of copper

4. Within reasonable limits, the more rapid the flow
satisfactory the precipitation, both in quality and qua
most modern practice is to construct the plant with t
amount of fall to the first flumes in the system, in order
rapid flow at first. The fall is gradually diminished until
of the operation, the water is flowing quite slowly.

5. There has been much argument as to whether tower
will give the best results; but after long-continued use o
close comparison, the conclusion is that the flumes are sup
towers in almost every way. The first cost of the flume
can be more easily and economically handled; the cost
nance is less; the yield of copper per unit area is great
quality of the precipitate is just as good.

The tower is much more difficult to handle, and in co
is almost impossible to care for properly, owing to the f
ice in large masses in such positions as to make its remov
and in some cases impossible.

In fact, about the only advantage that can be claimed fo
is, that it permits the use of large and irregular pieces of
cannot be used to advantage in flumes. For example,
roll shells, trommels, and such cumbersome material c

in a tower, while their economical use in a flume would re-
ıtting or breaking, adding expense to handling.

most any merchant iron or mild steel will give satisfactory re-
ovided its surface is kept clean. The results obtained by the
ery hard steel, or spring steel, are not so good. Cast iron is the
of all, having little or no value for precipitating. The un-
ed carbon in the cast iron seems to interfere, to a very marked
with the reaction. The copper adheres very strongly to the
of the cast iron, making its removal so difficult that the sur-
ınot be properly cleaned. After the copper coating has be-
ıite thick it will break off in hard flakes and scales, with pieces
n adhering to them.

best iron for the purpose seems to be old rails, merchant bar,
e of small diameter. Horse shoes, punched screen, heavy
on, nails, bolts, and iron wire are very good; roofing iron,
, and similar scrap are not so good, but are extensively used
they are plentiful and cheap. All paper should be removed
ns before using. Painted iron and galvanized iron should be
before using; in fact, annealing at a low temperature im-
ılmost any iron for the purpose of precipitation.

re should be used in the distribution of the iron, in order to
the maximum precipitating surface. It should be packed
losely, but in such a manner as to allow the water to flow
Rails, pipe, and bar iron should be placed horizontally in
me, with short transverse pieces between the layers, to
pace for the free flow of the solution. Material of this kind
used to the best advantage in the upper part of the system,
he flow is most rapid and the water richest in copper. Smaller
nd cans may be placed indiscriminately. Flat sheets should
ed one above another, with enough small scrap between to
interruption of the flow, and in such a way as to present the
ım surface. Large and heavy pieces are used to the best ad-
in the towers.

Vhere rails, small pipe, and bars are used, and the flow is suffi-
rapid, very little or no handling of the iron is necessary, an
nal sweeping with a stiff brush being sufficient. The precipi-
opper, thus freed from the iron, will be carried to the settling
elow.

smaller scrap, sheets, wire, and cans, should be turned over
nally, with hooks and rakes, in order to remove the precipi-
opper.

iron in the towers requires very little attention, as in most

cases the fall of the water will keep the surface of the iro
free from copper.

Where the grade of the flume is not sufficient for t
carry the precipitate to the settling tank below, the flume
up at intervals by removing the iron (first separating all
possible from it) and the copper is sluiced into the settler.
flumes are constructed in two or more parallel sections, th
be handled through one section while another is being cle

After the expiration of Mr. Ledford's lease, the Anacon
tinued the precipitation of copper at the St. Lawrence
1901. About this time the High Ore pumping plant was
which enabled the company to handle the water of the e
through the High Ore shaft. A new precipitating plan
built, convenient to the High Ore. Since then practically
Anaconda Co.'s precipitating operations have been carrie
plant.

About the time the present plant was started (1901 or 1
experiments were made for the purpose of precipitating
by neutralizing the water with lime. The Gagnon mine
method of neutralizing the mine water, in order to protect
and columns from corrosion by the acid in the water.[2]

The use of lime in this way, at the Gagnon, probably su
use for the recovery of copper from mine water. This w
abandoned, however, owing partly to the low copper valu
sulting precipitate and partly to inability to handle the
economically. The precipitate is very light and floccule
sequence of which it settles very slowly, and is so sticky a
ous that it cannot be filtered.

The following shows the result of an analysis of a pre
tained by adding a 10 per cent. excess of quicklime to a
the High Ore mine water. The sulphuric acid was first
and the proper amount of lime to combine with it was ca
per cent. in excess of this amount being added:

| | Pe |
|---|---|
| Copper, | 8 |
| Iron, | 10 |
| Sulphuric acid, | 8 |
| Alumina, | 4 |
| Lime, | 15 |
| Magnesia, | 0 |

The filtrate from the above precipitate, upon being
found still to contain sulphuric acid. Upon the additi

[2] U. S. Patent No. 567,312, Sept. 8, 1896; C. W. Goodale and H. W

owever, nothing precipitated but the sulphates of lime and ia, and they came down very slowly.

lding a very large excess of lime and allowing to stand several filtrate can be obtained which is practically free from sul- acid. Analysis of a precipitate obtained in this way is given

| | Per Cent. |
|---|---|
| Copper, | 3.20 |
| Iron, | 2.00 |
| Sulphuric acid, | 3.50 |
| Alumina, | 0.71 |
| Lime, | 61.20 |
| Magnesia, | 2.16 |

where the precipitation is carefully made, and a large excess avoided, the precipitate obtained in this way is of little or no or the reasons stated above.

## The High Ore Plant.

water is pumped by steam and electric pumps to the 300-ft. here it runs by gravity through a tunnel, at the outlet of t enters the first set of precipitating flumes. There are three , placed parallel to each other. They average about 4 ft. in nd are 2 ft. deep. Their length is about 300 ft. to the first tanks, the average grade being 2 per cent.

e flumes are filled with selected scrap iron, such as rails, pipe, d bars, and such other iron as will give a rapid, clean precipi- his heading water being comparatively rich in copper. Under working conditions about 1,200 gal. of water per minute is d into these flumes, containing about 0.0500 per cent. of cop- he temperature of the water as it enters the plant is approxi- 85° F.

iron in these flumes is turned over and handled constantly, in separate the precipitated copper and leave a clean iron surface her precipitation. Every few days the water is shut off from t, allowing the remaining two units to carry all, so that the ay be temporarily removed, while the precipitate is washed settler below. This work is done in rotation, so that all receive an equal amount of attention.

passing the first settling tanks the water continues in the as described above, the only change being that the grade is d to about 2.5 per cent. and the flumes are increased in width t 8 ft. Settling tanks are placed approximately 75 ft. apart

from this point for a distance of 500 ft., which, with th
flume above the first settlers, constitutes the first unit of

These settling tanks are merely large wooden boxes,
width as the flume, about 15 ft. long and 8 ft. deep, and
that the precipitate can be sluiced through a hole in the
a trough connected with drying vats which are convenie
the tops of the drying vats being somewhat lower than th
the settlers.

This first unit, which consists entirely of flumes, is ap
800 ft. in length, and has a total fall of 30 ft., or an avera
3.75 per cent. The average velocity of the water in this p
plant is 60 ft. per minute.

The water, after leaving this first set of flumes, again pa
settlers, from which it is discharged into the first tow
merely a heavy wood frame, the main supports of which
square timbers. It is filled with pieces of scrap iron wh
large, or for some other reason are unsuitable, for use in
Baffle boards are nailed to the sides and ends, to prevent
water from splashing outside the tower. The dimensions
are 130 by 8 ft. by 19 ft. high. with a settling tank be
extends the entire length.

A summary of the work accomplished by the above des
and tower is given below:

| | |
|---|---|
| Temperature of water entering flume, | |
| Temperature of water entering tower, | |
| Temperature of water leaving tower, | |
| Per cent. of copper in water entering flume, | |
| Per cent. of copper in water entering tower, | |
| Per cent. of copper in water leaving tower, | |
| Per cent. of copper extracted by flume, | |
| Per cent. of remaining copper extracted by tower, | |
| Per cent. of copper extracted by flume and tower, | |
| Total time of water in flume, | |
| Average per cent. of copper in precipitate from settler above tower, | |
| Per cent. of copper in settler under tower, | |

The water then passes through another tower, from wh
two large parallel flumes, 9 ft. wide by 300 ft. long.
flume is filled with tin cans and small scrap. The tota
flume is 4 ft., or a 1.3 per cent. grade. The velocity of
this point is about 50 ft. per minute, the time consume
through being about 6 min. The water entering this s
assays 0.0019 per cent. of copper, and leaving, 0.0017 p
extraction in this flume being 10.52 per cent. The preci
by this flume contains 12.50 per cent. of copper.

water, upon being discharged from this flume, passes through er towers, the first of which is a large one, being 81 by 11 ft. . high. The remaining four are smaller, averaging only 8 or 9 eight. From the last of these towers the water enters a settler 8 ft. by 4 ft. deep, from which it is allowed to run to waste. wing is a summary of the work done by the last four towers:

| | |
|---|---|
| ent. of copper in water entering, | 0.0017 |
| ent. of copper in water at outlet, | 0.0010 |
| ent. of copper extracted by last four towers, | 41.2 |
| extraction of plant (per cent.), | 98.6 |
| perature of water at outlet, | 54° F. |
| ent. of copper in precipitate from last settler, | 8.60 |
| ent. of iron oxide in precipitate from last settler, | 47.71 |
| ent. of alumina in precipitate from last settler, | 8.69 |
| ent. of silica in precipitate from last settler, | 7.60 |

following table shows the result of a complete analysis of both d tail water of the High Ore precipitating plant, arranged in way as to be easily compared:

| | | | | No. 1. | | No. 2. | |
|---|---|---|---|---|---|---|---|
| | No. 1. Heading Water. | No. 2. Tailing Water. | Gain + or Loss — of No. 2 as Compared with No. 1. | All Bases Calculated to Normal Sulphates. | $SO_3$ Required. | All Bases Calculated to Normal Sulphates. | $SO_3$ Required. |
| . . . | 0.0504 | 0.0384 | — 0.0120 | 0.0504 | . . . . | 0.0384 | . . . . |
| . . . | 0 6683 | 0.0125 | — 0.6558 | 1.3405 | 0.6722 | 0.0251 | 0.0126 |
| . . . | 0.2102 | 0.1926 | . . . . | 0.7046 | 0.4944 | 0.6452 | 0.4526 |
| . . . | 0.5931 | 0.0729 | . . . . | 1.4845 | 0.8914 | 0.1826 | 0.1097 |
| . . . | 0.2194 | 1.0427 | . . . . | 0.4637 | 0.2443 | 2.2035 | 1.1608 |
| . . . | 0.0714 | 0.0574 | — 0.0173 | 0.1519 | 0.0805 | 0.1151 | 0.0610 |
| . . . | 0.6969 | 0.4679 | — 0.2290 | 1.3823 | 0.6854 | 0.9281 | 0.4602 |
| . . . | 0.3224 | 0.3296 | + 0.0072 | 0.7825 | 0.4601 | 0 8000 | 0.4704 |
| . . . | 0.1200 | 0.1376 | + 0.0176 | 0.3580 | 0.2380 | 0.4106 | 0.2730 |
| . . . | 3.5246 | 2.7614 | — 0.7632 | . . . . | . . . . | . . . . | . . . . |
| . . . | 0.0724 | 0.0862 | + 0.0138 | . . . . | . . . . | . . . . | . . . . |
| . . . | 6.5491 | 5.1959 | . . . . | 6.7184 | 3.7663 | 5.3486 | 3.0003 |

| | No. 1. | No. 2. |
|---|---|---|
| mal sulphates by evaporation (Fe oxidized), | 6.9856 | 6.1504 |
| phates by calculation (Fe oxidized), | 6.8675 | 6.0454 |
| e (alkali sulphates?), | 0.1181 | 0.1050 |
| suspension as "ocher" (included in above Fe), | 0.2250 | 0.0683 |

(Analysis by Charles M. Palmer.)

n the settling tanks become filled with precipitate it is sluiced e drying vats, where, after sufficient time has elapsed for set-

tling, the supernatant water is decanted and the ceme allowed to dry by evaporation until sufficiently compac dled. The precipitate, still containing considerable sacked and shipped in car-load lots to the smelter for ment. Below is given an average analysis of cement co

| | Per Cent. |
|---|---|
| Moisture, | 12.2 |
| Copper (electrolytic), | 70.9 |
| Silica, | 2.3 |
| Oxide of iron (ferrous), | 8.4 |
| Alumina, | 2.9 |
| Lime, | 0.3 |
| Sulphur, | 0.8 |
| Arsenic, | 0.27 |

FIG. 1.—LEONARD MINE PRECIPITATING PLANT, BUTTE, M FRONT VIEW OF TOWER.

Working under conditions similar to those here des plant will produce approximately 2,200,000 lb. of pure c ally, which amounts to the shipping of about 10 cars of cipitate each month.

The iron consumed in the process weighs from 1.1 to much as the copper produced. The lower the value in the higher the ferric sulphate or free acid, the greater th tion of iron. Good, clean, compact iron will precipitate

FIG. 2.—VIEW OF LOWER FLUME.

FIG. 3.—VIEW OF UPPER FLUME.

than tin cans and similar light iron. At the Silver Bow plant, where a large percentage of railroad rails and similar desirable iron is used, 1,700 lb. of copper recovered per ton of iron used is claimed. A fair estimate for normal conditions would be 1 lb. of copper recovered for each 1.5 lb. of iron consumed. At the Silver Bow plant it is claimed to be cheaper to use old rails at $11 per ton than cans and miscellaneous scrap at $7.50 to $8.

A full description of the Leonard and Silver Bow plants would be largely repetition, as in the main they are very similar to the one just described. Views of the tower and flumes at the Leonard plant are given in Figs. 1 to 3, but the following table will give a good idea of the comparative size and performance of the three plants, these being the principal ones now in operation:

| | High Ore Plant. | Leonard Plant. | Silver Bow Plant. |
|---|---|---|---|
| Total length of flumes (excluding towers), feet, | 1,769 | 1,943 | 805 |
| Total fall, feet, | 180 | 87 | 56 |
| Average fall per 100 ft., feet, | 10.25 | 5.6 | 7 |
| Total height of towers, feet, | 100 | 38 | 20 |
| Total length of towers, feet, | 630 | 224 | 25 |
| Average width of towers, feet, | 11.45 | 12.7 | 7 |
| Total area of towers, square feet, | 7,213.5 | 2,315. | 175. |
| Average width of flumes, feet, | 12.5 | 5.7 | 4.9 |
| Total area of flumes, square feet, | 22,169 | 13,400 | 3,908 |
| Average temperature of water entering, | 29°C = 85° F | 36° C = 97° F | 24° C = 75° F |
| Average temperature of water leaving, | 12°C = 54° F | 11°C = 52° F | 17° C = 62° F |
| Average flow of water, gallons per minute, | 1,200 | 1,500 | 356 |
| Average velocity in flumes, feet per minute, | 70.8 | 61 | 50.25 |
| Total time of contact, minutes, | 26.5 | 29 | 16 |
| Average copper content entering, per cent., | 0.0516 | 0.0482 | 0.0748 |
| Average copper content leaving, per cent., | 0.0010 | 0.0020 | 0.0082 |
| Copper extracted, per cent., | 98.6 | 95.85 | 90.37 |
| Total length of settlers, feet, | 938 | 1,150 | 128 |
| Average width of settlers, feet, | 11.31 | 7.65 | 9.3 |
| Total area of settlers, square feet, | 10,613 | 7,781 | 1,188 |
| Total precipitating area, square feet, | 29,382 | 15,725 | 4,300 |

## Other Products Precipitated.

The presence of the ferric hydrate, or "ocher," in the precipitate is objectionable, and is to be avoided as far as possible. There does not seem, however, to be any satisfactory way to control its precipitation. The formation of this precipitate is due to oxidation of iron in solution, probably by aeration, which results in its being thrown down both as ferric hydrate and as basic ferric sulphate.

The precipitation of this "ocher" is less at the Leonard plant than at any of the others, owing, most likely, to the presence of small particles of organic matter in suspension in the water, which has a

ency to prevent, or at least to retard, the oxidation of the ferrous
.

n analysis of the "ocher" from High Ore tail water gave the
wing results:

| | Per Cent. |
|---|---|
| Silica, | 3.11 |
| Copper oxide, | Trace |
| Alumina, | 1.71 |
| Ferric oxide, | 66.72 |
| Zinc oxide, | Trace |
| Manganese oxide, | None |
| Lime, | None |
| Magnesia, | None |
| Sulphuric anhydride, | 11.51 |
| Water, | 16.95 (calculated for limonite) |

alculating all $SO_3$ to basic ferric sulphate ($Fe_2(SO_4)_3$: ($Fe_2O_3$ + O) and the remaining $Fe_2O_3$ to limonite (2 $Fe_2O_3$: 3 $H_2O$), the lt is as follows:

| | Grams per Liter. | Per Cent. |
|---|---|---|
| Silica, | 0.0160 | 3.11 |
| Alumina, | 0.0088 | 1.71 |
| Limonite, | 0.3084 | 59.95 |
| Basic ferric sulphate, | 0.1812 | 35.23 |
| | 0.5144 | 100.00 |

he "ocher" was filtered from water which had stood in the labora- about 10 days, and had been frequently exposed to the atmosphere. amount of "ocher" obtained was 0.5072 g. per liter, dried at ° C.

he mine waters of Butte contain small amounts of arsenic, part of ch is precipitated with the copper. The average of four weekly ples of Leonard mine water gave the following results for copper arsenic:

| | Copper. Grams per Liter. | Arsenic. Grams per Liter. |
|---|---|---|
| Heads, | 0.236 | 0.00125 |
| Tails, | 0.017 | 0.00082 |

he amount of water pumped during the month was 37,000,000 the copper and arsenic precipitate being:

Copper, 0.0219 per cent., or 68,000 lb.
Arsenic, 0.000043 per cent., or 135 lb.
The ratio of arsenic to copper in the mine water is 1 : 189.
The ratio of arsenic to copper in the precipitate is 1 : 509.

his would give 0.135 per cent. of arsenic in a precipitate assaying er cent. of copper.

An analysis of mine water, made at the Great Falls Sme
results as follows:

| | Grams per Liter. |
|---|---|
| Free acid, | 0.1200 |
| Arsenic, | 0.00915 |
| Antimony, | 0.00570 |
| Copper, | 0.1573 |
| Total iron, | 0.2310 |

This analysis shows that the ratio of arsenic to copper in
water was about 1 to 17, and on this basis, if all the arse
down with the copper, a product assaying 65 per cent.
would run 3.8 per cent. of arsenic. As analysis of the p
shows no such proportions, it would appear that only pa
arsenic is precipitated from the solution.

Four samples of precipitate were taken from the High
at different places, and sent to the Great Falls Smelter fo
in order to show the relation of copper to arsenic precip
various places. The results of these analyses are as follows

| | Copper. |
|---|---|
| | Per Cent. |
| Precipitate from first settler, 300 ft., | 73.54 |
| Precipitate from settler, 900 ft., | 48.74 |
| Precipitate from settler, 1,400 ft., | 27.43 |
| Precipitate from last settler, | 8.60 |

This seems to show conclusively that the arsenic does no
tate proportionally with the copper.

Large samples of both head and tail water from the
plant were evaporated, and silver assays made on the residu
ing more than traces of silver were obtained from eithe
although a precipitate containing from 70 to 80 per cent.
will carry from 0.3 to 0.4 oz. of silver per ton.

## Costs.

An average of cost figures, extending over one year's
of the Leonard plant, is as follows:

| | Per Pound of Copper Produced. |
|---|---|
| Labor, | $0.0352 |
| Supplies and iron, | 0.0140 |
| Sundries, | 0.0002 |
| | $0.0494 |

The above figures are upon the basis of the net pounds
contained in the precipitate shipped and paid for.

## Mine Pumps and Columns.

ief description of the methods employed in Butte for the on of the pumps and water columns from the acid water may terest.

vater ends of all pumps are constructed, as far as possible, of phor-bronze alloy analyzing approximately 85 per cent. of and 15 per cent. of tin. Experience has shown that this alloy nd the corrosive action better than any material as yet tried. gers and valves made of this bronze are not altogether proof the action of the water, however, having to be replaced from time. The corroded parts are, upon removal, sent to the e department for repairs, a large supply of interchangeable ing kept on hand.

columns are constructed of cast-iron pipe, lined with either wood. The lead-lined pipes are flanged, the lead lining so that when bolted together, with a rubber gasket between, tight joint is insured, and the iron casing is completely pro- rom the water flowing through. The wood linings are made ow strips, the last strip being driven firmly home, forming a hich holds the lining in place. The whole inside is then with acid-proof paint, and all joints are carefully calked and to prevent leakage. The exterior of all pump columns is d with acid-proof paint.

## Leaching Dumps and Tailings.

ay be of interest to state, in closing, that after the tail water, hich the copper has been recovered, has been discarded, it is d by parties, who use it for leaching the soluble copper from e dumps, which are being operated under lease. These old contain small amounts of copper, which was originally in the sulphide; but owing to years of exposure to weathering, has more or less oxidized to sulphate.

tail water from the precipitating plants, containing consider- ric sulphate, is quite suitable for leaching this material. Thus the copper still in the form of sulphide, in addition to all the sulphate, can be recovered, owing to the solvent action of the ulphate upon sulphide and other copper minerals.

method of procedure is to convey the water to the top of the where it is confined in shallow ponds, thus causing it to per- through the dump. A trench is dug around the bottom of np, from which tunnels are run into the interior. The perco- water, collected by these tunnels, is conveyed to the trench,

from the sump of which it is pumped by electrically drive ugal pumps to the top of a precipitating system simila described above. Water containing as high as 0.015 pe copper is obtained in this way, and, while not as satisfact mine water, is profitable to work.

## Discussion.

Willis T. Burns, Great Falls, Mont. (communication to tary*):—In connection with Mr. Febles's interesting paper count of an attempt made at the Great Falls electrolyt recover the copper from the mine water by direct electroly of interest.

A barrel of mine water of the following composition wa from the Leonard mine, at Butte, Mont.

| | Grams per Lite |
|---|---|
| Free sulphuric acid, | 0.1200 |
| Arsenic, | 0.00915 |
| Antimony, | 0.00570 |
| Copper, | 0.1573 |
| Iron, total, | 0.2310 |

The insoluble anode tanks in use at the Great Falls e plant are 9 ft. 7 in. long, 2 ft. 4 in. wide and 3 ft. 9 in. de These tanks contain 22 lead anodes and 21 copper cathode operated at a current density of approximately 40 amperes foot, with a drop in electromotive force of 2.4 volts per tan

Not having sufficient mine water to permit the use of ta size, a small tank, 14 in. long, of the same proportions as t tanks, was used. The mine water was treated in this tan rent density of 40 amperes per square foot of cathode sur circulating the solution at a speed of 1 gal. per 32 ft. of ca face per minute, equivalent to 8 gal. per minute in a sta tank operating at 40 amperes per square foot.

While operating a tank under these conditions it was 44 volts per tank were required.

The following figures show the copper contents of the fore and after treatment:

| | Copper Grams per Lite |
|---|---|
| Entering tank, | 0.1573 |
| Leaving tank, | 0.0629 |
| Grams per liter removed, | 0.0944 |
| Per cent. extraction, | 60. |

The material deposited was a fine black slime contain cent. of copper.

* Received June 9, 1913.

he water were treated in a standard sized tank at 40 am-
per square foot while circulating at the rate of 8 gal. per
e the power consumption would be 10,000 amperes at 44 volts,
kilowatts, per tank, with an extraction of 60 per cent. of the
r content of the water.

Febles states that 37,000,000 gal. of water per month are
ed from the Leonard mine alone. Calling this figure 850 gal.
inute, 850 ÷ 8 = 106 electrolytic tanks would be required to
his water, with a recovery of but 60 per cent. of the copper. The
nption of power would be 444 × 106 = 4,700 kilowatts, or
h-p. consumed in the recovery of 60 per cent. of 68,000 lb. of
r per month.

high resistance of the water and the excessive voltage required
ctrolysis are due to the low free acid and copper sulphate con-
f the solution. Because of the large volume of the water to be
d, the addition of a sufficient quantity of sulphuric or other acid
ng the resistance within commercial limits is out of the question.
se the free sulphuric acid in the water to 1 per cent. would
e an expenditure of over $30,000 per month.

cost of power per ton of copper recovered could be reduced by
ng the current density and increasing the number of electro-
anks. The percentage of recovery of copper could be increased
creasing the speed of flow of the water and employing a larger
er of tanks; but as the excessive voltage required made the pro-
pear impossible from a commercial standpoint no further exper-
s were made along these lines.

*clusions.*—The recovery of the copper from the mine water by
electrolysis does not appear to be possible from a commercial
point.

ome method of economically improving the electrical conduc-
of the mine water were available the problems of electrolysis
be more attractive.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.
[SUBJECT TO REVISION.]

USSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the ting, Aug. 18 to 21, 1913, when an abstract of the paper will be read. If this is impossible, ussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 Street, New York, N. Y., for presentation by the Secretary or other representative of its Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1913, when . of the *Transactions* will go to press. Any discussion offered thereafter should preferably be n of a new paper for publication in Vol. XLVII. (with suitable cross-references in both

# The Laws of Jointing.

BY BLAMEY STEVENS, TEMASCALTEPEC, MEXICO.

(Butte Meeting, August, 1913.)

following paper aims to make a full explanation of the phe-
a of rock jointing.

ay be unnecessary to give any general description of what are
l joints in rocks, but Professor Geikie's clear and faithful gen-
tions are so illuminating that no excuse need be given for
ucing the following and other quotations into this paper:

ost all rocks are traversed by vertical or highly inclined divisional planes termed
These have been regarded as due in some way to contraction during consolidation
of retreat); and this is no doubt their origin in innumerable cases. But, on the
nd, their frequent regularity and persistence across materials of very varying
uggest rather the effects of internal pressure and movement within the crust."[1]

*eral Explanation.*—In this paper I show that the phenomenon
ting may to a large extent be accounted for by the pressure of
ter contained in the pores of the rock.

understand this, each constituent grain of the porous material
e considered to be separately compressed by the water which
rges it. As all the grains are held together by cohesion, the
mass is contracted by this pressure.

n indefinitely extended mass, however, there must be a limit
possible contraction without rupture, because the material is
ned from moving freely by other material around its bounda-
The mass of rock thus becomes broken up.

s phenomenon is known as "shrinkage" jointing. The term
kage" is an apt one, if it be understood that the shrinkage of
ck is often brought about by change of stress.

*sses.*—The compressive component of water stress on the indi-
grains is evidently the same as that of the water with which it
ontact. This is so, both before and after the jointing is formed.
the jointing is formed there is, however, a tensile stress in one
re directions which keeps the material from contracting.

[1] *Text Book of Geology*, 3d ed., p. 318 (1893).

At the moment of the formation of joints this stress is e
cohesive compressive stress. This is more commonly
engineers as the tensile breaking stress.

Table I. shows this tensile breaking stress for several roc
corresponding water head which will produce it.

TABLE I.—*Tensile Strength of Rocks and Equivalent De*

| Kind of Stone. | (1) Modulus of Rupture. | (2) Equivalent Water Head. | (3) Equivalent Rock in Water Head. |
|---|---|---|---|
| | Pounds per Sq. In. | Feet. | Feet. |
| Granites. | | | |
| Very hard, coarse grained St. Gotthard, oblique to bed, | 1,940 | 4,770 | 2,98 |
| Very hard, coarse grained St. Gotthard, parallel to bed, | 1,309 | 3,010 | 2,01 |
| Very hard, striped grained St. Gotthard, perpendicular to bed, | 2,773 | 6,380 | 4,25 |
| Very hard, medium fine grained, from Cham, | 1,991 | 4,580 | 3,06 |
| Limestones. | | | |
| Muschelkalk, from Würzburg, | 680 | 2,260 | 1,51 |
| Best quality, | 1,252 | 2,880 | 1,92 |
| Poorest quality, | 939 | 2,160 | 1,44 |
| Dolomites. | | | |
| From the white Frankenjura, Poppenheim, | 2,560 | 5,900 | 3,93 |

NOTE: Column 2 gives the minimum depth of formation of secondary vert
joints.
Column 3 gives the minimum depth of formation of primary vertical shrin

*Causes of Water Pressure.*—The most evident cause of
sure existing in rocks is its natural head, as explained in
paper.[3] This exists in what is known as the subterranean
tension-producing effect of the natural water head existin
of the subterranean sea is, however, counterbalanced by
stresses. The proof of this, which involves a little math
explained in the Appendix.

There are, however, other large bodies of interstitial w
subterranean lakes, where the water pressure is much g
the natural head would be at that depth, and the pressure
great as that due to the whole weight of the overlying r
2.5 times what the water head would be for the same deptl

An evidence of the existence of these great fluid stresse
the uncontrollable pressure of some new oil and gas wells.

Lanza, *Applied Mechanics*, 5th ed., p. 718 (1891).
The Laws of Igneous Emanation Pressure, *Trans.*, xliii., 167 (191

ere only because of the expansive forces of the gases, but there oom for doubt that similar pressures have existed at some time ly all buried stratified rocks. There are also other evidences of essure, which I intend to treat in another paper.

pressure is brought about by the natural settling of the rocks always occurs. The pore space is thus considerably reduced, e water, imprisoned by the strata of impervious material, finally a pressure nearly equal to the weight of the overlying rocks. ertain calculations given in the Appendix, and made on the otion that the elastic stress ratios of glass apply to rock, the water es which will rupture the various rocks are found, and these en in the second and third columns of Table I.

naking these calculations it is presumed that the whole rock s balanced by water pressure. This is not always quite the nd perhaps seldom so, but the figures so found give the *minimum* of formation of shrinkage joints.

mn 3 is only applicable to primary vertical shrinkage joints; to say, to joints in the direction which is first ruptured.

onsiderable increase of water pressure is necessary to cause g at right angles to this, as is shown in the Appendix and the column of Table I.

r *Joints.*—After a certain depth is reached and the excess of pressure released, by the formation of joints, etc., the simple l compression of the rock, partly buoyed up by the water, s.

pressure (which is approximately represented in water head H) becomes at a certain depth great enough to crush rocks pen shrinkage joints. The immediate cause of crushing frac-, of course, shearing.

corresponding depth of submersion in the subterranean sea re- to produce this stress will be found in the third column of II. This table is for the same kind of rocks as Table I.

to be noted that the depth required for shear jointing is very greater than for primary and secondary vertical joints. This ns their absence in the less altered sedimentary rocks.

angle[4] between the shear and contraction jointing and the n is more or less the same as it would be with a fault. It is ined, however, to some extent, by the size of the block in it occurs, its relation to other blocks, which have or have not imilarly fractured, and by other local factors.

[4] See Laws of Fissures, *Trans.*, xl., 475 (1909).

TABLE II.—*Compressive Strength of Rocks and Equivalen*

| Kind of Stone. | (1) Crushing Strength. | (2) Equivalent Water Head. | (3) Equivalent Rock in Water |
|---|---|---|---|
| | Pounds per Sq. In. | Feet. | Fee |
| Granites. | | | |
| Very hard, coarse grained St. Gotthard, perpendicular to bed, | 11,740 | 27,000 | 18,0 |
| Very hard, coarse grained St. Gotthard, parallel to bed, | 12,660 | 29,100 | 19,4 |
| Very hard, striped grained St. Gotthard, perpendicular to bed, | 15,650 | 36,000 | 24,0 |
| Very hard, medium fine grained, from Cham, | 22,190 | 51,000 | 34,0 |
| Limestones. | | | |
| Muschelkalk, from Würzburg, | 6,260 | 14,400 | 9,6 |
| Best quality, | 6,900 to 8,390 | 15,900 to 19,300 | 10,6 to 12,9 |
| Poorest quality, | 3,560 | 8,200 | 5,4 |
| Dolomites. | | | |
| From the white Frankenjura, Poppenheim, | 16,780 to 18,490 | 38,600 to 42,500 | 25,7 to 28,3 |
| Sandstones. | | | |
| Very variable, | 2,500 to 20,000 | 5,700 to 46,000 | 3,8 to 30,6 |

NOTE: Column 3 gives the minimum depth of formation of sheared jointi

*Joints in Stratified Rocks.*—Geikie says of these:

"To the presence of joints some of the most familiar features of rock-s Joints vary in the angles at which they cut the planes of bedding, in the sh definition, in the regularity of their perpendicular and horizontal course, persistence, in number, and in the directions of their intersection. As a most sharply defined in proportion to the fineness of grain of the rock. In close-grained shales, for example, they often occur so clean-cut as to be inv vealed by fracture or by the slow disintegrating effects of the weather. The along these concealed lines of division, whether the agent of demolition be frost. In coarse-textured rocks, on the other hand, joints are more apt selves as irregular sinuous rents.

"As a rule, they run perpendicular, or approximately so, to the planes o descend vertically at not very unequal distances, so that the portions o them, when seen in profile, appear marked off into so many wall-like masses. metry often gives place to a more or less tortuous course with lateral jo random directions, more especially where the different strata vary consid logical characters. A single joint may be traced for many yards, sometime many miles, more particularly when the rock is fine-grained, as in limesto the texture is coarse and unequal, the joints, though abundant, run into eac a way that no one in particular can be identified for more than a limited number of joints in a mass of stratified rock varies within wide limits. which have undergone little disturbance the joints may be separated from intervals of several yards. But in other cases where terrestrial movemen

, the rocks are so jointed as to have acquired therefrom a fissile character that
y or wholly obliterated their tendency to split along the lines of bedding.
nportant feature in the joints of stratified rocks is the direction in which they
each other. In general they have two dominant trends, one coincident, on the
ith the direction in which the strata are inclined from the horizon, and the other
transversely at a right angle or nearly so. The former set is known as *dip-joints*,
hey run with the *dip* or inclination of the rocks; the latter is termed *strike-joints*,
as they conform to the *strike* or general outcrop. It is owing to the existence
uble series of joints that ordinary quarrying operations can be carried on. Large
ular blocks can be wedged off, which would be shattered if exposed to the risks
g."

* * * * * * * * *

nary household [soft] coal presents a remarkably well-developed system of joints.
of such coal may be observed to be traversed by fine laminæ, the surfaces of many
are soft and soil the fingers. These are the planes of stratification. Perpendicu-
em run divisional planes, which cut each other at right angles or nearly so, and
de the mineral into cubical fragments. One of these sets of joints makes clean
lefined surfaces, and is known as the *face*, *slyne*, *cleat*, or *bord*; the other has rougher,
lar surfaces, and is known as the *end*. The face remains persistent over wide
serves to define the direction of the roadways in coal-mines, which must run
[5]

hout having to repeat the reasoning given in former papers,[6]
be clear that one set of parallel, vertical joints is first formed
rection perpendicular to the least rock stress, then when the
pressure increases enough the other set is formed at right
to the first set.

"face" joints in coal, described by Geikie, evidently belong
first-formed set and the "end" joints to the second set. The
e" joints and "dip" joints have a relation to the directions of
ock pressure, for it is to be noted that a sedimentary rock is
made to dip by gentle folding; and folding is caused by a
essive rock stress.

nany folded rocks, the main, or "master," joints are observed
perpendicular to the axes of the folds.

ere there has been no particular direction of least stress it is
t that "more or less tortuous courses with lateral joints in
s random directions" may be formed.

en the water pressure exceeds that due to the overlying rock,
so evident that horizontal or inclined shrinkage joints may be

*ts.*—The formation of joints in rock under pressure no doubt
leads to the draining out of part of the water which has been
ned in the rock.

s water makes its exit from the rock mass under the pressure

[5] *Text Book of Geology*, 3d ed., p. 524 (1893).
[6] *Ibid.*, and The Laws of Intrusion, *Trans.*, xli., 650 (1910).

DH + C, which is the same as for emanation and intru
in equipressure rocks.[7]

Water at this excessive pressure is therefore capable of
passage for itself in the rocks above it and of acting in oth
emanations.

*Shearing Joints.*—When a brittle rock alternates with
(as an alternation of sandstone and shale) the brittle one
tured, while the plastic one escapes. The master joints,
the brittle rock, rapidly become obliterated after they reac

A critical examination of the shale often shows tha
properties are due to the ease with which it shears.

*Joints in Massive Rocks.*—Geikie says of these:

"While in stratified rocks the divisional planes consist of lines of beddi
cutting each other usually at a high, if not a right, angle, in massive (ign
include joints only; and as these do not, as a rule, present the same par
of bedding, unstratified rocks, even though as full of joints, have not
arrangement of stratified formations. Some massive rocks indeed may
of divisional planes so largely developed as to acquire a bedded or fissile
structure, characteristically shown by phonolites, may also be detected
porphyries. Most massive rocks are traversed by two intersecting sets of
joints, whereby the rock is divided into long quadrangular, rhomboidal, o
columns. A third set may usually be noticed cutting across the columns
them into segments, though generally less continuous and dominant than t
these last-named cross-joints are absent or feebly developed, columns mar
can be quarried out entire. Such monoliths have been from early times
construction of obelisks and pillars.

"In large masses of granite, an outward inclination of the natural div
the rock may sometimes be observed, as if the granite were really a rud
having a dip towards and under the strata which rest upon its flanks. It
arrangement of the constituent minerals analogous to the foliation of gne
traced in perfectly amorphous and thoroughly crystalline granite, but
form of jointing by reason of which the rock weathers into large blocks
another like a kind of rude cyclopean masonry. In the quarrying of gran
recognise that the rock splits into blocks much more easily in one directio
nally there is no trace of any structure which could give rise to this tende

Igneous rocks, especially the more basic ones, contrac
lization. In dikes and lava flows the consolidation proce
sides towards the center and a columnar structure is so
pendicular to the walls.

The more massive igneous rocks are usually cooled at
and consolidation takes place much more slowly. Und
ditions the solidifying rock adjusts itself to the contr
place, and the jointing from this cause is either absent,
not at all prominently displayed.

[7] Laws of Igneous Emanation Pressure, *Trans.*, xliii., 167 (19
[8] *Text Book of Geology*, 3d ed., p. 527 (1893).

*ts in Schistose Rocks.*—Geikie says of these:

schists likewise possess their joints, which approximate in character to those ie massive igneous rocks, but they are on the whole less distinct and continuous, eir effect in dividing the rock into oblong masses is considerably modified by the e lines of foliation. These lines play somewhat the same part as those of stratifi- mong the stratified rocks, though with less definiteness and precision. The joint- ie more massive foliated rocks, such as the coarser varieties of gneiss, approaches ely to that of granite; in the finely fissile schists, on the other hand, it is rather ith that of sedimentary formations. Upon these differences much of the charac- ariety of outline presented by cliffs and crests of foliated rocks depends."[9]

erally, it might be said that metamorphic rocks have been at greater depths than the less altered sedimentaries. They , therefore, be more jointed.

there are several modifying factors, Having been once well and perhaps faulted, some rocks can no longer retain any con- ole excess of water pressure. It is rare to find any but fresh n highly metamorphic rocks, while sedimentaries usually con- lt water in depth.

most pronounced joints are usually those which are vertical and dicular to the foliation.[10] These are the same as the primary of the sedimentary rocks.

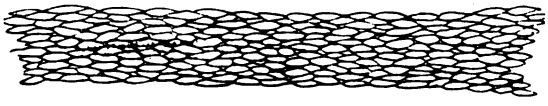

FIG. 1.—CLEAVAGE STRUCTURE OF SLATE.

istose and slaty rocks necessarily result from great horizontal re and plastic yielding. The individual grains of the material ereby flattened as in Fig. 1. It is evident that a large portion of undaries of these grains are parallel to one directional plane. s called the plain of cleavage or of fissility and is naturally a on of small cohesion. Along it the material will easily split. ndicular to it the rock has considerable strength, the flattened being bonded together in such a manner as to offer much area tional resistance to one another. When any attempt is made to te them by a pull in this perpendicular direction, a great area cture is presented.

ny rocks other than slate show a slaty cleavage which may easily taken for jointing. Its general strike, however, is at right an- o the great horizontal pressure to which the rocks of the region

*Book of Geology*, 3d. ed., p. 531 (1893).

e arkose and slate series of Latouche Island, Alaska, is a good example which hap- have come to my notice.

have been subjected. There is often good evidence as to
plane of cleavage is not, properly speaking, a plane of jo
jointing planes are formed at various angles to it.

*Irregular Jointing.*—Many metamorphic rocks show muc
jointing. This may be explained by variation in structure
portant cause is the occurrence of internal (uneven) str
have their effect after the great earth pressures are relea
joints are usually curved or radial.

Geikie says of these:

"Among mountain-valleys, in railway-tunnels through hilly regions, or el
rocks subjected to much lateral pressure, or where owing to the removal o
running water, and the consequent formation of cavities, subsidence is in pr
as of explosions are occasionally heard. In many instances, these noises ar
relief from great lateral compression, the rocks having for ages been in a
from which as denudation advances, or as artificial excavations are made, the
This relief takes place, not always uniformly, but sometimes cumulatively
shocks or snaps. Mr. W. H. Niles of Boston has described a number of in
where the effects of such expansion could be seen in quarries; large block
rent and crushed into fragments and smaller pieces being even discharged
into the air. More recently Mr. A. Strahan has called attention to the occu
ensided surfaces in the lead-mines of Derbyshire which on being struck or
with a miner's pick break off with explosive violence, and he suggests tha
ores along those surfaces are in 'a state of molecular strain resembling that
Drop or of toughened glass, and that this condition of strain is the resu
movements which produced the slickensides.'

"If such is the state of strain in which some rocks exist even at the surfac
distance beneath it, we can realize that at great depths, where escape fro
long periods impossible, and the compression of the masses must be enormo
relief from this strain may well give rise to an earthquake shock. A conti
of strain must also influence the solvent power of water permeating the roc

Following the reasoning used for fissures[12] and intru
straightness or crookedness of a joint is no doubt largely
the value of stress components, other than that to which i
imately perpendicular.

An irregular rock structure may also be the cause of irre
ing. Such rocks as tuffs, conglomerates, etc., may be ver
jointed. Thin beds are usually less evenly jointed than

It is seldom, however, that there is not one obvious pla
which many of the principal joints approximate. Some
may be a linear direction which is common to planes whic
wise apparently irregularly placed. Such, for example, a
cal, or the perpendicular to the walls of a dike or vein.

*Size and Number of Joints.*—The distance between joints

[11] *Text Book of Geology*, 3d ed., p. 311 (1893).
[12] Ibid.

ount of strain which the surrounding rocks will allow of. A thick rock will have a much larger spacing of the joints than occurs in bed. Thus thick beds of limestone often have joints very spaced, while thin beds of the same material have joints very ogether.

*ct of Metasomatism.*—Sometimes water and solutes contained in absorbed by minerals, with consequent swelling, Serpentini- is an example of such a case. The effect is just the opposite of jointing: viz., to make a compact joint-free rock.

effect of mineral alteration is, however, often a shrinkage of e. Thus the alteration of limestone to dolomite. The effect in ases would always be to form joints, were it not that the highly d waters deposit material in the cracks and so re-weld them.

*mnar Jointing.*—Geikie says of this:

most characteristic structure, however, among volcanic rocks is the prismatic, or' ncorrectly termed, 'basaltic.' Where this arrangement occurs, as it so commonly asalt, the mass is divided into tolerably regular pentagonal, hexagonal, or irregu- ygonal prisms or columns, set close together at a right angle to the main cooling These prisms vary from 1 inch or even less to 18 or more inches in diam- l range up to 100 or even 150 feet in height. Many excellent and well- examples of columnar structure are exhibited on the coast-cliffs of the volcanic region of Antrim and the west coast of Scotland, as in the Giant's y and Fingal's Cave. In many cases no sharp line can be drawn between a col- asalt and the beds above and below, which show no similar structure, but into e prismatic mass seems to pass." [13]

agonal jointing is formed by the operation of laws of an entirely nt kind from those determining other kinds of joints. I have l that, in drying, silt starts with the opening of almost innu- e extremely small cracks, as shown in Fig. 2. These apparently n haphazard directions, but each one is very straight, is widest middle and closes gradually towards the ends. It is in fact ible to say exactly where a cracklet begins or ends.

he surface of the mud continues to dry these cracklets extend oaden themselves until one runs into another as in Fig. 3. The f junction then becomes considerably enlarged and the three ons of radiation are extended rapidly. But other cracklets for derable radius around this center cease to develop.

radial arms do not now follow straight courses, but as they cut racklets, tend to take their course, as shown in Fig. 4.

s to be noticed, however, that the arms tend to take mean s in such a way that angles of 120° are extended between them. ne meantime other similar centers of fracture have been form-

[13] *Text Book of Geology*, 3d ed., p. 529 (1893).

ing in just the same way and these are all more or les distance apart.

Finally, the arms of each of these centers intersect with adjoining center and a very rough irregular hexagonal pat now much-enlarged cracks is formed.

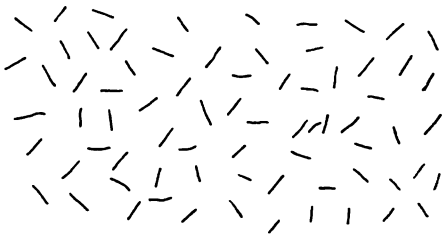

FIG. 2.—FIRST STAGE IN CRACKING OF MUD.

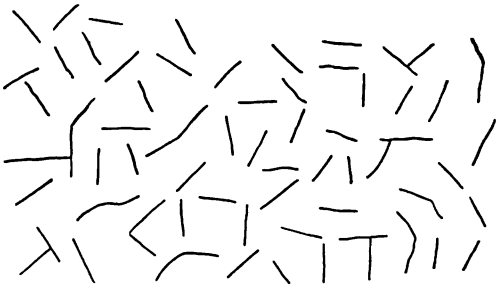

FIG. 3.—SECOND STAGE IN CRACKING OF MUD.

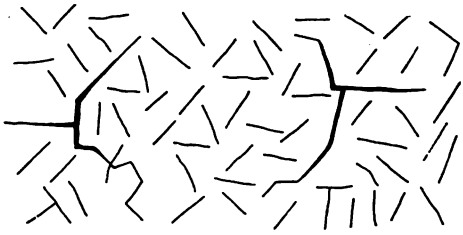

FIG. 4.—THIRD STAGE IN CRACKING OF MUD.

It was evident from the silt cliffs[14] of the vicinity that t become more straight and the polygons of more even size tion as the silt dries down. The jointing so formed is pe tical, and consequently the faces stand out like masonry 50 ft. or more in height, though the material can be cu finger nails, or crushed into powder in the hand. Crysta salt were found in the interstices, indicating clearly tha

[14] In vicinity of Okanagon lake, B. C.

ave dried by evaporation, and leaving no doubt as to the cause ting.

jointing of cooling lavas may, no doubt, be explained in the vay, although we cannot witness it so clearly. It is not neces- consider any center of cooling, as is usually done. General s of fracture are formed automatically by the evolution of crack- The larger of these survive and develop because they relieve ess to such an extent as to prevent the further development of aller cracklets.

*rical Jointing.*—Geikie says of this:

any prismatic massive rocks (basalt, diorite, &c.), segments of the prisms weather eroids, in which successive weathered rings forms crusts like the concentric coats ion." [15]

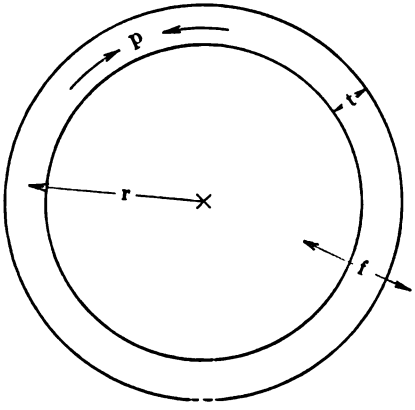


FIG. 5.—FORMATION OF SPHERICAL JOINTS.

erical jointing usually forms near the surface. The ground has usly been cut up into blocks by other forms of jointing. By ency of the atmosphere the rock becomes altered on each side joints and in doing so tends to expand. The corners are thus ed off so as to form new joints bordering a roughly spherical of rock. This in turn expands and causes a shell compression e surface of the sphere. Radial tensions f are thus developed. ratio of this radial tension to the shell compression is evidently $\frac{2t}{r}$, where t is the thickness of the shell and r the radius of the e. (See Fig. 5.)

alteration proceeds t increases until f becomes equal to the ten- reaking stress. The shell then becomes separated by a spherical

[15] *Text Book of Geology*, 3d ed., p. 348 (1893).

joint from the smaller remaining sphere inside. This process until alteration is complete.

*Some Misconceptions.*—An experiment of Daubrée in the f of cracks, by the twisting of a glass rod, is often used to torsion jointing. As a matter of fact, the cracks so formed traction joints, being formed at right angles to the direction tension.

It is to be noted that shear jointing proper is not forme planes of greatest tangential stress, but in the planes of grea of tangential to direct stress.

The assumption of the former hypothesis[16] leads to the c that shear joints are at right angles to one another, which is ally the case. The ordinary shrinkage joints may make rig with one another, but there is no reasonable way of consider as shear joints.

*Needed Experimental Work.*—Probably the most necessar mental work connected with the Laws of Joints is the deter of the coefficients of elasticity of rocks.

To do this it would only be necessary to take long drill co various rocks and note the deflections when used as beams a put under a torque. All the elastic coefficients and stre could be calculated from these easily obtained observations.

## Appendix.

The elastic properties of rocks appear to have been little In engineering books only special phases are treated of, and not those required in geological problems.

The general elastic properties of any homogeneous isotr stance are determined by two constants. These are most scie taken as the coefficient of volumetric elasticity, $\kappa$, and the c of rigidity, $\mu$. The values of $\kappa$ and $\mu$ have been determined few substances. Of these, glass is the only one which we sider as approaching the average rock. The elastic values are:[17]

$$\mu = 2{,}900{,}000 \text{ lb. per square inch.}$$
$$\kappa = 5{,}800{,}000 \text{ lb. per square inch.}$$

If a fluid pressure f is applied to a unit cube, Fig. 6, it w minished in bulk by $\frac{f}{\kappa}$.

[16] G. F. Becker, *Bulletin of the Geological Society of America*, vol. iv., p. Also, Torsional Theory of Joints, *Trans.*, xxiv., 130 (1894).

[17] *Encyclopedia Americana*, article Elasticity.

corresponding diminution of each unit dimension of the
$\frac{f}{3\kappa}$.

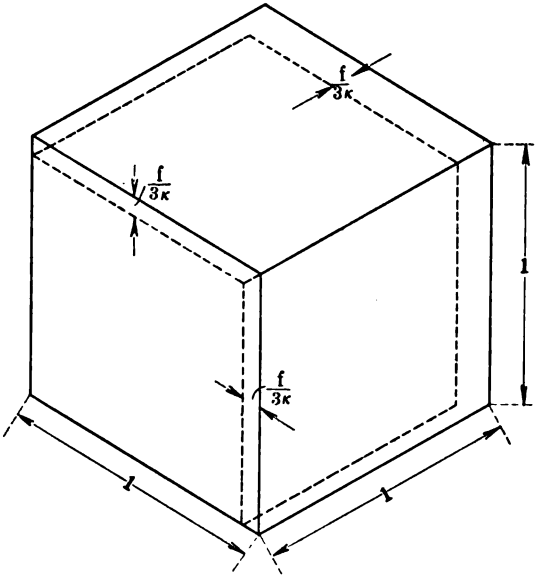


FIG. 6.—STRAIN PRODUCED BY FLUID COMPRESSIVE STRESS.

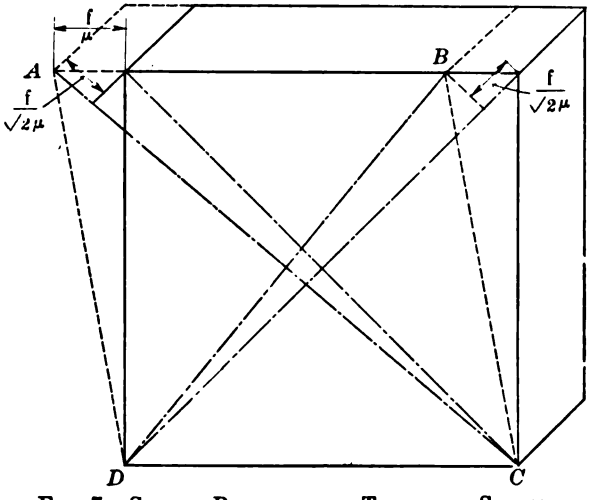


FIG. 7.—STRAIN PRODUCED BY TORSIONAL STRESS.

value of $3\kappa$, the linear component of volumetric elasticity, is,
s, 17,400,000 lb. per square inch.

shear is applied to a unit cube, Fig. 7, it will be distorted as
by $\frac{f}{\mu}$. This is equivalent to compressing it along the diago-

nal $BD$ by an amount $\frac{f}{\sqrt{2\mu}}$ and expanding it by an eq
along the diagonal $AC$. The proportional compression
sion along the direction of these diagonals are respectiv
by dividing the deflection by the length of the diagonals.
lengths are approximately $\sqrt{2}$ in each case, the proportion
sions and expansions are $\frac{f}{\sqrt{2\mu}} \times \frac{1}{\sqrt{2}} = \frac{f}{2\mu}$.

From an inspection of the stress relations in Fig. 8 it
seen that a shearing stress f on the unit diagonal is equiv
compressive forces $\frac{f}{\sqrt{2}}$ and two expansive forces $\frac{f}{\sqrt{2}}$ on

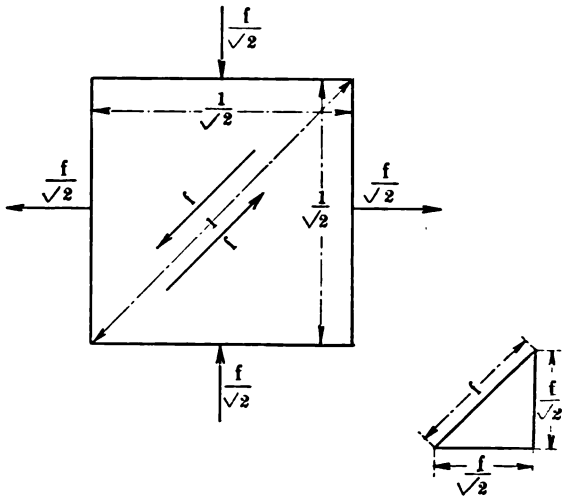


FIG. 8.—RELATION OF TORSIONAL TO DIRECT STRESSES.

the cube. As the sides of the cube on which these fo
$\frac{1}{\sqrt{2}}$ the corresponding compressive and expanding stres
each case.

The value of $2\mu$, the linear component of rigidity, is
5,800,000 lb. per square inch.

The "modulus of elasticity," used in engineering pro
combination of $\kappa$ and $\mu$. In finding it a direct stress is app
direction only.

In the diagram Fig. 9 presume a fluid compressive forc
applied equally in all directions on the unit cube shown.

of each single dimension is $\frac{f}{3\kappa}$. Now apply shears such that
re no resultant forces on the cube in the $y$ and $z$ directions.
ll be equivalent to expanding the cube by $\frac{f}{2\mu}$ in each of these
ns and compressing it in the $x$ direction by equal amounts.
ess in the $x$ direction is then 3f and the strain $\frac{f}{3\kappa} + \frac{f}{\mu}$. The
these is the "modulus of elasticity,"

$$E = \frac{3f}{\frac{f}{3\kappa} + \frac{f}{\mu}} = \frac{9\kappa\mu}{3\kappa + \mu} \quad . \quad . \quad . \quad (1)$$

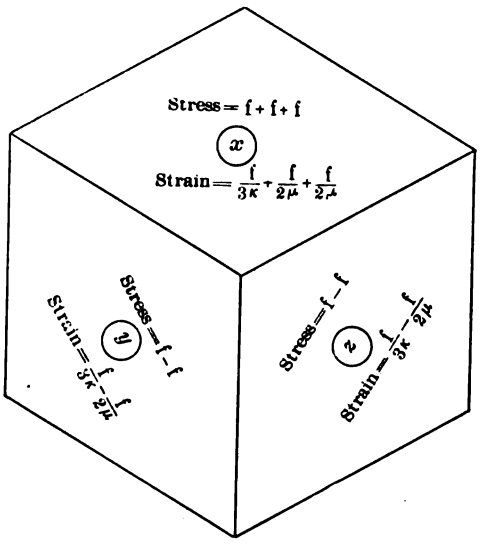


FIG. 9.—STRAINS PRODUCED BY A DIRECT STRESS.

; the coefficients of $\mu$ and $\kappa$, given for glass, the value of E is,
mately, E = 7,450,000.
strain in each direction at right angles to the direction of
$\frac{f}{3\kappa} - \frac{f}{2\mu}$. By dividing the $x$ stress, 3f, by this strain we get
ficient

$$B = \frac{-3f}{\frac{f}{2\mu} - \frac{f}{3\kappa}} = \frac{-18\kappa\mu}{3\kappa - 2\mu}, \quad . \quad . \quad . \quad (2)$$

For glass, therefore, $B = -27{,}000{,}000$. The ratio $\frac{E}{-B}$ is c
son's ratio and is given by

$$\frac{E}{-B} = \frac{3\kappa - 2\mu}{6\kappa + 2\mu}, \quad . \quad . \quad .$$

For glass, therefore, $\frac{E}{-B} = 0.286$.

Another case is when there is only a single strain, say t
direction $x$, Fig. 10. Presume again that a compressive
applied and creates a strain $\frac{f}{3\kappa}$ on each face of the unit c
strain can be counteracted by one component of a shearin

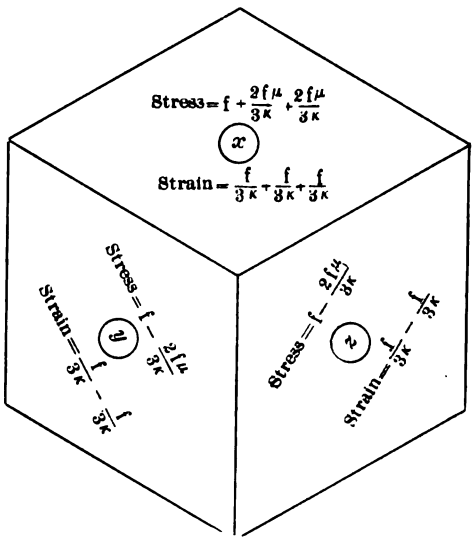


FIG. 10.—STRESSES PRODUCED BY A DIRECT STRAIN.

the same amount and opposite sign. The shearing stress r
this is obtained by multiplying the strain $\frac{f}{3\kappa}$ by $2\mu$: viz.,
soning as before, the coefficient of elasticity sought is

$$A = \frac{f + 4f\mu}{\frac{f}{3\kappa} + \frac{2f}{3\kappa}} = \kappa + \frac{4}{3}\mu, \quad . \quad .$$

For glass, therefore, $A = 9{,}700{,}000$.

stress at right angles to the direction of strain is $f - \frac{2f\mu}{3\kappa}$.
io of this stress to the stress in the direction of the strain is
e:

$$i = \frac{1 - \frac{2\mu}{3\kappa}}{1 + \frac{4\mu}{3\kappa}} = \frac{3\kappa - 2\mu}{3\kappa + 4\mu}, \quad . \quad . \quad . \quad (5)$$

lass, therefore, $i = 0.4$.
her case of geological strain is that of an applied stress in one
n, no strain in a second direction and no increment of force in
d direction.

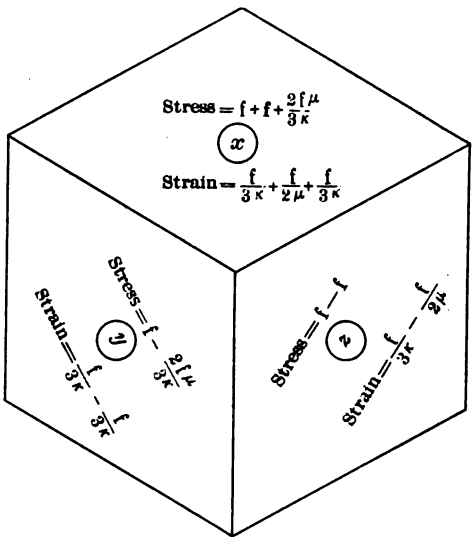


FIG. 11.—A COMMON GEOLOGICAL STRAIN SYSTEM.

11 shows how such a system may be made up of a unit bulk
shear from $y$ to $x$ and another from $z$ to $x$. The resultant
nt in the direction $x$ will be

$$G = \frac{2 + \frac{2\mu}{3\kappa}}{\frac{2}{3\kappa} + \frac{1}{2\mu}}, \quad . \quad . \quad . \quad . \quad . \quad . \quad (6)$$

lass, therefore, $G = 8{,}100{,}000$.

The ratio of collateral to lateral stress is

$$j = \frac{3\kappa - 2\mu}{6\kappa + 2\mu}, \quad . \quad . \quad . \quad .$$

This is the same in value as Poisson's ratio $\frac{E}{-B}$ (equati

. . .

For glass, therefore, $j = 0.286$.

The effect of a vertical rock stress may now be calcu
be the depth, the vertical rock pressure in water head wil
mately, $H \times 2.5$. If this head produces no horizonta
horizontal stresses necessary to balance it will be i
(= H for glass). So that this horizontal stress would
by the water pressure, and no tension produced. This
somewhat modified if the elastic ratios of rocks differ
glass. It is also to be noted that the water pressure act
rock material, while the rock pressure acts on the poro
of grains, whose elastic coefficient must be somewhat les

The extreme result of water pressure is attained wh
pressure becomes equal to the whole rock pressure. It
well become greater than this, as horizontal joints woul
along those planes where there is little cohesion and
open indefinitely.

When the pressure is equal to the rock head, the vert
the rock is $H \times 2.5$ and the resultant horizontal comp
is $i \times H \times 2.5 = H$ (for glass).

The tension stress produced horizontally by the wate
$H \times 2.5$, so that the net horizontal tension of the rock b
is the balance of 1.5 H (for glass.)

When the water pressure is equal to the rock head
tends to contract the rock and the depth producing jo
these conditions may be directly read from the third colu
I. As the fluid pressure is no doubt usually less than th
whole weight of rock, the column must simply be taken
the least possible depth of joint formation in stratified ro

The first set of joints formed may be presumed to be p
to the $z$ axis. (See Fig. 12.) The stress in each of th
tions perpendicular to $z$, reckoned as water head, is 2.5
these creates a stress along $z$ of $2.5\ H\ j = 0.715\ H$ (for
resultant compressive stress along $z$ due to these two is t
H (for glass). The tensile stress produced along $z$ by th
sure is 2.5 H, so that the net horizontal tension along $z$ is
$H = H$ (approx. for glass).

minimum depth necessary to produce jointing under these ons may be directly read from the second column of Table I. imperfect or partly formed nature of many secondary jointing s is thus explained by the fact that considerably more depth is d for their development than for the primary joints.

W. O. Crosby has speculated on the influence of earthquakes lucing joints.[18]

hquakes of the largest known violence have an amplitude of on (a) of about 2 in. and a complete period (t) of 0.5 sec. The

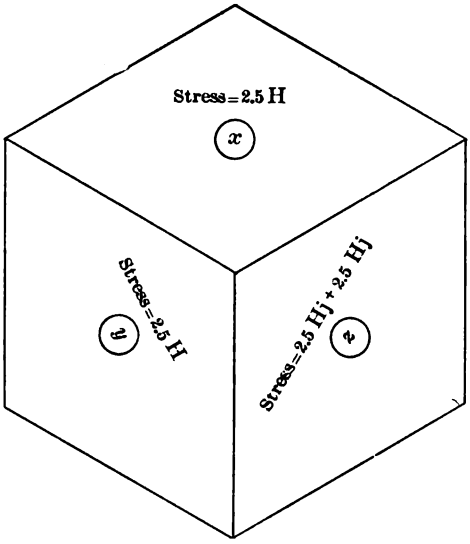


FIG. 12.—STRESSES PRODUCING SECONDARY JOINTS (MAXIMUM).

y (v) is about 4,000 ft. per second. The following formula he resultant stress: $h = \frac{\pi a v}{g t}$, where h is the maximum pres- roduced in rock head. Space is not taken here to show how rmula is obtained because it is easily derived from those given ks on earthquakes.

greatest difference of stress from the normal, produced by an uake, is thus seen to be equivalent to 130 ft. of rock head. s a small amount relatively to the joint-forming stresses and ore earthquakes cannot be considered as a factor in jointing.

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*Proceedings of the Boston Society of Natural History*, vol. xxii., p. 72 (1882–83).

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.
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CUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the eeting, Aug. 18 to 21, 1913, when an abstract of the paper will be read. If this is impossible, cussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 h Street, New York, N. Y., for presentation by the Secretary or other representative of its Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1913, when VI. of the *Transactions* will go to press. Any discussion offered thereafter should preferably form of a new paper for publication in Vol. XLVII. (with suitable cross references in both ).

# The Metaline Plant of the Inland Portland Cement Co., Metaline Falls, Wash.

BY MILO W. KREJCI, GREAT FALLS, MONT.

(Butte Meeting, August, 1913.)

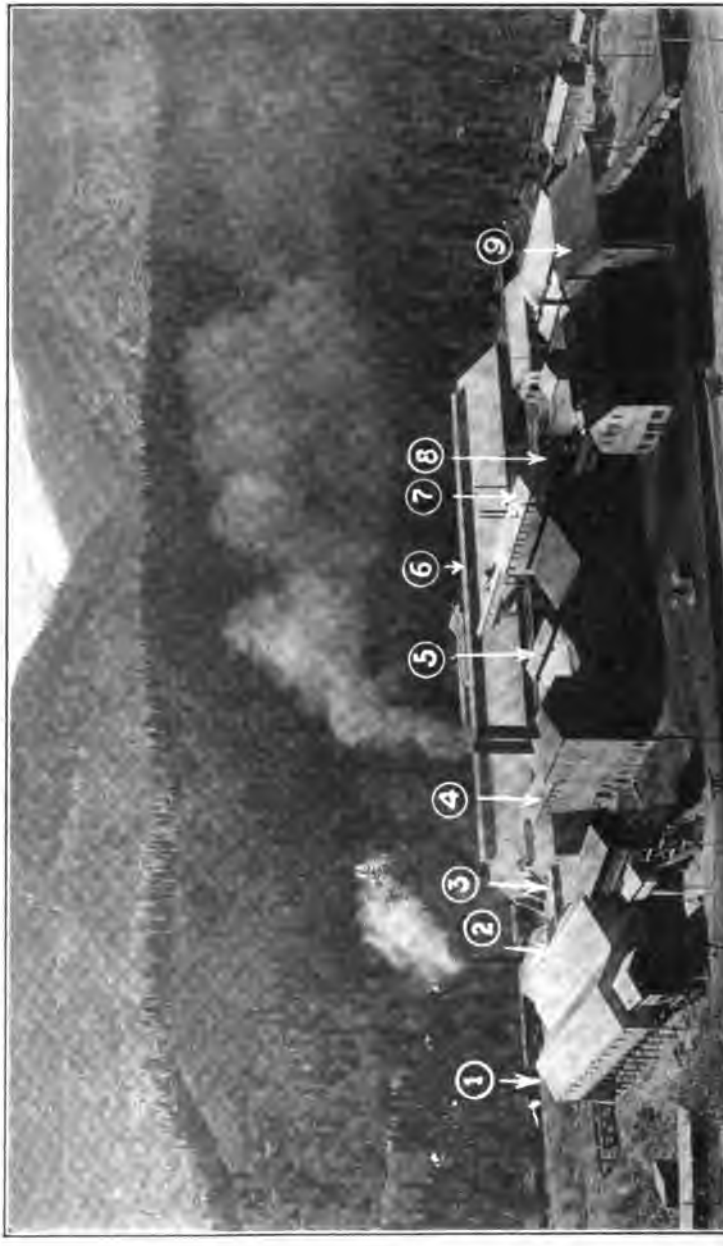
E plant and quarries of the Inland Portland Cement Co. are d at Metaline Falls, Wash., about 128 miles north of Spokane, e Pend Oreille river, and within 10 miles of the Canadian r. The plant is one of a chain of cement plants controlled by ehigh Portland Cement Co., of Allentown, Pa. Work on this was commenced in June, 1910, and was completed in April, but on account of some delays in the construction of the power the works were not put into operation until August, 1911. then it has been in continuous operation and has furnished the d Empire with a high grade of cement. Fig. 1 is a view and 2 is a plan of the works.

*ter Power Development.*—The source of the water supply for the -electric plant of the Inland Portland Cement Co. is Sullivan lake, y of water approximately 5.5 miles long and 0.5 mile wide, situ- n the Kaniksu Forest Reserve. A wooden crib dam was installed to raise the elevation of the lake 25 ft., and the additional stor- apacity was taken up by the diversion of Sullivan creek into the by means of a wooden flume, 0.25 mile long, 6 by 8 ft. in cross- n, and a ditch 1,200 ft. long.

e water from the lake follows the natural water course to an nediate storage reservoir, which was created by the construction earthen dam. This pond was used in connection with the saw which was installed at this point.

sides the earth dam, there was also built a crib, at the point n in the sketch map, Fig. 3. This crib dam is 35 ft. high. From point the excess water follows the natural course of Sullivan creek e Pend Oreille river.

e water is taken from this intermediage storage by means of a 8 ft. wooden flume approximately 13,000 ft. long. This flume ilt up on the side hill, piling being driven most of the way on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ınt of the lack of rock foundation. From the end of the flume is a ditch 850 ft. long, which leads into the head gates. The r at this point has a vertical drop of 41 ft. and an angle of 45° tunnel through the hill, down to the elevation of the wheels. tunnel is lined with 36-in. steel pipe throughout its entire length. total head is 465 ft. and the working pressure at the wheels is lb. per square inch.

*ower House.*—In the power house are installed two 1,865 kv-a. tinghouse generators, connected directly to Pelton water wheels e double-bucket type, two nozzles to each wheel, one 3.75 in. in

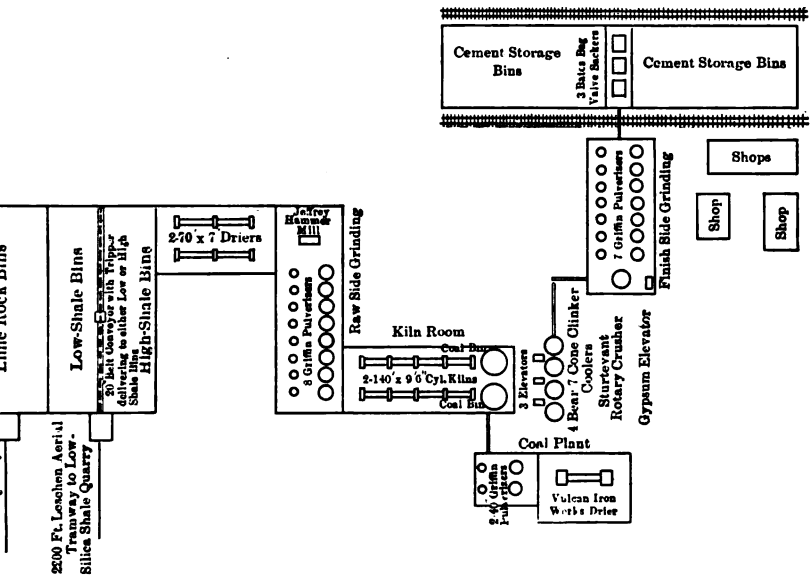



FIG. 2.—PLAN OF PLANT OF INLAND PORTLAND CEMENT CO.

neter and the other a needle valve nozzle. The generators are en at a speed of 360 rev. per min. and generate 260 volts. In ition to the two generators, there are two 125-kw. exciters.

he current for quarry purposes is transmitted at 2,300 volts, 500 being transformed to 550 volts at the quarry. The transmission n the power house to the cement plant is at 2,300 volts and is sformed to 550 volts with two water-cooled transformers of 0 kw. capacity. There is also an oil transformer for the light-circuits.

*Quarries.*—The company has three quarries of raw materials from ch the cement is made:

1. A limestone quarry, about 3,700 ft. east of the plant
2. A low-silica shale quarry, about 2,200 ft. east of the
3. A high-silica shale quarry, about 4 miles south of th

The raw materials are quarried by means of drilling wi of the Ingersoll-Rand and Jeffrey types and blasted down mite.

1. The limestone quarry is a very extensive one and is the open-quarry fashion, the rock being loaded into ho

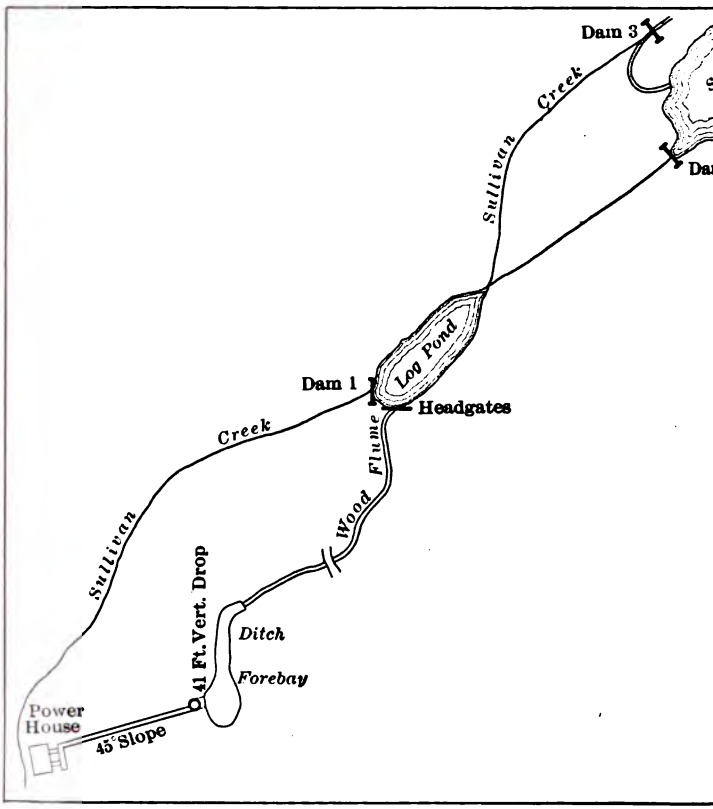



FIG. 3.—SKETCH-MAP SHOWING WATER POWER DEVELOPMENT INLAND PORTLAND CEMENT CO.

cars by means of a Marion-50 steam shovel and hauled t ing plant by horses.

The crushing plant is but a short distance from the is equipped with a No. 9 Gates gyratory crusher, whose delivered to two No. 5 Gates gyratory crushers, where it b to about 1.25-in. size. The crushers are all independently have a capacity of 200 tons per hour.

e broken lime rock is delivered to the loading bins, whence it ansferred to the cement plant by means of a Leschen aerial way, 3,700 ft. long. The tramway is equipped with 27 buckets on ne, each bucket having a capacity of 1,200 lb. These buckets are natically dumped into the lime rock storage at the cement plant. tramway requires no power to operate it, being operated by ty, and in fact it develops 16 h-p. It is controlled with a triple e.

The low-silica (calcareous) shale quarry is located about 2,200 st of the cement works and about 300 ft. from the power house. shale contains from 30 to 55 per cent. of silica and comes from a about 400 ft. below the limestone.

pen-cut quarrying is employed, the rock being loaded by hand tram cars and taken to a loading bin, from which it is delivered buckets on a Leschen aerial tramway, which conveys it to a No. tes gyratory crusher at the shale storage trestle.

e shale tramway is driven by a 30-h-p. motor. No particular ng for the buckets is required. About 12 h-p. is actually ired to operate this tram.

e product of the crusher falls on a conveyor belt fitted with a er, which dumps it on the low-silica shale storage pile.

The high-silica shale is obtained from an open cut along the o & Washington railroad, about 4 miles south of the cement s, and is loaded into hopper-bottom cars by means of a Marion-team shovel.

e shale is broken down in a No. 5 Gates crusher, and is deliv- to the storage pile by means of a 20-in. belt conveyor. It con- from 55 to 75 per cent. of silica.

*rying Limestone and Shale.*—The high-silica and low-silica shales now drawn from chutes, in tunnels underneath the storage piles, o 20-in. conveyors and delivered, in the proper volumetric pro- ions required in the mix, to an elevator, which feeds into a bin eding the driers.

a like manner, the limestone is fed on to a 20-in. conveyor belt elevated to a bin preceding the drier used for drying the crushed stone.

*he Driers.*—There are two cylindrical driers, one on limestone and on shale, each being 70 ft. long and 7 ft. in diameter, set on an nation of $\frac{3}{8}$ in. to the foot; Z bars are riveted longitudinally on inside, which turn the material into the path of the hot gases. y were built by the Vulcan Iron Works.

he dried materials are now elevated to separate bins and fed

therefrom to an automatic weighing machine, in the proper
required in the "mix." The automatic weighing machi
nished by the Automatic Weighing Machine Co., of Newa

*Pulverizing the Mix.*—From the weighing machine the
conveyed on a 20-in. belt conveyor to a type B Jeffrey ha
which reduces it to about 0.25 in. size. The mill is run at
rev. per min. and is operated by a 100-h-p. motor.

The product of the hammer mill is elevated and then c
bins which feed eight 40-in. Giant Griffin pulverizers, dri
h-p. vertical type alternating current motors. The ave
consumption is about 68 h-p. per mill on about 100 tons
day.

The resulting product, now ground so that 96 per ce
through a 100-mesh sieve, discharges from the mills in
conveyor which carries it to the kiln building.

*The Kilns.*—The pulverized material from the Griffin
vated to the kiln bins, which feed into two rotary kilns, 1
and 9 ft. 6 in. in diameter, set at an inclination of $\frac{3}{8}$ in. p
lined with silica brick, the feed being automatic. The kil
driven by a 50-h-p. variable-speed motor, which regulate
of calcination of the mix, under the supervision of a skille

*Coal Drying and Pulverizing.*—The drying and grinding
used in the firing of the kilns is performed in a buildin
to the kilns. The fuel used is Roundup coal from Mon
an ideal coal for cement work, on account of its low ash
atile matter (35 to 40 per cent.) and high calorific val
this coal is not available, Canadian coals are used. The
from $4.85 to $5 per ton.

The coal is unloaded from railroad cars into a bin und
road tracks, and conveyed on an inclined 20-in. conveyor
coal storage pile. From here the coal is drawn in a tu
runs underneath the storage pile, from chutes to a 20-in
veyor, which delivers it to an elevator. It is discharge
elevator into a bin which feeds into a Vulcan Iron Wo
drier, 30 ft. long and 3 ft. 6 in. in diameter.

The dried coal is elevated to two steel bins, which fe
40-in. Giant Griffin pulverizers, which reduce the coal
per cent. will pass through a 100-mesh sieve, after whi
vated and then conveyed in a screw conveyor to two steel
diately in front of the kilns, from which it is automati
and blown into the kilns with air furnished by two Sturteva

*Operation of the Kilns.*—The air, coal, and speed of th

absolute control of the operator at all times. The air com-
l by the blowers is drawn from the clinker pit and in conse-
is slightly warmed.

speed of the kilns varies; usually it is one revolution in from 1
in.

feed pipes for fuel to the kilns are to one side of the center,
so that the flame does not continuously impinge on the super-
"mix," and the mixture does not fall into the flame as the kiln
es, sintering the same to the point of incipient fusion. The
rature of the kilns is from 2,700° to 3,000° F.

brick used in the fuel-feed end of the kilns, for about 50 ft.,
ecial high-grade fire brick very low in iron, in order to with-
the high heat.

kilns each have an actual daily capacity of about 800 barrels,
e operated continuously with the exception of a semi-annual
own, for the purpose of repairs, in July and December.

clinker is discharged into the clinker pit, a narrow brick cham-
low the discharge end of the kilns.

*ing the Clinker.*—The clinker from the pit is discharged into
chain bucket elevators, which feed into four Bear cooling towers,
ing of seven cones each. These towers are shown in Fig. 4.
is fed into the buckets of the elevators, as they leave the pit,
slakes any free lime present, and partly cools and disintegrates
inker. In its passage through the cooling towers, through
is distributed 40,000 cu. ft. of air per minute, the clinker is
down and finally discharges into a steel pan conveyor.

ase there is a break down beyond this point, or too much clinker
g made, the clinker is delivered to the clinker storage pile,
h steel chutes, where it remains until it can be put through the
ng process.

*ition of Gypsum and Final Grinding.*—The gypsum, which is
to the cement in order to give it the desired set, is put in with
nker immediately following the coolers.

clinker which has been cooled, either in the cooling towers or
storage pile, is delivered to an automatic revolving scale. This
ne automatically weighs 500 lb. of clinker and then dumps.
time 500 lb. is weighed the required amount of gypsum is
, varying from 1.75 to 2 per cent.; roughly, a shovelful to each
. The gypsum used is furnished by the U. S. Gypsum Co., of
Falls, Mont. It comes in a ground form in railroad cars, which
l into a bin underneath the tracks, and is delivered to a storage
om which it is drawn as required.

The clinker with the gypsum added is now sent to a S rotary crusher by way of a 20-in. belt conveyor. The prod preliminary crusher is elevated to the top of the building and in a screw conveyor to bins over the final grinding machin

The final grinding is performed in seven 40-in. Giant Gri driven by 75-h-p. vertical type alternating current mot power consumption is from 72 to 78 h-p. per mill. The s the mills are 40 mesh, the resulting product (cement) bei cent. through 200-mesh and 95 per cent. through 100-mes

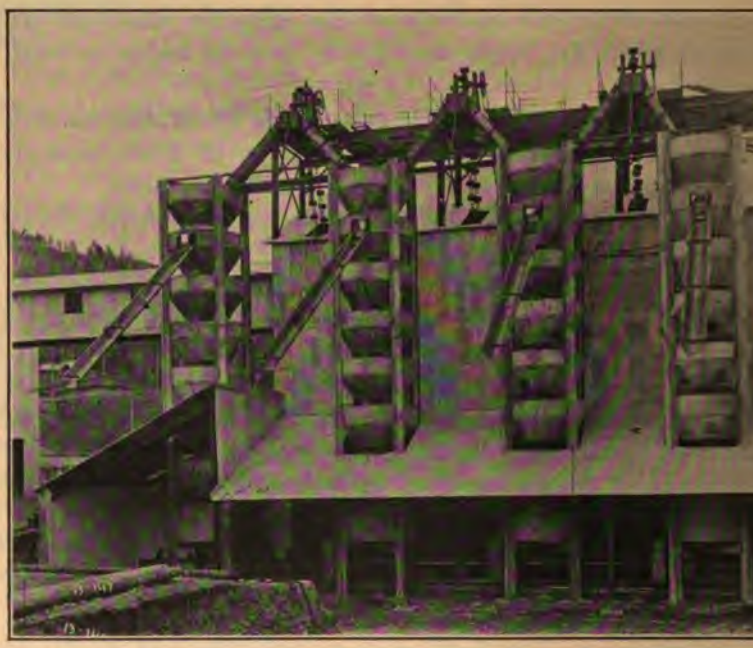
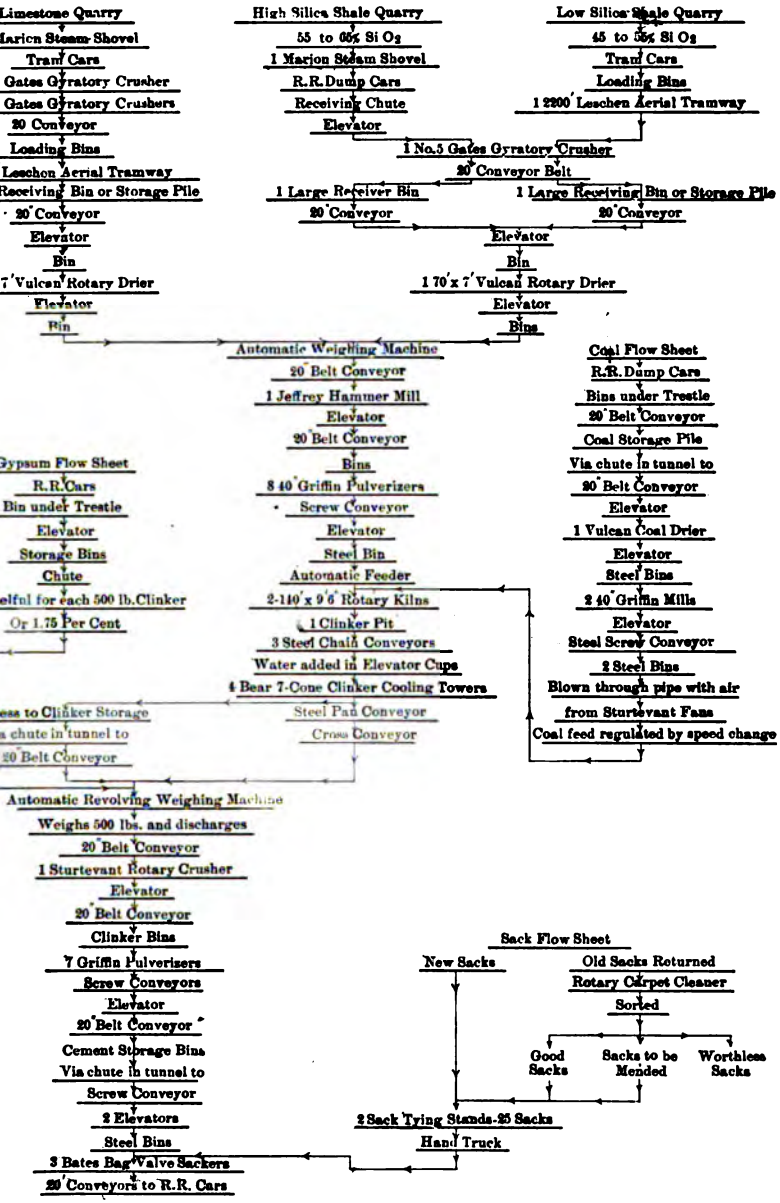


FIG. 4.—BEAR CLINKER COOLING TOWERS.

The finished cement is now sent to the cement storage means of an elevator and conveyors. The cement storage i into two parts, each capable of holding 110,000 barrels o This gives an ample capacity for holding the cement until i thoroughly analyzed and tested.

*Sacking the Cement.*—The cement after having been t tested is drawn from the bottom of the storage bins, throug to screw conveyors which operate in tunnels, and is deliver of two elevators, which discharge into bins over thre machines, which were furnished by the Bates Bag Valv




5.—Flow Sheet of the Plant of the Inland Portland Cement Co.

Chicago. From the sackers the sacked cement is discharged veyor and delivered directly into railroad cars.

The equipment includes a carpet-cleaning type revolving cleaning returned sacks, a sack repair room, and stands sacks before sending them to the Bates sackers.

The output of the plant is from 1,500 to 1,600 barrels per mill is operated in two shifts of 12 hr., labor being obtaine $2.50 to $2.75 per day of 10 hr. A complete flow sheet of ation of the plant is given in Fig. 5.

*Analyses of Raw Materials and Cement.*—The following represent samples of the lime rock, high-silica and low-si and the finished cement:

| | Lime-stone. | Calcareous Shale. | | | High-Silica Sha | |
|---|---|---|---|---|---|---|
| | | No. 1. | No. 2. | No. 3. | No. 1. | No. |
| Loss on ignition ($CO_2$).. | 38.98 | 19.77 | 21.01 | 15.2 | 3.73 | 5. |
| Silica ($SiO_2$)..... | 7.00 | 32.48 | 31.96 | 40.5 | 74.40 | 71. |
| Iron ($Fe_2O_3$)..... | } 5.32 | 19.92 | 18.00 | 5.2 | } 13.00 | 14. |
| Alumina ($Al_2O_3$)..... | | | | 16.4 | | |
| Lime (CaO)..... | 47.50 | 20.41 | 22.61 | 15.4 | 1.57 | 3. |
| Magnesia (MgO)..... | 1.52 | 3.42 | 2.97 | 2.7 | 2.28 | 3. |
| Sulphur trioxide ($SO_3$).. | ..... | ..... | ..... | ..... | ..... | ..... |
| Cementation index..... | ..... | ..... | ..... | ..... | ..... | ..... |

*General.*—Since commencing operations the Inland Portla Co. has supplied all the cement for the city of Spokane an or the Inland Empire, as it is called.

Practically all of the recent power installations in that vi used this cement. The most notable of all is the Long L dam of the Washington Power Co., in which about 275,00 cement will be used.

The equipment of the Inland plant includes a well-equip and chemical laboratory, shops, etc.

I wish to express my thanks to the officials of the Inlan Cement Co., for their kind aid in furnishing data for this pap larly Dan R. Brown, Treasurer, and Irving J. Kohler, Supe

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.
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# Valuation of Coal Land.

BY H. M. CHANCE, PHILADELPHIA, PA.

(Butte Meeting, August, 1913.)

QUATE treatment of the difficulties surrounding the valuation neral lands requires that agreement be first reached defining as understood for the purpose of appraisal. To define value as ice of a product is not sufficient, and does not assist us in using tors that cause variations in values and in relative values. The lue of any product depends primarily upon its relation to the ities, comforts and pleasures of mankind. As the necessities, rts and pleasures of the civilized races differ from those of more ive man, the value of products varies with the degree of civiliza- nd the environment of the consumer. Coal has no value to the es of warm climates, and arrow-head flint none to the residents v York. It has long been recognized that what has been termed nsic value" must be limited to materials that are essential to nce—air, water, food.

"intrinsic value" cannot be measured in monetary terms. ntrinsic value of a bushel of wheat is due to the fact that it upport the life of an individual for a certain definite time, but ct does not enable us to measure or to express that value in ary terms. It has often been proposed to adopt as a measure value of any product the quantity of labor necessary to produce this method is of limited application, and will not apply to may be termed "intrinsic values."

many reasons it might be well, while assenting to the fact that aterial may be intrinsically valuable, to abandon the use of the intrinsic value." One of these reasons is the tendency to con- he idea of intrinsic value with a value that is conferred by mic conditions, that is purely artificial and that may be perma- r evanescent. Intrinsic value must from its nature be permanent nchangeable.

sibly for present purposes the value of any material may be ered as having a dual origin: First, that due to economic

conditions, whereby, demand having made the material d
comes to have value wherever found; and second, that d
labor expended in producing the material. The term la
used in the broadest sense to include labor of all kinds e
production (capital being regarded as accumulated reserve
preparation for use, and in transportation to the consumer.

The first subdivision of value defined above is that w
we have to deal and concerns value conferred upon a mate
relation of supply to demand. If the supply be abund
demand be small, the material will have no definable val
deposits of limestone in countries where limestone is th
rock have no definable value.

In attempting to determine the value of coal or miner
therefore seems desirable to dismiss the idea of inherent
value; and to confine consideration solely to those econo
tions that justify valuation expressed in terms of a monetar

As these values are due to economic conditions whic
permanent, but are subject to fluctuation and change, the
likewise rise or fall in correspondence with such changes, a
thus never hope or expect to do more than reach a result w
time will fairly represent existing value.

Under the conditions of modern civilization coal is neces
continuance of human life. It has thus become valuabl
savage it has no value. As it becomes more necessary
increases. It is not a substance inherently valuable as n
the continued existence of the race. Food has such valu
has such value, and outside tropical and semi-tropical coun
some sort has such value. The savage never used coal an
races have developed and maintained highly organized civi
countries where it was practically unknown, so that its val
predicated entirely upon those requirements of modern
which have made it a necessity.

We are not able to determine the intrinsic value of
value of products which are not necessities, and which c
duced in unlimited quantities, is fixed by the cost of prod
this law does not hold good for products which are neces
continued existence of civilized life, and since coal has
necessity, and as it is a substance of which the readily
supply is not unlimited, its value increases as the dem
grows.

Hence we find throughout the world that the value of c
constantly increasing, and the rate of increase is more rapi

se in the value of farm land, or land containing limestone, ng stone, or other similar useful materials.

s rapid increase in value, often marked by great relative dis- s in current values in different localities, has stimulated study subject with a view to the development of systematic methods termining value. The subject has attracted the attention not f coal operators, but of investors, financiers, and political econo- and has become important to those charged with the raising enues for defraying the expenses of government, for national, county, and township requirements.

as also become a matter of great, if not of vital, importance to ation, as affecting the sale, or other disposal, of government containing coal.

hods used for determining the price at which it is deemed ex- t to sell coal lands owned by the United States may not be able to appraisals of value upon which the financing of coal ions is to be based, and methods suitable for either of these ses may not properly be applicable to valuations made for the g of taxes.

value of coal may or may not be a factor in fixing the price ch coal lands, or coal-mining rights, are offered for sale by the l States. In the past it was deemed desirable te encourage the g of coal mines by offering the lands (subject to certain condi- easy of fulfillment by an intending purchaser) at the nominal of $10 and $20 per acre, the higher price being placed upon lands as were within 20 miles of an existing railroad, and r, to secure such development as well as to secure the construc- f railroads through the public domain, large grants of coal were given as a bonus for the construction of such roads. In localities it may still be good policy for the government to rage or stimulate the development and working of coal lands ering them at relatively low prices to those who will develop ork them, but with few exceptions this policy is no longer ary to insure the opening of mines.

ill perhaps be conceded without argument that the price paid operator for coal land or for coal-mining rights must be repaid through the sale of coal. The coal operator must fix a selling for his product sufficient to include the cost of mining, the paid for the coal land, and interest on the amount invested in nent and working capital, for if his product be sold for less uch price, the operation will be unprofitable and will soon be oned. As it is thus clear that the consumer must repay to the

coal operator the price paid for the land, the sale of coal land
land prices" by the government becomes in fact merely a
of raising money by taxing the consumer. The wisdom
method of raising revenue is open to argument.

For some years we have heard the advocates of conser
ploiting their pet hobby. One of the principal, and perha
the most popular, slogans of these enthusiasts has been the
tion of coal. As commonly understood this means the ret
the government of most of its coal lands, presumably for t
of future generations.

If we have good reason to fear that the coal supply will
cient for those who will follow us, it is indeed a beautiful
that requires us to preserve it for posterity, even if such pr
requires self-denial and imposes some hardships upon th
generation. It is, however, not clear that the present ge
willing to impose upon itself such economies or self-deni
bring about this result. If such a policy be actually ad
carried out, the preservation of the supply would follow w
lands containing coal were owned by the government or b
uals or corporations, for whatever fuel the people require
mined and delivered to them, and this is quite independe
question of ownership.

True conservation points rather to the efficient and eco
of our fuels (rather than to parsimony in their distribution)
mining methods and to improvements in power-generating m
to the end that more of the coal be recovered in mining and
of the heat units generated by combustion be transformed i
and utilized in doing useful work.

The established policy of the government is, however,
defined as a plan to utilize its coal lands by selling them at
price as the coal operator can be induced to pay. As alread
out, the effect of this policy is to increase the cost of coal t
sumer, and this in turn tends to promote that form of co
(which many are now preaching) which curtails the consu
decreasing the ability of the consumer to purchase as mu
would like to use. Those who advocate this policy justify
argument that it is the only way in which the government
for itself the unearned increment in value due to the stea
prices of coal land. This may readily be granted, but it mu
admitted that in adopting this policy the government is it
exactly what conservationists condemn and object to when
by the individual owner of coal land. In both cases the pr

g the unearned increment due to the enhanced value of coal
s paid by the consumer.

owner of coal or mineral property rightly estimates its value
al to the price for which it can be sold. Whether the property
rked under lease, the coal or mineral being paid for as it is ex-
l, or the property be sold outright and paid for at the time of
r in deferred payments extending over a prolonged period, is
erial—the principle of valuation is the same in both cases. If
nt be made in deferred payments, or by royalties extending
long term of years, the net amounts to be paid, or estimated
paid after deducting taxes and other expenses, are discounted
e computed present money value of the combined payments is
as the value of the property. If such property is being worked
owner, that does not alter the principle of valuation, and the
ing profit—which is the owner's reward for working capital
ed, business risks assumed, and for his own services—should
credited to the property and used as a measure for determining
ue.

eterminations of value based upon future revenue, the rate of
t used in calculating the present value is a most important
When there is no doubt concerning the quantity and quality
or other mineral present in any property, the interest rate may
ably be taken at the lowest current rate for stable investment
ies, because the owner is reasonably secure against loss and the
ed capital is subjected to a minimum risk. In fact, many capi-
are quite willing to permit investments of this kind to remain
urbed for years, yielding no revenue, provided they show an in-
in value equal to 2 or 3 per cent. annually.

he other hand, it is almost impossible to secure funds for financ-
al-land purchases or coal-mining operations by the issuance of
bearing less than 6 per cent. interest, and such bonds are often
a discount, or accompanied by stock or other bonus in order
ct their sale.

e can agree upon an interest rate to use in computing the pres-
lue of deferred payments (and prospective royalties); if we can
present current royalty value per ton, either by existing con-
or by contracts possible of execution; if we can determine the
ty and quality of workable and recoverable coal or mineral;
we can agree upon a rate of working that will insure the mining
the deposit in a fixed time, we will then have the principal ele-
necessary for calculating the present value of the property. In
on to these it is of course necessary to know the cost of

looking after the property, taxes and miscellaneous expe
are to be deducted from the revenue upon which the cal
value is based. The essential factors are then:

1. Rate of interest.
2. Royalty per ton.
3. Quantity of merchantable coal (recoverable coal).
4. Rate at which coal will be mined.
5. Taxes and miscellaneous expenses.

This method is that currently used in commercial appra
properties made for guidance in purchasing or in selling,
upon which the property may be used as security for the
bonds secured by mortgage.

It is subject to serious criticism on several grounds. It
that for complete appreciation of the result reached th
should state the interest rate, royalty, rate of mining, tim
to exhaust and total quantity of recoverable merchantabl
in making the appraisal, because what may appear to be
ences in any of these factors may cause great discorda
values as estimated by different engineers. The method i
able because it assumes a fixed royalty value per ton exte
the whole period necessary to exhaust. The history of
development almost without exception shows a tendency
increase in royalties, and while failure to recognize this te
be thought to constitute a factor of safety against over
must not be forgotten that a decline in future royalty
impossible.

For these and other reasons, it does not seem entirely sa
valuations made by this method unless they are substant
mony with prices at which similar property has been or c
but in making this statement I do not intend to imply t
sales values should be accepted in preference to values ca
this method, for the method frequently enables us to det
coal lands in any locality are selling at less than their rea
that their purchase can therefore be recommended, and o
hand it may often enable the owner to avoid the mistake
a price materially below the real value.

It is, however, the custom of engineers to check valuatio
by comparison with actual sales prices whenever this can

But one of the five elements entering into such com
value is fixed by nature—namely, the quantity of mercha
all of the other factors vary with changing economic condi
rate of interest is fixed by world-wide financial condition

al economic conditions; the tax rate is affected principally by conditions, but indirectly may be influenced by State or national ements; the rate at which the property can be worked is often ly within the control of the owner, or owner and operator com- and the royalty per ton, while affected by general economic ions, is more likely to be controlled by local conditions.

nce we may state that of the five factors which enter into such d of fixing value, one (quantity) is fixed by nature; one (rate erest) varies with world-wide and also local financial conditions; axes) varies with local governmental revenue requirements; one ty) varies with general economic conditions and is controlled by ompetition; one (time to exhaust property) is largely within the l of the owner if he has the means and ability to secure rapid ng. The problem to be solved is therefore one including one quantity and four variables.

ile the object of this paper is to discuss methods adapted to ercial valuations, a review of the methods used by the U. S. nment may assist us in getting a broader view of the subject, ally in relation to the influence of facts that affect the commer- lability of the coal and the cost of mining. In the preceding these have not been dwelt upon because the effect of thick- quality, depth, etc., upon the value is measured by the royalty on, a thin coal, a poor coal, or a coal deep beneath the surface anding correspondingly small royalty.

method now used by the government for fixing the selling of coal lands is outlined and discussed at length in a recent *in*[1] of the Geological Survey. The procedure involves the use ystem that has been invented to cover almost every conceivable tion, and to include most, if not all, of the variable physical fac- hat affect the value of coal land. In the order of their import- these are thickness, depth beneath the surface, and quality. ingenuity has been shown by Messrs. Mendenhall, Ashley, and in reducing to mathematical expression some of the schemes d for simplifying the calculations by which the relative values l differing widely in thickness, quality, and depth are determined. f course evident that most such values are necessarily based upon arily assumed units, and the valuations are therefore not of the cter required for commercial and financial purposes. This does owever, constitute a valid or forceful criticism, because the gov- ent may ask any price for its lands, being under no obligation at the true value or at any predetermined price.

[1] *Bulletin* No. 537 (1913).

Some of these assumed units are as follows:

Land containing coal having a thermal value of less t B.t.u. on an air-dried, unweathered sample is assumed t value as coal land and is classified as non-coal territory.

Coal less than 14 in. in thickness is considered worthles of that thickness is likewise classified as of no value wl more than 500 ft. beneath the surface or when its heat va than 12,000 B.t.u., thus:

| Heat Value. B.t.u. | Minimum Workable Thickness. Inches. | Heat Value. B.t.u. | Minimum Workable Thickness. Inches. | Heat Value. B.t.u. |
|---|---|---|---|---|
| 15,000 to 12,000, . . . | 14 | 9,600 to 9,500, . . . | 23 | 8,800 to 8,700, |
| 12,000 to 11,000, . . . | 15 | 9,500 to 9,400, . . . | 24 | 8,700 to 8,600, |
| 11,000 to 10,500, . . . | 16 | 9,400 to 9,300, . . . | 25 | 8,600 to 8,500, |
| 10,500 to 10,250, . . . | 17 | 9,300 to 9,200, . . . | 26 | 8,500 to 8,400, |
| 10,250 to 10,000, . . . | 18 | 9,200 to 9,100, . . . | 27 | 8,400 to 8,300, |
| 10,000 to 9,900, . . . | 19 | 9,100 to 9,000, . . . | 28 | 8,300 to 8,200, |
| 9,900 to 9,800, . . . | 20 | 9,000 to 8,900, . . . | 29 | 8,200 to 8,100, |
| 9,800 to 9,700, . . . | 21 | 8,900 to 8,800, . . . | 30 | 8,100 to 8,000, |
| 9,700 to 9,600, . . . | 22 | | | |

It is assumed that a coal 6 ft. thick can be mined to th depth at which any coal of given quantity can be considered The following formula, deduced from the Survey rules, illu method adopted for calculating the depth to which a co thickness less than 6 ft. and yielding any heat value betw and 15,000 B.t.u. may be considered workable.

$$\text{MWD} = \frac{\text{t} - \text{mwt}}{72 - \text{mwt}} \times \left(\frac{\text{B.t.u.}}{3} - 500\right) + 500$$

Where MWD is the required maximum workable dept is the thickness of the coal in inches, mwt is the minimum thickness (as given in the above table) in inches; B.t.u. value of the coal in question, in British thermal units.

This formula is based on the assumption that the maxin to which any coal can be worked can be found by dividing value by 3; thus a 15,000-B.t.u. coal is considered workabl more in thickness to a depth of 5,000 ft., a 9,000-B.t.u. coal of 3,000 ft., etc.

The above formula, for coal showing 9,800 B.t.u., giv mum workable depth for a thickness of 48 in. of 1,989 ft.,

$$\text{D} = \frac{48 - 20}{72 - 20} \times \left(\frac{9{,}800}{3} - 500\right) + 500 = 1{,}989$$

while the same coal if 6 ft. thick would be considered wo depth of 3,266 ft., although a 4-ft. coal with 15,000 B.t.u.

l as unworkable at any depth greater than 3,138 ft. This us directly to the issue underlying any such system of valu- for it must fail if its results do not coincide with the experi- of operators and financiers in the actual working of coal ties. It is patent that under exceptional circumstances only a coal 6 ft. thick yielding 9,800 B.t.u. be workable to the depth as a coal 4 ft. thick yielding 15,000 B.t.u. In many the coal of lower heat value would be absolutely unsalable and valueless, while that with the higher heat value would be in d at the highest market price, so that an increase in thick- om 4 to 6 ft. cannot be thought to compensate for the differ- between 9,800 and 15,000 B.t.u. It is of course true that the eat units which can be obtained from all of the coal contained given area of two such coal beds will be nearly equal, that t. of 9,800-B.t.u. coal will yield nearly as much available s 4 ft. of 15,000-B.t.u. coal, but to put the two on an equal ercial basis it would be necessary to mine and deliver to the ner 1.5 tons of the poorer coal to compete with one ton of the coal, and, if the better coal were available, it is probable that nsumer would not accept the poorer coal unless he could buy two tons of it for the price of one ton of the better coal. ver, disregarding entirely the commercial phase of the ques- nd considering the matter solely from an operative standpoint, evident, other things being equal, that it is impossible to 1.5 tons of coal from a 6-ft. bed at the same cost as one ton a 4-ft. bed; but even if this were possible it would not justify acing of the two coals upon a plane of equality as to workability, e the inferior coal while being produced at the same cost have a value less than two-thirds that of the better coal.

er a consideration of many matters affecting the varying of coal in the ground, the statement is made that one cent per recoverable coal was finally fixed as a unit to apply to coal g a heat value of 12,500 B.t.u., existing at a depth of 500 ft. s, and having a thickness of from 6 to 10 ft., allowances being to increase or diminish this value for higher or lower heat and for decrease in thickness and increase in depth.

nbining the stated rules and formula we find that the value, ted for differences in thickness and heat values, may be shown following expression :

$$V_1 = \frac{\text{B.t.u.}}{12{,}500} \left(4T + T^2\right)$$

in which T is the thickness of the coal in feet;

B.t.u. is the heat value in British thermal units; a

$V_1$ is the value of the land in dollars per acre;

value as finally corrected for depth is expressed

$$V = \frac{MWD - D}{MWD - 500} \times \frac{B.t.u.}{12,500} \left(4T + T^2\right)$$

in which MWB is the maximum workable depth and

D is the depth of the coal whose value is sought.

The development of the rules which have made it possible the mode of treating the subject by algebraic formulæ is l esting and instructive, and while in the opinion of the writ little prospect of working out a means for adapting them to t ments of the engineer engaged in commercial appraisals the possibilities of analytical study of the subject better tha that has been published.

It is hardly possible to discuss the subject intelligent explaining more fully the principles adopted by the Survey ing factors, and I therefore will abstract from the *Bul* referred to some of the more important statements, some c believe are not entirely in accord with the experience of op investors. These latter are indicated by an interroga inserted thus (?).

"Workability of coal . . . depends on factors of two the first type,—quality, thickness, depth,—is intrinsic . . ond type,— . . . transportation and markets,—is extrins new railroad may make a . . . district or may break it. new coal field . . . only the most accessible, thickest coal can be worked at a profit. . . . It must be considered if its value . . . exceeds the cost of extraction. . . . Mo unworkable because it is too thin than from any other c The price paid for coal in the ground should be recover investor during the early years of mining, when the cost is and should be refunded within the first 20 years of the life o . . . If the royalty rate is 10 cents a ton, the value of coal the ground when a mine is opened is at least 5 cents (?) . . a ton is not far from the average royalty paid under private l 2½ cents a ton is a fair sale price for unmined coal that is t immediately, (?) where 10 cents per ton is the prevailin The government valuations cannot take account of change petition, markets, transportation facilities, or freight rates factors that affect the profit. (?) . . . The base price of g coal land is fixed at one cent per ton of 12,500 B.t.u. .

: to one-tenth the assumed average royalty . . . the royalty is d on the profit. . . . Whatever affects profits affects the royalty. The depth to which any coal can be mined is assumed to be ctly proportional (?) to the B.t.u. value of the coal and inversely ortional to the cost of mining (?) for different thicknesses. . . . value of coal is inversely proportional (?) to the cost of mining table on page 84)."

his last view is doubtless the result of abstract reasoning, it being that if one coal can be worked at half the cost of mining another it must be just twice as valuable, because with the same labor an produce from it a product of just twice the value. This is otless another example of the confusion in reasoning that arises a belief in an inherent intrinsic value. If we are to reach sound lusions it is essential to remember that the values we are dealing are those that are measurable by the profits the material can be e to yield, and that the profit does not bear a fixed relation to the of production. In the example just cited it is easily conceivable the profit in working the more cheaply mined coal might be ten s as great as in working the other coal, in which event it would be imes as valuable instead of only twice as valuable. The adoption of a principle or rule for determining the effect of the cost of mining the value of coal cannot be accepted as correct in principle. It ainly would be impossible of application in making commercial ations. If two coals are of equal quality and will sell for the e price per ton any excess in the cost of mining one over the cost ining the other will of course reduce the profit per ton by a sum tly equal to such excess, and as the profit on coal is usually not e than 15 or 20 per cent. of the cost of production, large differences e cost of mining, such as come from variation in thickness, depth, must inevitably destroy all possibility of profitably working the e expensive coal. Such coal cannot be held to have any definable ppreciable value until the competition of more cheaply mined coal iminated by its exhaustion.

the value is not inversely proportional to the cost of mining, it not seem reasonable to estimate the depth to which a coal can be ked by a rule that fixes the depth as in such inverse proportion, it is difficult to find any other reason that would justify such a d statement. It does not by any means follow that because a can be mined for half the cost of working another coal that it be worked to twice the depth. The similar assumption as to the tion between the heat value of any coal and the depth to which it be worked does not appear to have sufficient force to be adopted

as a principle of general application. The interrogation
the above abstracts concerning the fixing of values at 5 an
per ton are intended merely to indicate the personal opi
writer that such values are much higher than any figures
used as representing general averages. Thousands of a
are constantly being sold at prices ranging from 0.5 to 1.
ton, coal that is easily accessible to existing railroads, wh
that is some distance from transportation prices comm
from 0.25 to 1 cent per ton. These figures apply to larg
our Eastern coal fields and are intended to illustrate pri
containing coal of fair grade and of fair workable thickn
grade coal and coal of more than average thickness b
prices, but the figures to which exception is taken are re
in the case of the higher-grade coal in districts where it
evident that the supply of such coal is or will be insuffici
the demand.

While it may be impossible to derive any formula that
real use in simplifying the commercial valuation of co
algebraic expression of the principles involved may be o
in giving a clearer insight into the relations of the variabl
stants with which we have to deal than can be gained b
alone.

The simplest form of an equation illustrating current
appraisal is,

$$V = c\ d\ P\ r\ T \quad . \quad . \quad . \quad . \quad .$$

where V is the value in cents per ton;
P is the profit in cents per ton;
T is the thickness of the coal in feet;
r is the recovery in tons per acre per foo
ness—this may be taken at 1,000 to 1,30
c is a coefficient whose value depends upon t
quired to exhaust the property, interest ra
etc., and will usually have a value betw
0.5;
d is a coefficient to represent the proportion of
that is to be credited to the coal (the bal
credited to working capital, etc.), and
be taken at 0.3 to 0.6.

A 4-ft. coal yielding a total profit of 20 cents per ton, 0.
is credited to the coal, on 1,200 tons per foot-acre recove
years to exhaust, 5 per cent. being the interest rate adop

margin to the purchaser of about 100 per cent., may be used to ate, thus

$$V = 0.3 \times 0.4 \times 20 \times 1{,}200 \times 4 = \$115.20.$$

o profit to purchaser be provided for, the value of c is about nd the value per acre will be $249.

ation (*A*) shows at once that the value per acre increases ly in proportion with each of its factors. As r and T are fixed, he variations of c and d in any given case will be confined small limits, the important variable is P.

this quantity, P (the profit) is the difference between the sale realized per ton and the cost of production, which may vary y, slight increases or decreases in the cost of production, in the rice, or in both, may reduce its value to zero or increase it several Hence it is evident that the value per acre depends prin- y upon this one item, all others sinking into insignificance it.

e sale price is affected principally by the quality of the coal.

e cost of mining is affected principally by the thickness of the and to lesser degree by the physical conditions, slate partings epth beneath the surface.

enter upon a discussion of the increase in mining cost due to cal conditions, underground and at the surface, to slate and refuse in the coal bed, and to depth beneath the surface, would d this discussion far beyond reasonable limits and it is therefore sed to consider only the principal cause (thickness) of large ences in mining costs, and the principal cause (quality) of varia- in sale price.

e term mining cost or cost of mining is used to include selling issions, taxes, and all other costs incident to extracting and ring coal to the consumer, excepting only freight and interest pital invested.

the purpose of illustration let us assume that in the markets the coal is to be sold, coal of average grade, say 13,000 B.t.u. value, commands an average sale price (ASP) of $2.70 per ton, eight rate being $1.50 and the net price $1.20 per ton. If the f mining (CM), including a selling commission of 10 cents, is $1 n, the profit (P of the above formula) is 20 cents.

t if the coal be of better or poorer grade, the actual sale price will not be equal to the current average sale price for coals of 0 B.t.u. value, but will be decreased or increased thus

$$SP = ASP \frac{\text{B.t.u.}}{13{,}000}, \quad . \quad . \quad . \quad . \quad (B)$$

and if the coal has a heat value of 14,300 B.t.u. the sale
ized should be about

$$SP = \$2.70 \frac{14,300}{13,000} = \$2.97 \text{ per ton,}$$

or 27 cents more than the average current sale price, thus
the profit from 20 cents to 47 cents. We thus have a con
increases the value per acre by 135 per cent., the value o
preceding example being $115.20, which is increased to

$$V = 0.3 \times 0.4 \times 47 \times 1,200 \times 4 = \$270.72.$$

This example has been worked out to show graphical
dominating importance of the quality of the coal. The
this predominating importance is not generally underst
found in the fact that the operator having a coal worth 1
more than average competitive coals, collects this 10 pe
only upon the cost of mining but upon the freight also.
the freight rate the greater his profit. He also collects the
10 per cent. on the profit which he would derive from th
13,000-B.t.u. coal, for the normal sale price (SP) is equal
of mining (CM) plus freight plus profit (P), thus

$$SP = CM + \text{freight} + P, \text{ and as } SP = ASP \frac{\text{B.t.u.}}{13,00}$$

$$P = ASP \frac{\text{B.t.u.}}{13,000} - CM - \text{freight}, \quad . \quad . \quad .$$

and while the selling price is increased 10 per cent. in
example from $2.70 to $2.97, the net price is increased f
to $1.47, an increase of 22.5 per cent., while the profit rise
to 47 cents, an increase of 135 per cent. This ability of
of high-grade coal to collect premiums on the freight paid
the higher the freight the greater his profit, is the reason
coals naturally seek far distant rather than nearby markets
nearby market the premium is correspondingly less.

This phase of the commercial end of the business, espec
the purchase of coal on a B.t.u. basis has extended to
sumers, is accountable for the very large prices at which
coal lands are now sold, and the general appraisal of such
class by themselves at valuations that are very much large
be justified by the average profits of the business.

It has already been stated that any increase in the cost
reduces the profit by exactly the amount of such increase.

is therefore extremely dangerous, for if we have a normal profit ting to 15 to 25 per cent. of the cost of mining, very small s in physical condition or decrease in the thickness of the coal crease the cost by an amount equal to or greater than the nor- ofit. Increase in cost from poor management or other reme- ndition (for the reason that such cause is remediable and should edied) is not considered as a factor affecting valuations.

icient has already been written concerning the rapid increase t which attends each, even small, decrease in thickness. The l point seems to be at about 4 ft., coal of 4 ft. being workable slight increase over the cost of mining a coal 5 ft. thick, and 5 ft. thick being workable at almost the same cost as a coal 6 k; 6 ft. commonly being thought the ideal thickness, giving oom for men, mules, motors, etc., and requiring less timber or timber than beds of greater thickness.

the thickness decreases below 4 ft. the rise in mining cost is the cost in a 3-ft. coal often being from 50 to 80 per cent. than the cost of mining coal 4 ft. thick, and the increase in coals less than 3 ft. thick is still more rapid.

districts where coal 4 ft. thick is available in quantity, this nce in mining cost, unless accompanied by more than com- ing difference in quality, absolutely prevents the working of ft. thick. Thin coals existing under such conditions can be no assignable present value except by assuming that in a given er of years the thicker coal will be exhausted and mining of inner coal will become possible. In most cases the time ed to exhaust the thicker coal is so long that the prospective of the thinner coal, when discounted to a present money value shrinks to small proportions, becoming quite unimportant as red with the value conferred upon the property by the thicker For this reason it has become quite common to base commer- praisals upon the thicker and better coals and to ignore the ely thinner and poorer coals.

algebraic formulæ have been introduced by the writer for the se of graphic illustration and without intending to suggest their the actual appraisal of coal lands, because it seems impossible ise any equation to include the effect of variations in all the that tend to increase or decrease costs. The influence of thickness, irregularities, partings, dip, roof, floor, gas (fire- , spontaneous combustion, dust, swamps, faults, floods, water and other local physical conditions cannot be represented by aic symbols.

Equation (*A*) is merely one expression of the meth common use for valuing mineral properties and has repe elaborated in technical literature in this and other co the opinion of the writer it is about as far as we can go i to formulate general rules for coal appraisals. It may be as of metalliferous properties, that no two are exactly property must be treated separately, its peculiarities stud measured, sampled, analyzed, and their heat value deter it is possible to attempt an appraisal of value. In makin ination careful study should be made of all those conditio favorably or unfavorably affect the cost of mining. mining cost in the district in which the property is loc the standard by which the property should be judged. cost be less than the average cost in the district, and th a quality equal to the average of the district, the profit w spondingly larger than average profits of other operatio *versa.* In the same way great care should be taken to n the quality of the coal, not only of its quality as it e ground, but of its quality as mined and shipped to mark features are investigated with proper care it is usually q to reach conclusions as to the value of any property th the test of time.

In estimating profits the competition of coals mi same district and shipped to the same markets is not especi ant, as it is this competition that normally fixes t profit at from 15 to 25 per cent. of the actual net cost but it is important to look closely into the competitive coal from other districts, this being especially true when values in an undeveloped region. The coal of any dist considered not only with reference to nearby coals but als compared with those of other districts that may sell thei the same markets.

As an illustration of the importance of these considerat cessful working of relatively thin coals, often of rather p in Kansas and Michigan which justify values often ap or exceeding $100 per acre may be quoted. Coals of lik and quality existing in the principal mining district of P or West Virginia would be considered valueless.

*Enhancement in Value.*

In the preceding discussion no reference has been made upon present values of future enhancement in value. T

city, suburban and farming property, thought to be in an im-
ıg position, is sold are usually higher than the present value as
ıred by net earnings. Central property in large cities often
t prices upon which the rentals yield net earnings of 2 or 3 (or
ɔer cent. Investors in such property realize that in paying the
asked, they are anticipating the enhancement in value for 5 or
ɔre years. Only the most stable kind of real estate is valued
an investment basis, that is, upon its normal average net earn-

hough it often may be desirable in valuing coal lands to make
ınces for future enhancement upon present values, this can
be done. In the case of non-mineral real estate, the price that
r property commands usually includes full allowances for future
es in value, and this may also be true of coal land, but in many
istricts transfers of property are few and infrequent, the prop-
sold may not be similar to the lands in question, the real sale
may be concealed or misstated in the conveyance, so that it may
ficult if not impossible to determine current sale prices, or the
ɔn of such prices to the value based on earnings.
s question has often arisen in controversies over the assessment
l land for taxation, without, however, developing a rule or law
neral applicability by which such controversies can be settled.
at coal land tends steadily to enhance in value cannot be doubted.
nmerable instances of such increase in value might be quoted,
depreciation in value has occurred in few districts and such de-
tion has usually been the aftermath of an unwarranted inflation
normally followed the mining out of the best coal. In some
ts coal land has fallen in value through the competition of
-opened coal fields supplying cheaper or better coal. It may,
ver, safely be assumed that the value of coal land grows steadily,
ıg pace with the increasing demand for coal, and there seems
no reason to doubt the continuance of this tendency for as long
od as the consumption continues to increase.

*Royalties.*

s tendency is well illustrated by the increase in current royal-
In districts approaching exhaustion this increase is rapid and
es large proportions, especially in localities producing coal of
grade. In the high-grade steam-coal districts along the eastern
of the Allegheny mountains in Pennsylvania and Maryland,
ɔriginally leased at 6, 8, and 10 cents per ton, 30 years ago, has
ently been re-leased or sub-let at royalties of 12, 15, 18, or 20

cents per ton, and such later leases, made at the higher fig
often been sold for a larger bonus per acre. In the anth
tricts the enhancement in value has long been recognized,
tion of this increase is sometimes secured for the owner o
by making the royalty a percentage of the price at which
marketed.

*Legal Principles Affecting Values.*

As the value determined by the engineer may be used
purposes than that for which the appraisal was made, the
ciples affecting valuations should not be overlooked.

By decisions in many parts of the country, constituting
not likely to be reversed, the principle seems well establish
the purpose of taxation all property (including coal land, o
separately as a portion of the real estate) must be assesse
and equitably at its fair market value, or at a definite per
such market value. Many decisions have defined the ma
as the price that can be obtained after the property has bee
offered and advertised, but not necessarily the price re
forced sale at which the property is offered without upse
limit. When the value cannot be determined in this
methods of fixing values may be held to be valid, but the
seems to be to enforce the same method in fixing coal valu
used for other classes of real estate. Methods based on th
or any other plan depending upon the quantity of coal o
royalty, or other income, have not fared well as a basis fo
The situation in all States is not the same, differences in the
stitutions and laws governing taxation naturally causing
in the decisions, but the general tendency is as already st
however, there are constantly increasing difficulties in att
apply these principles to the valuation of coal lands, char
tax laws, even if they involve amendment to constitutions,
all improbable, especially in States in which coal mining is
tant industry.

Issues of bonds secured by mortgages upon coal land or
mining rights, or issues of stock (as full-paid) upon coal
rights, should be based upon real values. If the assume
fictitious, or overestimated, the directors and officers of th
making such issue may in some cases be held personally r
although there may have been no intent to mislead or defr
holders of stock—supposedly full-paid—may be held to b
the debts of the company to the extent of such unpaid val

Appraisals of value may be needed by those intending t

and, by owners contemplating its sale, by capitalists engaged in, anning, its development, by brokers or middlemen aiming to a profit by effecting its sale, by owners wishing to make it the of a bond or stock issue, by assessors for taxation, or by execu- or heirs for the purpose of subdivision or allotment. As an isal made for any of these purposes may at some time be used me other purpose, the appraisal should be accompanied by a lete statement of the facts, the method used in making the tion, the object for which it was made, and the personal opinion engineer as to the relative importance, value, etc., of the known not only as an explanation of the appraisal, but to guard against advertent or intentional misuse.

*Value and Sale Prices.*

nen the value indicated by current sale prices is greater than computed from the net income the property can be expected to the difference may be due to over-optimism or ignorance on the of the purchasers, or it may represent value predicated upon e prospects for higher prices. Whether such future enhance- should be included in fixing values is not entirely clear. Deci- of the courts that valuations be made at actual sale prices re assessments for taxation to include such value, although it not seem reasonable to compel the payment of taxes on a value does not and that never may exist. On the other hand, the pur- r of stock in a coal company may be satisfied to have a portion e par value of the stock represented by such possible future en- ement in value. When recent increase in value has occurred, surrounding conditions promise a continuance of the tendency, tions made for the guidance of buyers or sellers, executors and ees, etc., and similar purposes, can quite properly include a rea- ble allowance to represent such probable future increase.

*Quantity of Coal.*

is commonly believed that coal has such regularity in thickness quality over large areas that the quantity of workable coal can rately be estimated from data furnished by a few openings or ngs. This belief is not well founded. Throughout most of the fields of this country the coals are subject to variations in both kness and quality, and reliable estimates can be made only after area is thoroughly explored, either by shafts, pits, or open-cuts, oring, or by both. It does not follow that because a coal is of in thickness and quality, as shown by openings along the out-

crop on two sides of a hill, that the same conditions exist th
the intervening area. Disaster has overtaken many operati
upon such an assumption.

With the extension of mine workings to distances of fr
four miles from the outcrop, we are gaining knowledge of
tions both in thickness and quality that was not available
years ago, and systematic boring over large areas has in rec
also added largely to our knowledge of these matters.
know that coal beds are not usually of the same thickness a
over large areas.

In stating quantities, to distinguish between that which
and which is only partly known and that which is surmised
tice commonly used in valuing metalliferous deposits ma
iently be modified for use in coal-land appraisals, and the te
tive ore, probable ore, and possible ore, used in metal minin
replaced by the following terms, which are suggested as bei
understood by those not familiar with technical mining
Developed (workable coal) tonnage; partly developed (work
tonnage; and undeveloped (workable coal) tonnage.

To determine the average yield per acre, and the average
for use in making an estimate of value, we must either co
estimate absolutely to the proved territory, ignoring that
partly developed, and that which is undeveloped, or we m
some plan providing for the recognition of the factors wh
has not been fully ascertained, by including a portion of su
known values. This may be done by assigning to these ot
tities a certain weight to represent, as nearly as possible, the
of the engineer as to their relative value. As an example,
mates gave: Developed tonnage, 10,000,000 tons; partly
tonnage, 5,000,000 tons; undeveloped tonnage, 20,000,000
if we give weights of 1, $\frac{3}{5}$, and $\frac{1}{5}$, respectively, to these thr
ties, the tonnage on which the valuation estimate will be m

| | | Ton |
|---|---|---|
| Developed tonnage, . . . . . | $10,000,000 \times 1$ | 10,000, |
| Partly developed tonnage, . . . . | $5,000,000 \times \frac{3}{5}$ | 3,000, |
| Undeveloped tonnge, . . . . . | $20,000,000 \times \frac{1}{5}$ | 4,000, |
| Tonnage to be used for valuation, . . . . . | | 17,000, |

The same principle can be used in determining averag
and average thickness required for purposes of valuati
method is open to criticism on the ground that it does not gi
that are capable of being checked or proved, but it is not
tionable as methods which ignore important known facts.

can protect himself from adverse criticism for adopting such n by plainly stating the basis upon which the estimate is made supplementing the estimate by separate appraisal of the value e developed coal.

*Value as Affected by Ownership.*

the owner of a property be able to secure its rapid exhaustion, r by leasing or royalty, or by installing a mining plant of large city, the value of the property will be much greater than if its de- ment be delayed or its working prolonged over a long period. nding on the period required to take out all the coal, and assum- hat the coal is mined at a uniform rate, taking interest rate used omputing the value at 5 per cent., each $100 received in profit or ty has a present value, depending on the period covered by oper- of the property, about as follows:

| If Exhausted in | Present Value is About |
|---|---|
| 10 years, | $81.00 |
| 20 years, | 65.00 |
| 30 years, | 54.00 |
| 40 years, | 40.00 |
| 50 years, | 34.00 |

the working be deferred for 30 years and 10 years be then taken haust, the present value falls to about $15, the value declining ry small proportions if the time of working be postponed and ime required to take out all of the coal be extended over a long d.

*Cost of Plant and Improvements as Affecting Value.*

the estimates used to illustrate this subdivision of the subject, ost of miners' houses, store buildings, boarding houses, etc., is ded because the operator may not find it necessary to erect such ings and because the net revenue from them is sufficient to cover est on their cost after full deductions for maintenance, insurance, tization, etc., so that they do not constitute a charge against the ts of the operation—the investment being self-sustaining.

e cost of mining plant and equipment and its installation and the of development and improvement are often of greater importance the cost of the property upon which the operation is to be started. ng machinery and equipment and many other improvements be considered as having an average life of not more than 20 , and complete amortization of their cost is usually necessary in t that period, for even if a property has a life twice or three times ng, complete renewal, remodeling or rearrangement of the plant improvements must usually be made after an operation has

reached that age. Whether this renewal be gradual, or i
modeling to place the operation in competitive condition as
with other newer mines, is immaterial; the result is there i
in that plant of the sinking fund which was provided for its
tion. Provision must be made not only to cover interest o
of the plant, but to create and maintain a sinking fund,
interest and sinking fund to cover the cost of the land, l
operator can realize a profit. If the improvements and pla
much or more than the coal, the greatest concern of the op
be to select a property that can be cheaply equipped. Befo
ing this in detail it may be useful to consider the effect of
the coal, whether this be paid for as a royalty per ton or b
right purchase of the property.

*Profit and Surplus Profit.*

As in all competitive industries the profit is fixed by co
and in the present case is measured by the difference betwe
of mining and the net sale price. The value of any prop
rectly affected by conditions that govern the profits of prop
ducing a similar or competitive product.

If the earnings are not large enough to pay interest an
fund requirements of the capital investment in land an
is evident that the investment in land or in plant or in bo
large; that the plant cost more than the conditions justifi
the price paid for the land was too high, or that too muc
bought, or that the investment in both was too large.

The value of coal land to the owner is a sum upon
profit will pay interest and sinking-fund requirements after
interest and sinking-fund requirements of the investmen
and improvements. If the requirements of the latter are
large the coal-land value will be correspondingly decr
equation ($A$) the subdivision of the profit between land and pl
ment is provided by the coefficient d, the value of which
below may vary between wide limits.

Failure to realize the importance of the time necessary
a property and of cost of the plant, etc., is responsible
overestimates of the value of coal land. The following
introduced to illustrate the necessity for thorough study of
before an appraisal of value is attempted. Table I. show
cost of coal per ton depends not only upon the price pa
upon the quantity of land purchased, for land prices eq
prices of 1, 2, and 3 cents per ton of recoverable coal o

y, charging interest on the purchase price at 6 per cent. per n and with sinking fund to repay the purchase price when the rty is worked out calculated at an interest rate of 5 per cent.

E I.—*Relation of Price Paid and Time to Exhaust to Cost of Coal.*

| Required to ist Property. | Cost of Coal per Ton. | | |
|---|---|---|---|
| | Price Paid, 1 Cent. | Price Paid, 2 Cents. | Price Paid, 3 Cents. |
| | Cents. | Cents. | Cents. |
| 10 | 1.39 | 2.78 | 4.17 |
| 20 | 1.78 | 3.56 | 5.34 |
| 30 | 2.23 | 4.46 | 6.69 |
| 40 | 2.75 | 5.50 | 8.25 |
| 50 | 3.25 | 6.50 | 9.75 |
| 60 | 3.78 | 7.56 | 11.34 |
| 80 | 4.89 | 9.78 | 14.67 |
| 100 | 6.04 | 12.09 | 18.13 |

ose who are misled into purchasing at 3 cents per ton coal ves for 100 years by a belief that the coal is cheap at 3 cents on because they can make a profit of 10 or 12 cents per ton of mined and shipped, are merely saddling upon themselves a loss or 8 cents per ton. The only remedy for such a condition is ncrease in the capacity of the mining plant to mine the coal or 40 years, thus reducing its cost to about 8 cents per ton or less.

*Cost of Plant and Improvements.*

hen the coal to be worked is of good thickness, 5 ft. or more, arly level, with sufficient grade towards the outcrop for good age, is soft and easily cut, lies from 25 to 40 ft. above the railroad ing tracks, is in best demand as run-of-mines coal, is clean and res no preparation other than care in mining, the plant and ovements will be of the cheapest and most simple type and the tment in plant and mine development may not exceed 50 cents on of annual capacity, or, say, $50,000 to $60,000 for a capacity of 000 tons (240 days at 500 tons) annually.

the coal contains slate, requiring hand-picking, if its best market screened coal sold in several sizes, if it lies high above the rail- if it is hard and not easily cut, if it is not self-draining and is early level, the plant and improvements and development work require an investment of perhaps $1 per ton of annual capacity. difference is due to the extra cost of plane and equipment, to ional cost of shipping tracks, to cost of tipple with shaking ns and picking table, to larger power plant, more mining machines,

greater cost of driving entries and other mine developm to expense caused by drainage troubles, etc.

If the coal is relatively thin, requiring more entry worl cost per foot, if it lies below water level and must be opene or slope, if the slack and nut require jigging, if the propert distance from a railroad and a branch road must be built t the capital required may easily be $1.50 and in some cases $2 per ton of annual output.

Table II. shows the cost per ton chargeable to the inv plant and improvements for these three types of properti being taken at 6 per cent. and amortization calculated at In this table plant A costs $0.50, B, $1, and C, $2 per ton output.

TABLE II.—*Cost of Plant per Ton of Annual Outpu*

| Life of Plant. | Plant A. | Plant B. | Pla |
|---|---|---|---|
| Years. | Cents. | Cents. | C |
| 10 | 6.80 | 13.60 | |
| 20 | 4.44 | 8.88 | |
| 30 | 3.72 | 7.44 | |
| 40 | 3.43 | 6.87 | |
| 50 | 3.27 | 6.55 | |

For reasons already given the life of a plant should not b at more than about 20 years and the cost of plant per ton of therefore be taken as 4.44, 8.88, and 17.76 cents for the three t described. If these differences in cost of plant are unavoida by differences in the coal, then they represent actual differe value of the coal per ton, for coal from one property will much less per ton to produce and will therefore be wor ground) 4.44 or 13.32 cents per ton more than that of properties, the differences in value between these two prope 8.88. The differences in value in the case of a coal bed 5 yielding 5,000 tons per acre, to be worked out in 20 yea respectively to about $145, $435, and $290 per acre. We an illustration of one reason for the great disparity in va shown by actual sales of properties which on casual inspect to be of nearly equal value. Even this, however, does no emplify differences in value caused by varying local an conditions and trade requirements, for it is almost always coals that call for expensive equipment cost more to mine that can be cheaply opened. The cheap plant may be just as the more costly installation, the higher cost of operat property costing more to equip not being appreciably red higher efficiency of the more expensive plant.

*Ability to Meet Competition.*

r combining the figures of Tables I. and II. we may obtain the est and sinking-fund requirements covering the cost of both land plant, these two items including the whole investment excepting vorking capital to meet payrolls, current bills, etc. Table III. s such combined charges based on amortization of plant in 20 irrespective of the life of the property.

'ABLE III.—*Interest and Sinking Fund Requirements (Includes cost of land, plant and improvements.)*

| | Price of Land 1 Cent per Ton. | | | Price of Land 2 Cents per Ton. | | | Price of Land 3 Cents per Ton. | | |
|---|---|---|---|---|---|---|---|---|---|
| | Plant A. | Plant B. | Plant C. | Plant A. | Plant B. | Plant C. | Plant A. | Plant B. | Plant C. |
| | Cents. | Cents. | Cents. | Cents. | Cents. | Cents. | Cents. | Cents. | Cents. |
| | 8.19 | 14.99 | 28.59 | 9.56 | 16.36 | 29.96 | 10.97 | 17.77 | 31.37 |
| | 6.33 | 10.68 | 19.58 | 8.01 | 12.46 | 21.36 | 9.79 | 14.24 | 23.14 |
| | 6.68 | 11.13 | 20.08 | 8.91 | 13.36 | 22.26 | 11.14 | 15.49 | 24.49 |
| | 7.20 | 11.65 | 20.55 | 9.95 | 14.40 | 23.30 | 12.70 | 17.15 | 26.05 |
| | 7.70 | 12.15 | 21.05 | 10.95 | 15.40 | 24.30 | 14.20 | 18.65 | 27.55 |

n operation financed on capital borrowed at 6 per cent. must e net earnings larger than these requirements if there is to be surplus available as profit to compensate the operator for the assumed, for his own time and for the use of the funds em- ed as working capital.

his table shows how an operator may be handicapped by the hase of a large acreage of relatively high-priced land requiring nore expensive types of plant, and how almost impossible it will or him to meet the competition of those who have bought a ler acreage, or cheaper land, or land that can be developed by a costly plant.

he higher carrying charges shown by this table are in excess of average profits of the business, and operations so burdened can ive only when the product commands prices correspondingly er than those obtained for the competing product or when the does not meet that of the less heavily burdened operations in market competition. Coals of especially high grade, command- premiums over those of ordinary grade, such as those especially ted for domestic use, or for coke making, particularly for by- uct coke, for use in gas producers, etc., can be made the basis of cessful operation even when the carrying charges equal or exceed gures of this table, thus justifying valuations of 2, 3, or more cents on, or $100, $200 or more per acre for lands containing 5,000 verable tons per acre. Coals of average or inferior quality and

thickness, under present conditions, rarely warrant valuatior as $100 per acre. In most cases the value of such coal is and is often less, than $50 per acre.

On the other hand, coal of especially high grade, acc large markets, in thick beds, regular and persistent in both and quality, to be quickly worked, may readily be worth more per acre. Small areas of workable coal even if of p ity, when close to markets and far distant from better coal, surprisingly large value.

*Value of Leaseholds.*

As the lessee virtually is the owner of the coal, subject t ment of the royalty, all increase in the value of the coal w to his benefit. In many cases the value of the lessee's inter property exceeds that of the owner of the property. The lease may be determined by the profit that can be made d term of the lease, or by determining the value of the pro whole and deducting from this the value of the royalty Leaseholds can very properly be made the basis for a stock or bonds and when so used there is no reason should not be appraised by the same methods used in fixing of lands held in fee.

*Use of Formula.*

Modifying formula (*A*) by the variables illustrated by Tab by formula (*D*), the value per acre may be expressed thus:

$$V = crT\left(ASP\frac{B.t.u}{13,000} - CM - f - 0.088\,n\right),$$

in which V = Value of land in dollars per acre.

c = coefficient depending upon the time to exhaust the property and u have a value of between 0.1 and

r = recovery in tons per acre for eac thickness.

T = thickness of coal in feet.

ASP = Average selling price of coal of 13, value in markets where the co sold.

B.t.u. = heat value of the coal.

CM = Cost of mining.

f = freight.

n = cost of plant and development for of annual capacity based on 6 interest and amortization in 20 5 per cent.

ɔly future study of the subject may make it possible to derive la to express the effect of variations in thickness, depth, and ıysical conditions and local conditions upon the cost of min- he utility of such formula, however, is found principally in ng study of the subject, rather than in its actual use in com- esults.

: object of an appraisal is to reach a valuation representing value of any property, no fact is trivial or unimportant that bearing whatever upon the subject. Few realize the difficul- intricacies of the issues involved in each such valuation. The f the writer will have been attained if he has succeeded in how some of the problems can be studied and a better un- ling gained of the real issues involved in determining coal- ues.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.
[SUBJECT TO REVISION.]

USSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the eting, Aug. 18 to 21, 1913, when an abstract of the paper will be read. If this is impossible, ussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 Street, New York, N. Y., for presentation by the Secretary or other representative of its Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1913, when I. of the *Transactions* will go to press. Any discussion offered thereafter should preferably be rm of a new paper for publication in Vol. XLVII. (with suitable cross references in both

# Thermal Effect of Blast-Furnace Jackets.

BY ROBERT P. ROBERTS, GREAT FALLS, MONT.

(Butte Meeting, August, 1913.)

rder to obtain data on the thermal effect of the blast-furnace and on the water consumption in these jackets a series of tests un on the 56 by 180 in. blast furnaces at the Great Falls tion Department of the Anaconda Copper Mining Co., at Great Mont.

arrangement of the jacket is shown in the transverse section of the furnaces (Fig. 1) on which the tests were made. The spout, and breast jackets have independent water supplies and rges. The end and side jackets are connected in pairs, as re- water supply, the feed water entering at the bottom of the lower and overflowing into the bottom of the jacket immediately it, and discharging into an independent overflow pipe for each jackets.

s were run on the main side and end jackets, the breast jacket, out, and the nose. Measurements were taken of the quantity er discharged per minute and the rise in temperature of the in its passage through the jacket. The product of these two , expressed in the proper units, evidently gives the number of f heat removed per minute by the water. Sufficient work was n each type of jacket to determine its behavior under different ions, and the results from each set of conditions were checked eating the work from three to six times.

method used was as follows: The overflow from the jacket tested was arranged to discharge at will into a barrel, ed upon platform scales. The discharge temperature was read efore turning the water into the barrel, and just before turning to the sluice; if the test lasted more than a minute, additional rature readings were taken at 1-min. intervals from start to The elapsed time of running the water into the barrel was ined with a stop-watch, and this figure, combined with the in-

crease in weight of the barrel, gave the pounds of water di
per minute. The initial temperature of the water was measu
a thermometer immersed in the supply tank on the feed flo

For the breast jacket, spout, and nose, a length of flex
was attached to the end of the overflow pipe, by which means
could be turned into the barrel quickly and easily. For the
end jackets a more elaborate apparatus was necessary, on a
the large size of the overflow pipes. Iron piping was use
purpose, with combinations of elbows and loose joints, so

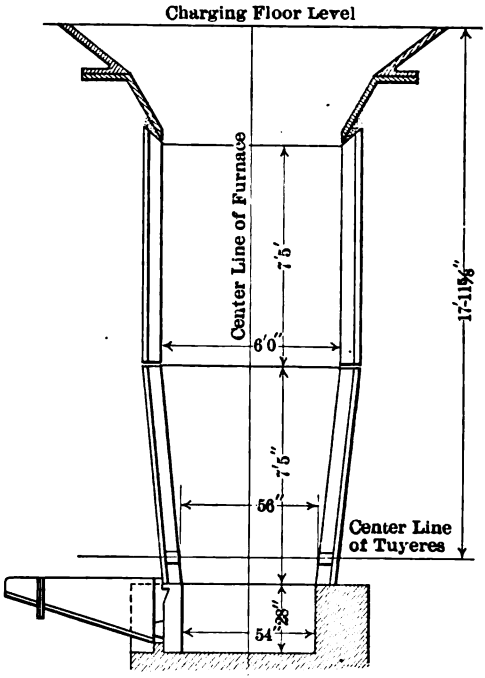



FIG. 1.—TRANSVERSE VERTICAL SECTION OF 56 BY 180 IN. BLAST FU

that the lower half length of the pipe could be swung ho
from the sluice to the barrel, and back. Both piping and b
pierced, near their junctions with the joint overflows, to allo
troduction of the thermometer for measuring the discharge
ture. The conditions were varied by changing the consu
water, so that the period of time covered by one test varied
ably, being less than a minute in the case of the main side
jackets with valves wide open, and as much as 19 min. for t
jacket with the water reduced as far as safety allowed.

following data will show the precision with which the work e: Capacity of platform scales, 2,500 lb., reading to 0.5 lb.; ales were checked with standard weights before beginning the found to be correct. The stop-watch read to 0.2 sec. Three neters were used altogether, one of them being broken in the f the work. To explain the corrections applied to the tem- readings in the tabulations of results, the following descrip- iven:

nometer A: Centigrade; graduated to 200° C. and reading

nometer B: Fahrenheit; graduated to 600° F. and reading

nometer C: Centigrade; graduated to 200° C. and reading

nometers A and B checked exactly, for the range of temper- equired in the test. Thermometer A read 0.3° C. lower than neter C. In all the tests on the nose and the breast jacket, neter A was used at the supply tank, and B was used to read harge temperature. No correction is made in the tempera- any of these tests. Thermometer B was broken just after the sixth test on the spout; previous to this, the tests on the ere run with the same arrangement of thermometers as in the the nose and the breast jacket. The seventh, eighth, and sts of the spout were run with one thermometer, A doing h at the supply tank and the jacket discharge, since the tem- at the tank was found to remain practically constant. No on is therefore necessary on any of the spout readings. All on the main end jackets and the first 12 tests on the main kets were run with thermometer A at the tank and C at the ischarge. The discharge readings for these tests are there- rected by reducing them 0.3° C., to obtain the net rise in ture of the water in the jackets. The last 36 tests on the de jackets were run with the thermometer C at the tank and jacket discharge. The discharge readings are therefore in- 0.3° C. in these tests. The amount of the correction is so at it is of no importance whether the initial or the discharge ture is the one to be corrected; in either case, the difference, se, in temperature is not affected.

case of the main end and side jackets, each test was run on two jackets, one upper and one lower, since there is an over- each set, but not for each jacket.

tabulations attached to the present report notes have been

added of the less important conditions affecting the results, s dumping of charges, changes in the amount and temperature etc. However, a more detailed description of conditions is ne in order to fully explain the results; especially as it was fou these outside factors sometimes caused conflicting readings.

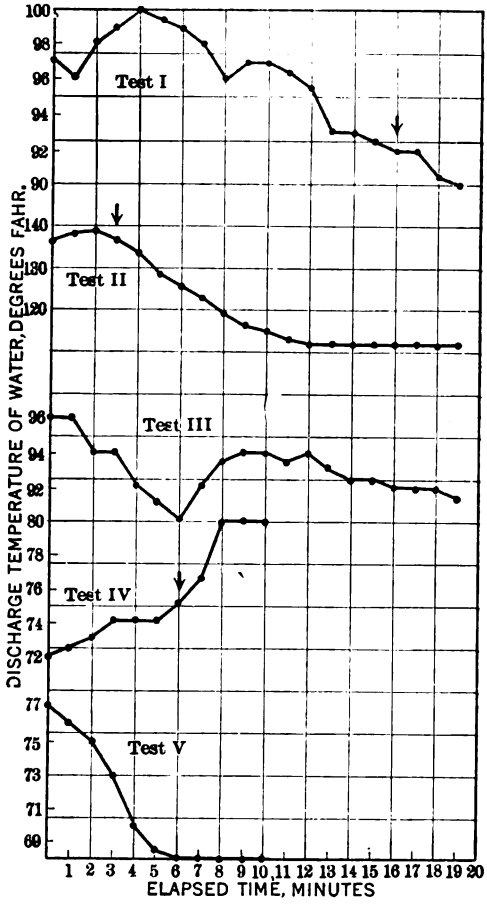
In the report on the tests of Nov. 3, 1910, a paragraph from *Steam Boiler Economy* was quoted to show that the rate of hea mission of a clean metal plate, with a fire on one side and w the other, depends on the difference of temperature on the tw of the plate. Also, that this relation is disturbed by any ac tion on the plate of material which is a poor conductor of he steam boilers, such an accumulation is furnished by scale on th or water, side; while in a furnace jacket it is represented b on the outer or fire side.

The thickness of the crust in the present test was found t the heat transmission to a considerable extent; although the temperature of the furnace, and the discharge temperature water, also had their effect. Usually, the readings were inf by more than one of these factors, and different types of jacket the effect of each factor in different degree; so that the resu be understood only when all the test conditions are known.

*Nose.*—Beginning with the simplest case, that of the n source of heat is a stream of liquid slag at a temperature not above its freezing point. In consequence, crusts are rapidly and are frequently barred out, so that the conducting powe jacket varies from that of clean metal to that of a slag crust inch thick. Slag being a poor heat conductor, compared w metals, we can expect a good deal of variation in the number units carried away by the water.

In the first test, a moderate amount of crust was present be readings began, and this slowly increased while they wer taken. It will be seen from the tabulation that after the minute the temperature readings gradually dropped from 10 90° F. In the second test, all the crust was removed just p to taking the first reading, but no barring was done after began. The temperature of the water rose for 2 min. a decreased for 10 min., finally becoming constant as the crust a thickness so great that the heat transmission became too s chill the slag below its freezing point. The remaining fo were run with a normal accumulation of crust, and it is evide with one exception, the temperature of discharge gradually de while the readings were being taken. The exception occurs

test, when a large amount of matte in the stream, and a large e of slag, due to dumping fresh charges in the furnace, caused g temperature. Charges were also dumped during the second nd may have retarded the downward trend of the temperature ngs for a minute or two, without altogether preventing it. The s are shown in Fig. 2.




Arrows indicate time of dumping charges.

. 2.—Nose. Variation in Temperature of Jacket Discharge During Tests I. to V., Inclusive.

e above discussion of temperature applies equally to the heat removed by the water per minute, because the measurements of scharge show that it remained practically constant unless altered justing the valves. Changes in the thickness of the crust were

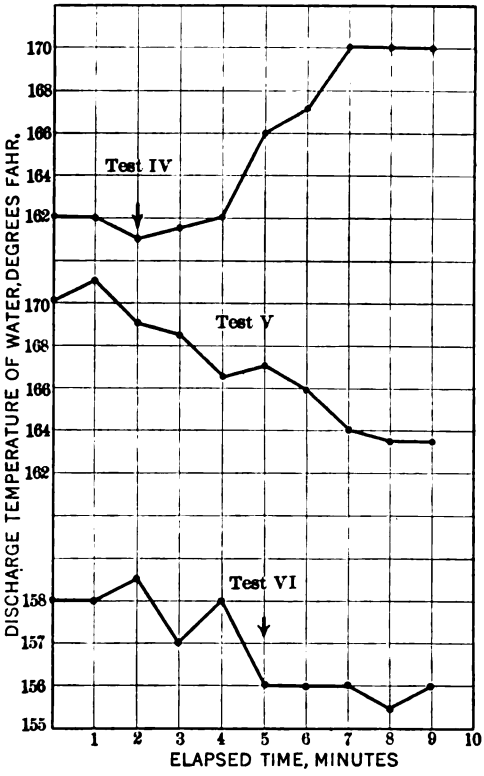
so rapid that they affected each individual test, and consequen series, a series being a group of tests in which the flow water was kept constant. Now, comparing individual tests, and sixth tests were run with a reduced volume of slag flowi the furnace, and the average discharge temperature in these t is lower than in the fourth test, which was run under the san tions as to flow of water. In all three of these tests the an crust is recorded as normal, and it seems probable that tl temperature of discharge in Tests V. and VI. is due to the amount of slag. That this should be expected may be seen formula for the rate of heat transmission given by Kent, in w temperature on the "fire" side of the transmitting plate is o factors. An increase in the volume of the slag stream greater surface of the jacket with hot slag, and thus ra average temperature over the whole surface of the "fire" si

Finally, comparing series to series, we find that the three t a smaller volume, and therefore a higher temperature, of wat a smaller heat loss, or smaller rate of heat transmission. Tl be expected from the other factor in Kent's formula, whic temperature on the "water" side of the transmitting plate. three factors—thickness of crust and temperature on the two the plate (or jacket)—have influenced these results, the thic crust being apparently the most important.

*Spout.*—The crust in the spout is rarely entirely removed in case of a shut-down or of a freeze-up. Usually it is left up or cut out, according to the temperature of the slag strea comes from the furnace. Whence the influence of the crust rate of heat transmission should be almost constant. Furtl since the crust is so thick, it should tend to mask the effec other factors, since its own influence is correspondingly im Of these other factors, the temperature on the "fire" side of tl should have greater effect than the temperature of the water is subject to greater variation. Just as in the case of the n temperature of the "fire" side varies more with the volume tl the actual temperature of the slag, since an increase in volum a large increase in the area of jacket covered by hot slag, rai average temperature of the "fire" side more quickly than a grees difference in actual slag temperature.

These conclusions are all justified by the results of th Within narrow limits, the quantity of heat removed per mi the rate of heat transmission, remained constant, showing th influence of the crust in retarding and fixing the rate of he

and in obscuring the effects of other factors. This is still hown by the average of the series, tabulated in the summary. ever, the discharge temperature readings show that the tem- e of the "fire" side also had its effect, and these conditions ap- ally well to the rate of heat transmission, since the volume of n each series remained very nearly constant. Charges were l into the furnace just before starting the first test, and caused al rush of slag from the furnace, the stream then falling off ne readings were being taken. The readings decrease rapidly




Arrows indicate time of dumping charges.

.—SPOUT. VARIATION IN TEMPERATURE OF JACKET DISCHARGE DURING TESTS IV. TO VI., INCLUSIVE.

w minutes, and never return to the high temperature shown irst reading. In the third test, charges were dumped soon e start, and the temperature rose rapidly from that time. also the case with Test IV., as may be seen from the curve Fig. 3. Charges were dumped again just as the last test was and the temperature again rose. In Test VI. the slag stream y low towards the end, and the temperature gradually de-

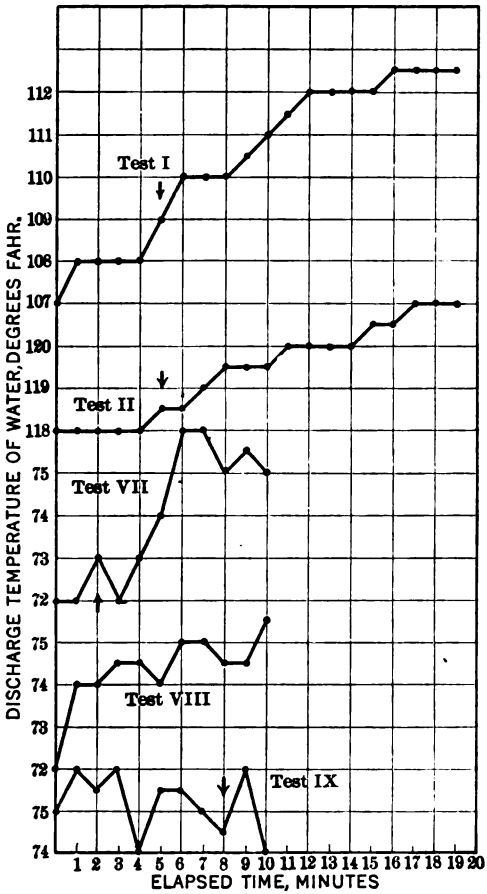
The results do not show the effect of the water temperatu first series, at medium volume and temperature of dischar the lowest heat loss, while the second series, with the smalle and highest temperature of discharge, gives a result interm tween the other two. The thickness of crust is again the portant factor, with the temperature of the "fire" side seco

*Breast Jacket.* Although the breast jacket is part of the the furnace, it usually differs from the main side and end j being less subject to crusting. This is due to the constan hot product past the breast jacket and out into the spout, a the omission of the two center tuyeres on the front of the which prevents any chilling from cold air at that point. The of crusting should therefore be less than with the nose and

The results vary considerably, indicating that crusting a of minor importance. The first two series were run in the when the slag stream was medium in both volume and tem The third series was run in the afternoon, and by that time was hotter and flowing in greater quantity. Thus the c could not be kept uniform, and the effect of the water ter is not properly shown by the results. However, the first tw Tests I. to VI., inclusive, were run under nearly the same c except as regards volume of water, and it will be seen that water carried away less heat than the colder, indicating the the water temperature on the rate of heat transmission wl factors remain constant.

The highest heat loss is shown by the third series, when perature on the "fire" side of the jacket rose high enough the rise in water temperature. Considering water temperat the third series should give results intermediate between two, and the fact that it gives the highest results of all s greater importance of the fire temperature. This is also the individual temperature readings. Although the bre forms a part of the furnace wall, it also resembles the spout in being in contact chiefly with the molten products, rather unsmelted or partly smelted charge materials. Therefore expect to find that the effect of dumping charges is to rais temperature, by causing a larger flow of those products bringing a greater number of heat units in contact with the a given time. Tests I., II., and VII. indicate that this is tr be seen from the curves, Fig. 4. Charges were also dumpe III., VI., and IX., but without producing any effect on the tures. Of the remaining tests, Nos. IV. and V. show almos

atures, and No. VIII. a small rise after the first minute. No
were dumped during these tests.
controlling factor seems to be the furnace and slag tempera-
ith the water temperature of next importance.




III. gives a straight line at constant temperature and is therefore omitted.
Arrows indicate time of dumping charges.

—BREAST JACKET. VARIATION IN TEMPERATURE OF JACKET DISCHARGE DURING TESTS I., II., VII., VIII., AND IX.

*End Jackets.*—The results on the main end jackets can be
ed with similar data given in the report on the test of Nov.
, in which the same method of work was followed and the
ind of conditions prevailed. Compared in this way, the pres-
ults are much lower than those of the previous test, though
se enough to serve as a general check.

Crust is nearly always present on the end jackets, even t may not appear at the surface of the charge column, and greatly in thickness. A loose, porous crust may allow co from below to cool the upper portions of the jackets, while hard crust protects the jacket by poor conduction. The u part of the jackets, above the charge column, may be ex higher temperatures than the portions covered by charge a or less crusted.

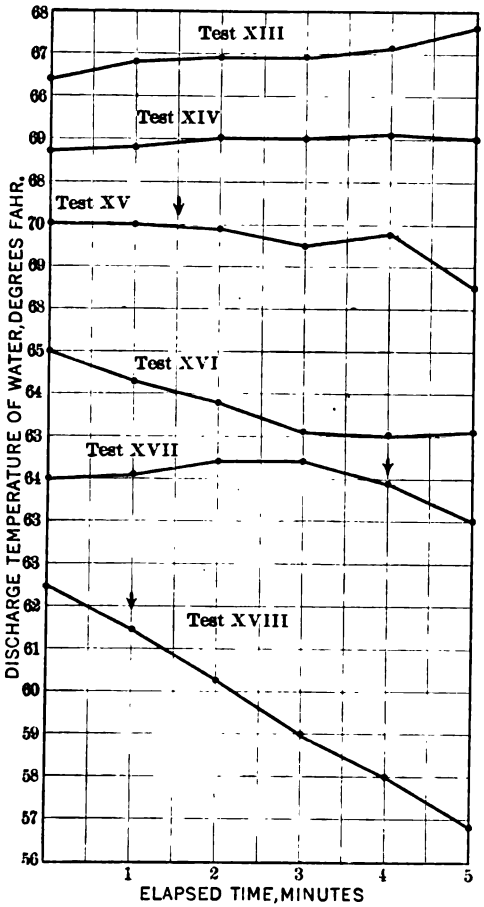
The effect of the crusts in the present test is evident in th constant value of the heat loss. The transmission of heat have been retarded to such an extent that the effect of water ture was of no importance. However, the fire (or furnace) ten had greater influence, as may be seen from the last three test were run at a time when the furnace was receiving charges succession, and being cooled down in the upper part of th column as a consequence. The heat loss in these three te lowest of any, although all the other conditions remained a same. Omitting these tests, the average B.t.u. removed per by the jacket during the last series (Tests XIII. to XVIII., i amounts to 9,583, which checks more closely with the 9 10,028 B.t.u. per minute of the other series. An inspecti individual temperature readings shows the effect of dump material against the jackets, and confirms the importance o nace temperature, for after each charge the water temperature rapidly. This is well shown by the curves, Fig. 5, in which temperature is plotted against the regular time interva readings.

*Main Side Jackets.*—The side and end jackets present ve the same conditions, the only difference being that the sides apt to be affected by blast spilling along the jackets, since th are placed on the sides. Otherwise, the influence of crust charge materials, and of general furnace temperature, shou same for both types of jacket.

Tests were run on two different furnaces, Nos. 2 and heat loss in No. 2 varied from 2,762 to 16,752 B.t.u. ute, and in No. 4 from 1,918 to 6,302 B.t.u. per minu average of the previous tests, November, 1910, gives a loss B.t.u. per minute, the furnace being No. 4. These result widely that some explanation of them must be looked for in dition of the furnace.

The first 12 tests of the present work were run on 1910. Tests I. to VI. were run between 10:00 and 10:45

udden drop in temperature in the last of the series, although me of water varied less than 1 per cent. through all the six Tests VII. to XII. were run between 1:00 and 1:30 p.m. and udden rise towards the end, followed by a smaller drop. The s in Test XI. is about 23 per cent. greater than that in Test




Arrows indicate time of dumping charges.

5.—Main End Jackets. Variation in Temperature of Jacket Discharge During Tests XIII. to XVIII., Inclusive.

e entirely to differences in temperature, since the volumes of eck almost exactly.

act that these changes in water temperature came near the he series in each case shows that they were not due to start- tests too soon after changing the volume of water, or before

the new temperature had become constant. Trouble of was encountered before, but in that case the temperatures v ing each separate test, and gradually became constant as t tions adjusted themselves. The reverse is evidently true in ent results, and the variations can have been caused only by conditions within the furnace.

Tests XIII. to XXX., inclusive, were run on Dec. 14 Furnace No. 2. The first six of these give a heat loss c with that found on Nov. 3, although much lower. They between 10:00 a.m. and noon, with a small amount of crus at the surface of the column. Tests XIX. and XXIV. we tween 1:00 and 3:00 p. m., and show only about 40 per c heat loss found in the first six, the volume of water being same. By this time a heavy, loose crust had developed, an nace was taking its blast poorly. The last six tests, XXV. were run with a larger volume of water, and gave a heat l ing fairly well with the second six. Finally, all the wate was turned into the jacket, but the discharge temperature almost unchanged, and the work was transferred to Furn

The seventh, eighth, and ninth series, Tests XXXI. to were run on Dec. 15, 1910, on Furnace No. 4. The wat duced on the northeast side jackets at 9:10 a.m. and te readings were then taken every 5 min. until no further ri served. These readings were as follows:

| | |
|---|---|
| 9:10 a.m., water reduced. | 10:10 a.m., 36° C. |
| 9:25 a.m., 16° C. | 10:15 a.m., 33° C. |
| 9:30 a.m., 19° C. | 10:20 a.m., 33° C. |
| 9:33 a.m., 22° C. | 10:25 a.m., 35° C. |
| 9:35 a.m., 25° C. | 10:30 a.m., 36° C. |
| 9:40 a.m., 27° C. | 10:35 a.m., 37° C. |
| 9:45 a.m., 28° C. | 10:40 a.m., 34° C. |
| 9:50 a.m., 31° C. | 10:45 a.m., 35° C. |
| 9:55 a.m., 33° C. | 10:50 a.m., 36° C. |
| 10:00 a.m., 37° C. | 10:55 a.m., 36° C. |
| 10:05 a.m., 38° C. | 11:00 a.m., 36° C. |

These figures show that care was taken to obtain a con perature, since from 10:00 a.m. to 11:00 the readings w uniform. The tests were then started, but during Test XX temperature began to rise again, with the result that one o tests gave a heat loss nearly 30 per cent. higher than th XXX. The last two series gave fairly uniform results, also the fourth and fifth series.

In reviewing all the results obtained on the side jacket

be assigned to the heat loss. This is due to the variations in temperature, occurring while the volume of water remained ically constant, and apparently following no law except that they influenced by the interior conditions of the furnace. In one he temperature of the water remained constant while the volume doubled, and in many cases it varied over a wide range while olume remained constant. There can be only one explanation e changes: namely, the furnace temperature and the effect of s. Evidently the relation between heat loss and water tempera- s much less important than the effect of these other factors, as e seen in the summary. The greatest heat loss occurred in the series, in which the water temperature was highest, while the t heat loss occurred in the sixth series, which had the second st water temperature. The series that furnish the best check he fourth, fifth, seventh, and eighth, with an average heat loss 736 B.t.u. per minute. The water temperature in the eighth is increased by about 300 per cent. in the fourth series, though .t.u. lost per minute check to within 7 per cent. This resem- he constant value of the heat loss found in the end jackets, and th end and side jackets on Nov. 3. The remaining four series the widest variation in heat loss, without any relation between nd the water temperature.

e curves, Fig. 6, show this temperature variation graphically llustrate the regularity of the changes in certain tests, suggest- hat a gradual change was going on at the same time within the ce.

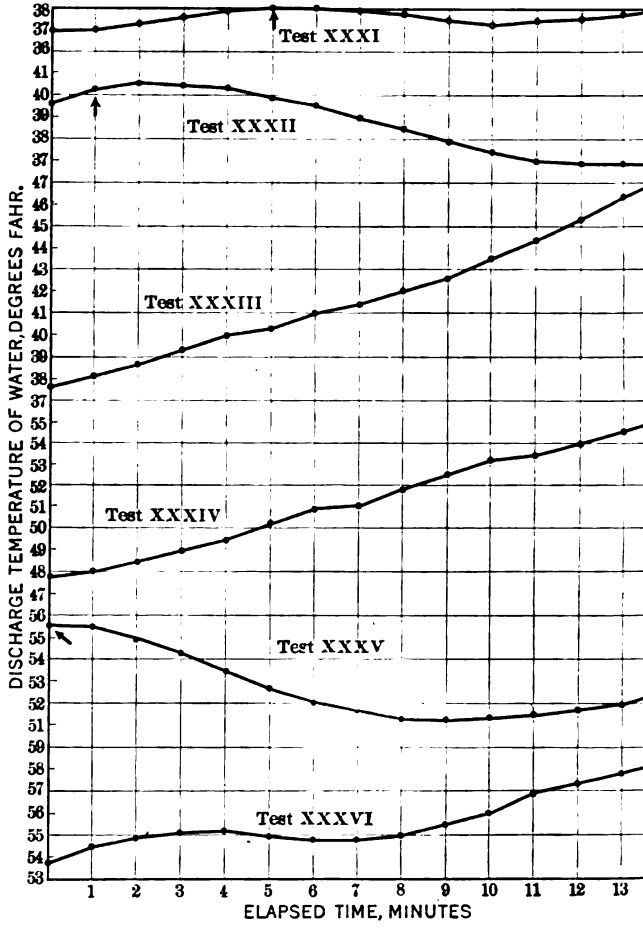
*Effect of the Length of Furnace on Heat Loss Through Jackets.*

abulation and a curve, Fig. 7, have been added, showing the per- ge of area in the end jackets for different lengths of furnace. percentage decreases as the furnace is lengthened, and the e feet of radiating surface (jacket area) per square foot of hearth hows a decrease, though less in amount. A study of the curve that the greater part of the decrease is produced by lengthen- he furnace from 15 to 60 ft. Beyond 60 ft. the saving in jacket is very small.

e lengths of furnace at the Anaconda plant are 87 ft. and 51 ft., being one of the former length and two of the latter. Calcu- the proportion of end jacket area in the total, we find that e 87-ft. furnace the end jackets furnish 6.1 per cent. of the e area, and in the 51-ft. furnace, 9.9 per cent. The ratios of t area to hearth area in the furnaces are 6.78 and 7.07, respect-

ively. These figures show that the 9 ft. of length added to furnace to make the 60 ft. furnace produce a relatively grea than the 27 ft. of difference between the 60-ft. and the 87-ft.

If reasonable figures can be deduced for the heat transr side and end jackets, from the tests now reported, these ca

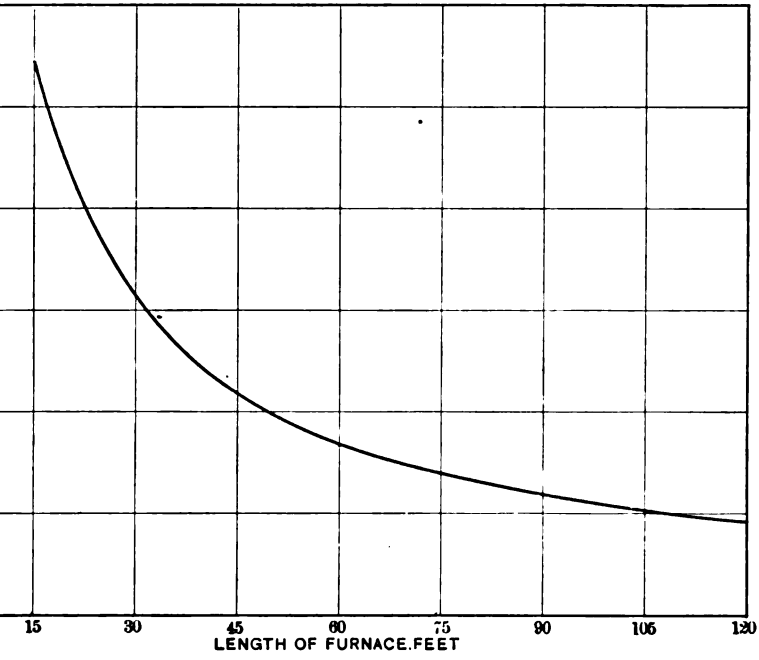



Arrows indicate time of dumping charges.

FIG. 6.—MAIN SIDE JACKETS. VARIATION IN TEMPERATURE OF J DISCHARGE DURING TESTS XXXI. TO XXXVI., INCLUSIVE.

of jacket area and saving in end jackets can be expressed i heat loss. A tabulation is presented, in which the most s results are averaged for the side jackets and all results ar for the end jackets, taking into account the work of No For this purpose, the side jacket tests of Dec. 3 are omitt sults are all very low, and the furnace was in such cond

had to be discontinued before all the desired conditions tested. Omitting these, we find that the average heat o side jackets (upper and lower) was about 8,400 B.t.u. per and of eight side jackets, 33,600 B.t.u per minute. This the unit in all subsequent calculations, as it represents the of all side jackets in the unit or 15-ft. furnace. For the end e find an average of 11,500 B.t.u. per minute with two nd 23,000 B.t.u. per minute with four jackets, which repre- full end-jacket area of any length of furnace.




DATA.

Section: Width between upper jackets, 6 ft.
Width at tuyeres, 4 ft. 8 in.
er Jackets: End, 7 ft. 5 in. by 6 ft. 6 in.
Side, 7 ft. 5 in. by 7 ft. 6 in.
er Jackets: End, 7 ft. 5 in. high, 6 ft. 6 in. wide at top, 5 ft. wide at bottom.
Side, 7 ft. 5 in. vertical height, 7 ft. 5.45 in. slant height, 7 ft. 6 in. wide.
End jackets overlap side jackets 3 in. on each side.

RVE SHOWING PER CENT. OF TOTAL BLAST-FURNACE JACKET AREA CON-NED IN THE END JACKETS, FOR VARIOUS LENGTHS OF FURNACE.

cond part of the tabulation shows the heat loss of various ranging in length from 15 to 120 ft., and including the two a types. Taking the heating value of blast-furnace coke for

December, 1910 (not ash-plant coke) as representing the average of our coke, these figures are converted into pounds of coke per minute. A column is also added showing the pounds of coke per minute equivalent to the heat loss of a number of 15-ft. furnaces whose total hearth area would equal the hearth area of the long, or combined, type. The latter figures are evidently greater than the actual coke equivalent of the heat loss for the combined furnaces and the ratio between the two may be expressed in percentage. This is done in the last column.

*Test on the Thermal Effect of Blast-Furnace Jackets.*

*Nose.*

Blast Furnace No. 2. Nov. 29, 1910.

| Test No. | Observed Data. Net Wt. of Water. Pounds. | Initial Temp. of Water. °C. | Elapsed Time. Min.-Sec. | Calculated Data. Average Disch'g Temp. °F. | Rise in Temp. °F. | Pounds Water Per Minute. | Gallons Water Per Minute. | B.t.u. Removed Per Minute. |
|---|---|---|---|---|---|---|---|---|
| I. | 175.5 | 0.0 | 19–3.2 | 95.6 | 63.6 | 9.21 | 1.10 | 585.8 |
| II. | 166.0 | 0.0 | 19–0.8 | 120.8 | 88.8 | 8.73 | 1.05 | 775.2 |
| III. | 148.5 | 0.0 | 19–1.6 | 93.0 | 6[illegible].0 | 7.80 | 0.94 | 475.8 |
| IV. | 237.5 | 0.0 | 10–1.2 | 75.5 | 43.5 | 23.70 | 2.84 | 1031.0 |
| V. | 232.5 | 0.0 | 9–56.4 | 70.9 | 38.9 | 23.39 | 2.80 | 909.9 |
| VI. | 138.5 | 0.0 | 5–55.0 | 68.3 | 36.3 | 23.41 | 2.81 | 849.8 |

Detailed Temperature Readings and Notes.

| Test I. Elapsed Time. | Test I. Disch'g Temp. | Test II. Elapsed Time. | Test II. Disch'g Temp. | Test III. Elapsed Time. | Test III. Disch'g Temp. | Test IV. Elapsed Time. | Test IV. Disch'g Temp. | Test V. Elapsed Time. | Test V. Disch'g Temp. | Test VI. Elapsed Time. | Test VI. Disch'g Temp. |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Min. Sec. | °F | | °F | | °F | | °F | | °F | | °F |
| 0–0.0 | 97.0 | 0–0.0 | 136.0 | 0–0.0 | 96.0 | 0–0.0 | 72.0 | 0–0.0 | 77.0 | 0–0.0 | 69.5 |
| 1–0.0 | 96.0 | 1–0.0 | 137.5 | 1–0.0 | 96.0 | 1–0.0 | 72.5 | 1–0.0 | 76.0 | 1–0.0 | 68.5 |
| 2–0.0 | 98.0 | 2–0.0 | 138.0 | 2–0.0 | 94.0 | 2–0.0 | 73.0 | 2–0.0 | 75.0 | 2–0.0 | 68.0 |
| 3–0.0 | 99.0 | 3–0.0 | 136.0*a* | 3–0.0 | 94.0 | 3–0.0 | 74.0 | 3–0.0 | 73.0 | 3–0.0 | 68.5 |
| 4–0.0 | 100.0 | 4–0.0 | 133.0 | 4–0.0 | 92.0 | 4–0.0 | 74.0 | 4–0.0 | 70.0 | 4–0.0 | 68.5 |
| 5–0.0 | 99.5 | 5–0.0 | 128.0 | 5–0.0 | 91.0 | 5–0.0 | 74.0 | 5–0.0 | 68.5 | 5–0.0 | 67.0 |
| 6–0.0 | 99.0 | 6–0.0 | 125.0 | 6–0.0 | 90.0 | 6–0.0 | 75.0*a* | 6–0.0 | 68.0 | Normal accumulation of crust. Amount of slag reduced as in Test V. | |
| 7–0.0 | 98.0 | 7–0.0 | 122.5 | 7–0.0 | 92.0 | 7–0.0 | 76.5*b* | 7–0.0 | 68.0 | | |
| 8–0.0 | 96.0 | 8–0.0 | 119.0 | 8–0.0 | 93.5 | 8–0.0 | 80.0 | 8–0.0 | 68.0 | | |
| 9–0.0 | 97.0 | 9–0.0 | 116.0 | 9–0.0 | 94.0 | 9–0.0 | 80.0 | 9–0.0 | 68.0 | | |
| 10–0.0 | 97.0 | 10–0.0 | 115.0 | 10–0.0 | 94.0 | 10–1.2 | 80.0 | 9–56.4 | 68.0 | | |
| 11–0.0 | 96.5 | 11–0.0 | 113.0 | 11–0.0 | 93.5 | Normal accumulation of crust. | | Normal accumulation of crust. Quantity of slag falling off. | | | |
| 12–0.0 | 95.5 | 12–0.0 | 112.0 | 12–0.0 | 94.0 | | | | | | |
| 13–0.0 | 93.0 | 13–0.0 | 112.0 | 13–0.0 | 93.0 | | | | | | |
| 14–0.0 | 93.0 | 14–0.0 | 112.0 | 14–0.0 | 92.5 | | | | | | |
| 15–0.0 | 92.5 | 15–0.0 | 112.0 | 15–0.0 | 92.5 | | | | | | |
| 16–0.0 | 92.0*a* | 16–0.0 | 112.0 | 16–0.0 | 92.0 | | | | | | |
| 17–0.0 | 92.0 | 17–0.0 | 112.0 | 17–0.0 | 92.0 | | | | | | |
| 18–0.0 | 90.5 | 18–0.0 | 112.0 | 18–0.0 | 92.0 | | | | | | |
| 19–3.2 | 90.0 | 19–0.8 | 112.0 | 19–1.6 | 91.5 | | | | | | |
| Considerable crust formed—none removed. | | Crust removed just previous to test—then left untouched. | | Normal accumulation of crust. | | | | | | | |

*a.* Charges dumped.

*b.* Large amount of matte in stream.

f we use the term "fuel loss" to mean the pounds of coke re-
red per minute to supply the heat loss through the jackets, it is
lent that the fuel loss decreases rapidly between lengths of 15 and
t., and very little beyond 60 ft. The 60-ft. furnace shows a fuel
equal to 69.6 per cent. of the fuel loss in four 15-ft. furnaces,
le the 120-ft. furnace shows a fuel loss of 64.5 per cent. of that in
nt 15-ft. furnaces. Even doubling the length of our present fur-
es would reduce the fuel loss about 20 per cent., and increasing it
60 ft. would reduce it another 10 per cent. Beyond this there
ns to be little chance of economy in heat, although other consid-
ons might make still longer furnaces advisable.

*Spout.*

last Furnace No. 2. Dec. 1, 1910.

| | Observed Data. | | | Calculated Data. | | | | |
|---|---|---|---|---|---|---|---|---|
| | Net Weight of Water. Pounds. | Initial Temperature of Water. | Elapsed Time, Min.-Sec. | Average Discharging Temperature. °F. | Rise in Temperature. °F. | Pounds Water per Minute. | Gallons Water per Minute. | B.t.u. Removed per Minute. |
| . | 360.0 | 0.0 °C | 5– 0.7 | 98.9 | 66.9 | 71.83 | 8.61 | 4,805 |
| . | 362.0 | 0.0 °C | 5– 3.6 | 90.7 | 58.7 | 71.54 | 8.58 | 4,199 |
| I. | 358.5 | 0.0 °C | 5– 0.3 | 92.2 | 60.2 | 71.63 | 8.59 | 4,312 |
| . | 313.5 | 0.0 °C | 9– 1.2 | 165.2 | 133.2 | 34.76 | 4.17 | 4,630 |
| . | 311.5 | 0.0 °C | 9– 1.3 | 166.9 | 134.9 | 34.53 | 4.14 | 4,658 |
| I. | 309.5 | 0.0 °C | 9– 1.2 | 156.9 | 124.9 | 34.31 | 4.11 | 4,285 |
| I. | 420.0 | 0.0 °C | 3– 0.7 | 67.4 | 35.4 | 139.46 | 16.72 | 4,937 |
| I. | 417.0 | 0.0 °C | 2–59 6 | 65.8 | 33.8 | 139.31 | 16.70 | 4,709 |
| . | 419.5 | 0.0 °C | 3– 0.6 | 64.9 | 32.9 | 139.37 | 16.71 | 4,585 |

Detailed Temperature Readings and Notes.

ests I. to VI., inclusive, discharging temperature read with Fahrenheit thermometer; Tests VII.
ntigrade.

| | Test II. | | Test III. | | Test IV. | | Test V. | | Test VI. | | Test VII. | | Test VIII. | | Test IX. | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| °F. | Elapsed Time. | Discharging Temperature. °F. | Elapsed Time. | Discharging Temperature. °F. | Elapsed Time. | Discharging Temperature. °F. | Elapsed Time. | Discharging Temperature. °F. | Elapsed Time. | Discharging Temperature. °F. | Elapsed Time. | Discharging Temperature. °C. | Elapsed Time. | Discharging Temperature. °C. | Elapsed Time. | Discharging Temperature. °C. |
| .0 | 0-0.0 | 93.0 | 0-0.0 | 90.0 | 0-0.0 | 162.0 | 0-0.0 | 170.0 | 0-0.0 | 158.0 | 0-0.0 | 19.75 | 0-0.0 | 18.00 | 0-0.0 | 17.50a |
| .5 | 1-0.0 | 92.5 | 1-0.0 | 90.0a | 1-0.0 | 162.0 | 1-0.0 | 171.0 | 1-0.0 | 158.0 | 1-0.0 | 20.25 | 1-0.0 | 19.00 | 1-0.0 | 18.00 |
| .0 | 2-0.0 | 90.0 | 2-0.0 | 91.0 | 2-0.0 | 161.0a | 2-0.0 | 169.0 | 2-0.0 | 158.5 | 2-0.0 | 19.25 | 2-0.0 | 19.00 | 2-0.0 | 18.00 |
| .5 | 3-0.0 | 90.0 | 3-0.0 | 92.0 | 3-0.0 | 161.5 | 3-0.0 | 168.5 | 3-0.0 | 157.0 | 3-0.7 | 19.50 | 2-59.6 | 19.00 | 3-0.6 | 19.50 |
| .5 | 4-0.0 | 89.5 | 4-0.0 | 94.0 | 4-0.0 | 162.0 | 4-0.0 | 166.5 | 4-0.0 | 158.0 | ..... | ..... | Sl | ag | ..... | ..... |
| .0 | 5-3.6 | 89.0 | 5-0.3 | 96.0 | 5-0.0 | 166.0 | 5-0.0 | 167.0 | 5-0.0 | 156.0b | ..... | ..... | str | eam | ..... | ..... |
| | Volu | me | ..... | ..... | 6-0.0 | 167.0 | 6.0.0 | 166.0 | 6-0.0 | 156.0 | ..... | ..... | | low | ..... | ..... |
| | of | slag | ..... | ..... | 7-0.0 | 170.0 | 7-0.0 | 164.0 | 7-0.0 | 156.0 | ..... | ..... | ..... | ..... | ..... | ..... |
| e | less | than | ..... | ..... | 8-0.0 | 170.0 | 8-0.0 | 163.5 | 8-0.0 | 155.5 | ..... | ..... | ..... | ..... | ..... | ..... |
| | in | Test I | ..... | ..... | 9-1.2 | 170.0 | 9-1.3 | 163.5 | 9-1.2 | 156.0 | ..... | ..... | ..... | ..... | ..... | ..... |
| | ..... | ..... | ..... | ..... | ..... | ..... | Sl | ag | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| | ..... | ..... | ..... | ..... | ..... | ..... | strea | m low | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |

rges dumped.
stream much reduced in volume.

*Breast Jacket.*

Blast Furnace No. 2. Nov. 30, 1910.

| Test No. | Observed Data. | | | Calculated Data. | | | |
|---|---|---|---|---|---|---|---|
| | Net Wt. of Water. Pounds. | Initial Temp. of Water. | Elapsed Time. Min.-Sec. | Average Disch'g Temp. °F. | Rise in Temp. °F. | Pounds Water per Minute. | Gallo Water Minu |
| I. | 197.0 | 0.0°C | 19–0.4 | 110.5 | 78.5 | 10.36 | 1.2 |
| II. | 184.5 | 0.0°C | 19–4.6 | 119.4 | 87.4 | 9.67 | 1.1 |
| III. | 178.5 | 0.0°C | 19–1.7 | 122.0 | 90.0 | 9.38 | 1.1 |
| IV. | 345.5 | 0.0°C | 5–6.2 | 47.2 | 15.2 | 67.70 | 8.1 |
| V. | 340.5 | 0.0°C | 5–2.6 | 47.6 | 15.6 | 67.51 | 8.0 |
| VI. | 339.0 | 0.0°C | 5–1.0 | 48.5 | 16.5 | 67.57 | 8.1 |
| VII. | 339.0 | 0.0°C | 10–3.6 | 74.0 | 42.0 | 33.70 | 4.0 |
| VIII. | 331.0 | 0.0°C | 10–1.4 | 74.3 | 42.3 | 33.02 | 3.9 |
| IX. | 327.0 | 0.0°C | 10–2.2 | 75.2 | 43.2 | 32.58 | 3.9 |

## Detailed Temperature Readings and Notes.

Tests VII., VIII., and IX. were run in the afternoon, with a larger and hotter slag st morning when Tests I. to VI. were run.

| Test I. | | Test II. | | Test III. | | Test IV. | | Test V. | |
|---|---|---|---|---|---|---|---|---|---|
| Elapsed Time. | Discharging Temperature °F | Elapsed Time. | Discharging Temperature °F. | Elapsed Time. | Discharging Temperature °F | Elapsed Time. | Discharging Temperature °F. | Elapsed Time. | Discharging Temperature °F. |
| 0.0.0 | 107.0 | 0-0.0 | 118.0 | 0–0.0 | 122.0 | 0–0.0 | 47.5 | 0–0.0 | 47.5 |
| 1.0.0 | 108.0 | 1–0.0 | 118.0 | 1–0.0 | 122.0 | 1–0.0 | 47.0 | 1–0.0 | 48.0 |
| 2.0.0 | 108.0 | 2–0.0 | 118.0 | 2–0.0 | 122.0 | 2–0.0 | 47.0 | 2–0.0 | 48.0 |
| 3.0.0 | 108.0 | 3–0.0 | 118.0 | 3–0.0 | 122.0 | 3–0.0 | 47.5 | 3–0.0 | 48.0 |
| 4.0.0 | 108.0 | 4–0.0 | 118.0 | 4–0.0 | 122.0 | 4–0.0 | 47.0 | 4–0.0 | 48.0 |
| 5.0.0 | 109.0*a* | 5–0.0 | 118.5*a* | 5–0.0 | 122.0 | 5–6.2 | 47.0 | 5–2.6 | 46.0 |
| 6.0.0 | 110.0 | 6–0.0 | 118.5 | 6–0.0 | 122.0 | Test VII. | | Test VIII. | |
| 7.0.0 | 110.0 | 7–0.0 | 119.0 | 7–0.0 | 122.0*a* | 0–0.0 | 72.0 | 0–0.0 | 72.0 |
| 8.0.0 | 110.0 | 8–0.0 | 119.5 | 8–0.0 | 122.0 | 1–0.0 | 72.0 | 1–0.0 | 74.0 |
| 9.0.0 | 110.5 | 9–0.0 | 119.5 | 9–0.0 | 122.0 | 2–0.0 | 73.0*a* | 2–0.0 | 74.0 |
| 10.0.0 | 111.0 | 10–0.0 | 119.5 | 10–0.0 | 122.0 | 3–0.0 | 72.0 | 3–0.0 | 74.5 |
| 11.0.0 | 111.5 | 11–0.0 | 120.0 | 11–0.0 | 122.0 | 4–0.0 | 73.0 | 4–0.0 | 74.5 |
| 12.0.0 | 112.0 | 12–0.0 | 120.0 | 12–0.0 | 122.0 | 5–0.0 | 74.0 | 5–0.0 | 74.0 |
| 13.0.0 | 112.0 | 13–0.0 | 120.0 | 13–0.0 | 122.0 | 6–0.0 | 76.0 | 6–0.0 | 75.0 |
| 14.0.0 | 112.0 | 14–0.0 | 120.0 | 14–0.0 | 122.0 | 7–0.0 | 76.0 | 7–0.0 | 75.0 |
| 15.0.0 | 112.0 | 15–0.0 | 120.5*a* | 15–0.0 | 122.0 | 8–0.0 | 75.0 | 8–0.0 | 74.5 |
| 16.0.0 | 112.5 | 16–0.0 | 120.5 | 16–0.0 | 122.0 | 9–0.0 | 75.5 | 9–0.0 | 74.5 |
| 17.0.0 | 112.5 | 17–0.0 | 121.0 | 17–0.0 | 122.0 | 10–3.6 | 75.0 | 10–1.4 | 75.5 |
| 18.0.0 | 112.5 | 18–0.0 | 121.0 | 18–0.0 | 122.0 | ..... | ..... | ..... | ..... |
| 19.0.4 | 112.5 | 19–4.6 | 121.0 | 19–1.7 | 122.0 | ..... | ..... | ..... | ..... |

*a* Charges dumped.

*Main End Jackets.*

Furnace No. 2. Dec. 2, 1910.

l temperature of water 0.0° C. or 32° F. Each test was run on two east end jackets and lower).

| Observed Data. | | | | | Calculated Data. | | | | |
|---|---|---|---|---|---|---|---|---|---|
| Net Weight of Water. Pounds. | Discharging Temperature of Water. ° C. | Elapsed Time. Min.-Sec. | Avge. Temp. ° C. As Read. | Avge. Temp. ° C. Corrected. *a* | Rise in Temp. ° F. | Pounds Water per Minute. | Gallons Water per Minute. | B.t.u. Removed per Minute. |
| 405.5 | 11.1 | 0- 0.0 | 11.05 | 10.75 | 19.35 | 551.7 | 66.15 | 10,675 |
| | 11-0 | 0-44.1 | | | | | | |
| 402.5 | 11.0 | 0- 0.0 | 11.05 | 10.75 | 19.35 | 547.6 | 65.66 | 10,596 |
| | 11.1 | 0-44.1 | | | | | | |
| 397.0 | 10.1 | 0- 0.0 | 10.10 | 9.80 | 17.6 | 548.8 | 65.80 | 9,659 |
| | 10.1 | 0-43.4 | | | | | | |
| 395.5 | 10.0 | 0- 0.0 | 10.00 | 9.70 | 17.5 | 546.8 | 65.56 | 9,569 |
| | 10.0 | 0-43-4 | | | | | | |
| 408.0 | 10.1 | 0- 0.0 | 10.10 | 9.80 | 17.6 | 547.7 | 65.67 | 9,640 |
| | 10.1 | 0-44.7 | | | | | | |
| 403.0 | 10.0 | 0- 0.0 | 10.00 | 9.70 | 17.5 | 550.8 | 66.04 | 9,639 |
| | 10.0 | 0-43.9 | | | | | | |
| 411.0 | 16.1 | 0- 0.0 | 16.05 | 15.75 | 28.35 | 341.6 | 40.96 | 9,684 |
| | 16.0 | 1-12.2 | | | | | | |
| 411.5 | 16.0 | 0- 0.0 | 16.00 | 15.70 | 28.3 | 341.0 | 40.89 | 9,650 |
| | 16.0 | 1-12.4 | | | | | | |
| 406.5 | 16.0 | 0- 0.0 | 16.05 | 15.75 | 28.35 | 343.0 | 41.13 | 9,724 |
| | 16.1 | 1-11.1 | | | | | | |
| 409.5 | 16.8 | 0- 0.0 | 16.65 | 16.35 | 29.4 | 341.7 | 40.97 | 10,046 |
| | 16.5 | 1-11.9 | | | | | | |
| 407.0 | 17.0 | 0- 0.0 | 16.95 | 16.65 | 30.0 | 339.6 | 40.72 | 10,188 |
| | 16.9 | 1-11.9 | | | | | | |
| 412.5 | 18.0 | 0- 0.0 | 18.00 | 17.70 | 31.9 | 340.9 | 40.88 | 10,875 |
| | 18.0 | 1-12.6 | | | | | | |
| 385.0 | 66.4 | 0- 0.0 | 66.95 | 66.65 | 120.0 | 78.4 | 9.40 | 9,408 |
| | 66.8 | 1- 0.0 | | | | | | |
| | 66.9 | 2- 0.0 | | | | | | |
| | 66.9 | 3- 0.0 | | | | | | |
| | 67.1 | 4- 0.0 | | | | | | |
| | 67.6 | 4-54.6 | | | | | | |
| 377.5 | 68.7 | 0- 0.0 | 68.93 | 68.63 | 123.5 | 77.8 | 9.33 | 9,608 |
| | 68.8 | 1- 0.0 | | | | | | |
| | 69.0 | 2- 0.0 | | | | | | |
| | 69.0 | 3- 0.0 | | | | | | |
| | 69.1 | 4- 0.0 | | | | | | |
| | 69.0 | 4-51.2 | | | | | | |
| 384.5 | 70.0 | 0- 0.0 | 69.62 | 69.32 | 124.8 | 78.0 | 9.35 | 9,734 |
| | 70.0 | 1- 0.0 | | | | | | |
| Furnace Charged. | 69.9 | 2- 0.0 | | | | | | |
| | 69.5 | 3- 0.0 | | | | | | |
| | 69.8 | 4- 0.0 | | | | | | |
| | 68.5 | 4-55.6 | | | | | | |
| 382.0 | 65.0 | 0- 0.0 | 63.72 | 63.42 | 114.2 | 77.9 | 9.34 | 8,896 |
| | 64.3 | 1- 0.0 | | | | | | |
| | 63.8 | 2- 0.0 | | | | | | |
| | 63.1 | 3- 0.0 | | | | | | |
| | 63.0 | 4- 0.0 | | | | | | |
| | 63.1 | 4-54.2 | | | | | | |
| 393.5 | 64.0 | 0- 0.0 | 63.97 | 63.67 | 114.6 | 78.2 | 9.38 | 8,962 |
| | 64.1 | 1- 0.0 | | | | | | |
| | 64.4 | 2- 0.0 | | | | | | |
| | 64.4 | 3- 0.0 | | | | | | |
| Furnace Charged | 63.9 | 4- 0.0 | | | | | | |
| | 63-0 | 5- 2.1 | | | | | | |
| 396.5 | 62.5 | 0- 0.0 | 59.68 | 59.38 | 106.9 | 77.8 | 9.33 | 8,317 |
| Furnace Charged | 61.5 | 1- 0.0 | | | | | | |
| | 60.3 | 2- 0.0 | | | | | | |
| | 59.0 | 3- 0.0 | | | | | | |
| | 58.0 | 4- 0.0 | | | | | | |
| | 56.8 | 5- 5.8 | | | | | | |

nperatures of discharge water taken with thermometer reading 0.3° C. too high.
n of Corrected Average Temperatures gives the true values.

*Main Side Jackets.*

Blast Furnace No. 2 — Dec. 3 and 14, 1910.

| Test No. | Date, December 1910. | Observed Data. Net Weight of Water. Pounds. | Disch. Temp. °C. Start of Test. | Disch. Temp. °C. End of Test. | Elapsed Time. Min.-Sec. | Calculated Data. Av'ge Temp. As Read. | Av'ge Temp. Corrected. | Rise in Temperature. °F. | Pounds Water per Minute. |
|---|---|---|---|---|---|---|---|---|---|
| I. | 3 | 408.5 | 4.2 | 4.1 | 0–43.5 | 4.15 | 3.85 | 6.9 | 563.4 |
| II. | 3 | 399.0 | 4.0 | 4.0 | 0–42.5 | 4.00 | 3.70 | 6.7 | 563.3 |
| III. | 3 | 401.5 | 4.0 | 4.0 | 0–42.5 | 4.00 | 3.70 | 6.7 | 566.8 |
| IV. | 3 | 413.5 | 4.0 | 4.0 | 0–44.0 | 4.00 | 3.70 | 6.7 | 563.9 |
| V. | 3 | 400.0 | 4.1 | 4.1 | 0–42.4 | 4.10 | 3.80 | 6.8 | 566.0 |
| VI. | 3 | 412.5 | 3.5 | 3.3 | 0–43.8 | 3.40 | 3.10 | 5.6 | 565.1 |
| VII. | 3 | 391.5 | 5.0 | 5.0 | 1–12.3 | 5.00 | 4.70 | 8.5 | 324.9 |
| VIII. | 3 | 418.5 | 5.0 | 5.1 | 1–16.7 | 5.05 | 4.75 | 8.6 | 327.4 |
| IX. | 3 | 406.5 | 5.1 | 5.0 | 1–15.0 | 5.05 | 4.75 | 8.6 | 325.2 |
| X. | 3 | 392.5 | 5.1 | 5.5 | 1–13.0 | 5.30 | 5.00 | 9.0 | 322.6 |
| XI. | 3 | 401.5 | 6.2 | 6.1 | 1–14.1 | 6.15 | 5.85 | 10.5 | 325.1 |
| XII. | 3 | 415.5 | 6.0 | 5.7 | 1–17.0 | 5.85 | 5.55 | 10.0 | 323.8 |
| XIII. | 14 | 385.5 | ..... | ..... | 3–6.4 | 58.7 | 59.0 | 106.2 | 124.1 |
| XIV. | 14 | 385.0 | ..... | ..... | 3–11.8 | 53.3 | 53.6 | 96.5 | 120.4 |
| XV. | 14 | 397.0 | ..... | ..... | 3–12.8 | 52.1 | 52.4 | 94.3 | 123.5 |
| XVI. | 14 | 383.0 | ..... | ..... | 3–5.8 | 56.5 | 56.8 | 102.2 | 123.7 |
| XVII. | 14 | 394.5 | ..... | ..... | 3–9.8 | 60.3 | 60.6 | 109.1 | 124.7 |
| XVIII. | 14 | 401.5 | ..... | ..... | 2–54.4 | 67.1 | 67.4 | 121.3 | 138.1 |
| XIX. | 14 | 374.0 | ..... | ..... | 3–1.6 | 33.2 | 33.5 | 60.3 | 123.6 |
| XX. | 14 | 387.0 | ..... | ..... | 3–10.4 | 26.5 | 26.8 | 48.2 | 122.0 |
| XXI. | 14 | 399.0 | ..... | ..... | 3–0.4 | 24.9 | 25.2 | 45.4 | 132.7 |
| XXII. | 14 | 382.0 | ..... | ..... | 3–11.6 | 22.4 | 22.7 | 40.9 | 119.6 |
| XXIII. | 14 | 377.5 | ..... | ..... | 3–6.4 | 22.1 | 22.4 | 40.3 | 121.5 |
| XXIV. | 14 | 373.5 | ..... | ..... | 3–6.0 | 22.8 | 23.1 | 41.6 | 120.5 |
| XXV. | 14 | 384.0 | 11.3 | 11.2 | 1–8.0 | 11.25 | 11.55 | 20.8 | 338.8 |
| XXVI. | 14 | 369.0 | 10.5 | 10.5 | 1–4.4 | 10.50 | 10.80 | 19.4 | 343.8 |
| XXVII. | 14 | 365.5 | 8.5 | 8.4 | 1–4.6 | 8.45 | 8.75 | 15.8 | 339.5 |
| XXVIII. | 14 | 369.5 | 8.6 | 8.6 | 1–4.4 | 8.60 | 8.90 | 16.0 | 344.3 |
| XXIX. | 14 | 369.5 | 9.3 | 9.6 | 1–5.2 | 9.45 | 9.75 | 17.6 | 340.0 |
| XXX. | 14 | 371.5 | 9.7 | 10.1 | 1–6.0 | 9.90 | 10.20 | 18.4 | 337.7 |

## Detailed Temperature Readings, Tests 13 to 24, inclusive.

Initial temperature of water read 0.0 °C. Tests run on N. E. jackets.

| Test No. | XIII. | XIV. | XV. | XVI. | XVII. | XVIII. | XIX. | XX. | XXI. | XXII. |
|---|---|---|---|---|---|---|---|---|---|---|
| Readings taken at | Temperature °C. | Temperature °C. | Temperature °C. | Temperature °C. | Temperature °C. | Temperature °C. | Temperature °C. | Temperature °C. | Temperature °C. | Temperature °C. |
| Start of test..... | 58.5 | 55.0 | 51.5 | 54.9 | 58.2 | 65.5 | 34.0 | 27.5 | 25.5 | 23.0 |
| End of 1st minute.. | 58.8 | 52.5 | 51.7 | 55.8 | 59.7 | 67.0 | 33.1 | 26.5 | 25.1 | 22.5 |
| End of 2nd minute | 58.9 | 54.0 | 52.0 | 57.0 | 59.8 | 68.0 | 32.5 | 26.5 | 24.9 | 22.4 |
| End of 3rd minute | 58.5 | 52.3 | 53.0 | 57.5 | 62.0 | 68.0 | 33.0 | 26.0 | 24.0 | 22.0 |
| End of test..... | 59.0 | 52.5 | 52.5 | 57.5 | 62.0 | ..... | ..... | 26.2 | ..... | 22.0 |

*Main Side Jackets.*

rnace No. 4. Dec. 15, 1910.

Tests run on N. E. jackets. Initial temperature of water, 0° C. = 32° F.

| Observed Data. | | | | | Calculated Data. | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Disch'g. Temp. | | | | Av'g. Temp. °C. | | | | | |
| Net Weight of Water. Pounds. | Start—°C | End of 30 Seconds. | End of Test—°C. | Elapsed Time. Min.-Sec. | As Read. | Corrected. | Rise in Temperature. °F. | Pounds Water per Minute. | Gallons Water per Minute. | B.t.u. Removed per Minute. |
| 410.0 | ..... | ..... | ..... | 14– 1.4 | 37.6 | 37.9 | 68.2 | 29.24 | 3.51 | 1,994 |
| 381.5 | ..... | ..... | ..... | 14– 0.4 | 38.8 | 39.1 | 70.4 | 27.24 | 3.27 | 1,918 |
| 401.0 | ..... | ..... | ..... | 14– 0.4 | 41.9 | 42.2 | 76.0 | 28.63 | 3.43 | 2,176 |
| 386.5 | ..... | ..... | ..... | 14- 1.6 | 51.3 | 51.6 | 92.9 | 27.55 | 3.30 | 2,559 |
| 368.0 | ..... | ..... | ..... | 14– 0.2 | 52.9 | 53.2 | 95.8 | 26.28 | 3.15 | 2,518 |
| 347.0 | ..... | ..... | ..... | 14– 0.6 | 55.7 | 56.0 | 100.8 | 24.77 | 2.97 | 2,497 |
| 387.0 | 9 9 | 9.9 | 9.9 | 1– 7.8 | 9.9 | 10.2 | 18.4 | 342.48 | 41.06 | 6,302 |
| 375.5 | 9.2 | 9.3 | 9.2 | 1– 5.8 | 9.23 | 9.53 | 17.2 | 342.40 | 41.06 | 5,889 |
| 374.5 | 9.2 | 9.2 | 9.2 | 1– 5.6 | 9.2 | 9.5 | 17.1 | 342.53 | 41.07 | 5,857 |
| 370.0 | 9.0 | 9.0 | 9.0 | 1– 4.8 | 9.0 | 9.3 | 16.7 | 342.59 | 41.08 | 5,521 |
| 368.0 | 9.0 | 9.0 | 9.0 | 1– 4.6 | 9.0 | 9.3 | 16.7 | 341.80 | 40.98 | 5,708 |
| 372.0 | 8.6 | 8.5 | 8.4 | 1– 5.2 | 8.5 | 8.8 | 15.8 | 342.33 | 41.05 | 5,409 |
| 388.0 | 5.8 | ..... | 5.8 | 0–46.8 | 5.8 | 6.1 | 11.0 | 497.44 | 59.65 | 5,472 |
| 388.0 | 5.5 | ..... | 5.8 | 0–46.8 | 5.65 | 5.95 | 10.7 | 497.44 | 59.65 | 5,323 |
| 387.0 | 5.8 | ..... | 5.8 | 0–46.4 | 5.8 | 6.1 | 11.0 | 500.43 | 60.00 | 5,505 |
| 384.0 | 5.5 | ..... | 5.5 | 0–46.0 | 5.5 | 5.8 | 10.4 | 500.87 | 60.06 | 5,209 |
| 392.5 | 5.5 | ..... | 5.5 | 0–47.0 | 5.5 | 5.8 | 10.4 | 501.06 | 60.08 | 5,211 |
| 380.5 | 5.4 | ..... | 5.4 | 0–46.0 | 5.4 | 5.7 | 10.3 | 496.30 | 59.51 | 5,112 |

Detailed Temperature Readings, Tests 31 to 36, inclusive.

| No. | XXXI. | XXXII. | XXXIII. | XXXIV. | XXXV. | XXXVI. |
|---|---|---|---|---|---|---|
| aken at | Discharging Temperature. °C. | Discharging Temperature. °C. | Discharging Temperature. °C. | Discharging Temperature. °C. | Discharging Temperature. °C. | Discharging Temperature. °C. |
| of test.... | 36.9 | 39.6 | 37.7 | 47.8 | 55.5a | 53.7 |
| minute, | 37.0 | 40.2a | 38.2 | 48.0 | 56.5 | 54.5 |
| minute, | 37.3 | 40.5 | 38.7 | 48.5 | 55.0 | 54.9 |
| minute, | 37.6 | 40.4 | 39.3 | 49.0 | 54.3 | 55.1 |
| minute, | 37.9 | 40.3 | 40.0 | 49.5 | 53.5 | 55.2 |
| minute, | 38.0a | 39.9 | 40.3 | 50.2 | 52.7 | 55.0 |
| minute, | 38.0 | 39.6 | 41.0 | 50.9 | 52.1 | 54.9 |
| minute, | 37.9 | 39.0 | 41.4 | 51.1 | 51.8 | 54.9 |
| minute, | 37.8 | 38.5 | 42.0 | 51.9 | 51.4 | 55.1 |
| minute, | 37.5 | 38.0 | 42.6 | 52.6 | 51.4 | 55.6 |
| minute, | 37.3 | 37.5 | 43.5 | 53.2 | 51.5 | 56.1 |
| minute, | 37.5 | 37.1 | 44.5 | 53.5 | 51.6 | 57.0 |
| minute, | 37.6 | 37.0 | 45.5 | 54.0 | 51.9 | 57.5 |
| minute, | 37.8 | 37.0 | 46.5 | 54.6 | 52.1 | 58.0 |
| ..... | 38.0 | 37.0 | 47.2 | 55.2 | 52.7 | 58.4 |

s dumped.

*Summary of Results.*

| Series. (*i. e.*, a Group of Tests Run Under Similar Conditions). | Tests Nos. | Pounds Water per Minute. Average. | Gallons Water per Minute. Average. | Rise in Temperature °F. Avera |
|---|---|---|---|---|
| **Nose.** | | | | |
| First series, hot discharging . | 1–3 inclusive . | 8.58 | 1.03 | 71. |
| Sec'd series, warm discharging | 4–6 inclusive . | 23.50 | 2.82 | 39. |
| Average . . . . . | . . . . . | . . . | . . . | . . |
| **Spout.** | | | | |
| First series, warm discharging | 1–3 inclusive . | 71.67 | 8.59 | 61. |
| Second series, hot discharging. | 4–6 inclusive . | 34.53 | 4.14 | 131. |
| Third series, cold discharging | 7–9 inclusive . | 139.38 | 16.71 | 34. |
| Average . . . . . | . . . . . | . . . | . . . | . . |
| **Breast Jacket.** | | | | |
| First series, hot discharging . | 1–3 inclusive . | 9.80 | 1.17 | 85. |
| Sec'd series, cold discharging. | 4–6 inclusive . | 67.59 | 8.10 | 15. |
| Third series, warm discharging | 7–9 inclusive . | 33.10 | 3.97 | 42. |
| Average . . . . . | . . . . . | . . . | . . . | . . |
| **Main End Jackets.** | | | | |
| First series, cold discharging. | 1– 6 inclusive. | 548.9 | 65.81 | 18. |
| Sec'd series, cool discharging. | 7–12 inclusive. | 341.3 | 40.93 | 29. |
| Third series, hot discharging. | 13–18 inclusive. | 78.0 | 9.36 | 117. |
| Average . . . . . | . . . . . | . . . | . . . | . . |
| **Main Side Jackets.** | | | | |
| The average B.t.u. removed per minute by the Main Side Jackets, as calculated from | | | | |
| series, amounts to 5,645. However, the individual series show such variations in thi | | | | |
| average is of little value, the furnace conditions having varied widely while the tests | | | | |
| First series, cold discharging. | 1– 6 inclusive. | 564.8 | 67.72 | 6 |
| Sec'd series, cold discharging. | 7–12 inclusive. | 324.8 | 38.95 | 9 |
| Third series, hot discharging. | 13–18 inclusive. | 125.8 | 15.08 | 104 |
| Fo'th series, warm discharging. | 19–24 inclusive. | 123.3 | 14.79 | 46 |
| Fifth series, cold discharging. | 25–30 inclusive. | 340.7 | 40.85 | 18 |
| Sixth series, hot discharging. | 31–36 inclusive. | 27.29 | 3.27 | 84 |
| Sev'th series, cold discharging. | 37–42 inclusive. | 342.36 | 41.05 | 17 |
| Eighth series, cold discharging | 43–48 inclusive. | 498.92 | 59.83 | 10 |

*Average Heat Consumption of Side and End Jackets.*

Taken from Tests of Nov. 3, and Dec. 2, 14, and 15, 1910.

Side Jackets.

| of Test, 1910. | No. of Tests Represented. | Furn No. | B.t.u. Removed per Minute by | | |
|---|---|---|---|---|---|
| | | | Two Jackets. | All Jackets. | All Jackets. Approximate Figures for Computing. |
| 3..... | 4 | 4 | 26,099 | 104,397 | ..... |
| 14..... | 18 | 2 | 8,355 | 33,420 | ..... |
| 15..... | 18 | 4 | 4,465 | 17,860 | ..... |
| nd Averages... | 40 | ..... | 8,379 | 33,517 | 33,600 |

End Jackets.

| | | | | | |
|---|---|---|---|---|---|
| 3..... | 7 | 4 | 16,120 | 32,240 | ..... |
| 2..... | 18 | 2 | 9,715 | 19,430 | ..... |
| nd Averages.... | 25 | ..... | 11,508 | 23,017 | 23,000 |

*Effect of Length of Furnace on Heat Loss and Coke Consumption.*

on average heat value of blast furnace bin coke for December, 1910; 11,525 pound.

| Hearth Area Square. Feet. | Total B.t.u. Removed per Minute by | | | Equivalent Pounds Coke per Minute. | Pounds Coke forEquivalent Number of 15-Ft. Furnaces. | Ratio, Fuel for Combined Furnace to Fuel for 15-Ft. Furnaces. |
|---|---|---|---|---|---|---|
| | Side Jackets. | End Jackets. | All Jackets. | | | |
| | | | | | | Per Cent. |
| 70 | 33,600 | 23,000 | 56,600 | 4.91 | 4.91 | 100.0 |
| 140 | 67,200 | 23,000 | 90,200 | 7.83 | 9.82 | 79.7 |
| 210 | 100,800 | 23,000 | 123,800 | 10.74 | 14.73 | 72.9 |
| 238 | 114,240 | 23,000 | 137,240 | 11.91 | 16.69 | 71.4 |
| 280 | 134,400 | 23,000 | 157,400 | 13.66 | 19.64 | 69.6 |
| 350 | 168,000 | 23,000 | 191,000 | 16.57 | 24.55 | 67.5 |
| 406 | 194,880 | 23,000 | 217,880 | 18.90 | 28.48 | 66.4 |
| 420 | 201,600 | 23,000 | 224,600 | 19.49 | 29.46 | 66.2 |
| 490 | 235,200 | 23,000 | 258,200 | 22.40 | 34.37 | 65.2 |
| 560 | 268,800 | 23,000 | 291,800 | 25.32 | 39.28 | 64.5 |

of Anaconda blast furnaces.

*Percentage of Total Blast Furnace Jacket Area Contained in End Jackets, for* 
*of Furnace.*

Data.

Constant Width.—Between upper jackets, 6 ft. 0 in.
At tuyere level, 4 ft. 8 in.

Size of End Jackets.—Upper: Height, 7 ft. 5 in.; width, 6 ft. 6 in.
Lower: Height, 7 ft. 5 in.; width at top, 6 ft. 
bottom, 5 ft. 0 in., overlapping side jackets
side.

Size of Side Jackets.—Upper: Height, 7 ft. 5 in.
Lower: Vertical height, 7 ft. 5 in.; slant height,
length and number according to length of

NOTE:—With a length between end jackets of 15 ft., the above dimensio
the present 56 by 180 in. furnace.

| Length of Furnace. | Hearth Area. | Total Jacket or Radiating Surface. | Total Side Jacket Area. | Total End Jacket Area. | Square Feet of Radiating Surface per Square |
|---|---|---|---|---|---|
| Feet. | Sq. Ft. | Square Feet. | Square Feet. | Square Feet. | |
| 15 | 70 | 613.00 | 446.12 | 166.88 | 8.7 |
| 30 | 140 | 1,059.12 | 892.24 | 166.88 | 7.5 |
| 45 | 210 | 1,505.24 | 1,338.36 | 166.88 | 7.1 |
| 60 | 280 | 1,951.36 | 1,784.48 | 166.88 | 6.9 |
| 75 | 350 | 2,397.48 | 2,230.60 | 166.88 | 6.8 |
| 90 | 420 | 2,843.60 | 2,676.72 | 166.88 | 6.7 |
| 105 | 490 | 3,289.72 | 3,122.84 | 166.88 | 6.7 |
| 120 | 560 | 3,735.84 | 3,568.96 | 166.88 | 6.6 |

The amount of radiating surface per square foot of hearth is reduced 20
lengthening the furnace from 15 ft. to 60 ft., and 23.9 per cent. by length
15 ft. to 120 ft.

The relative area of the end jackets is reduced 68.4 per cent. by lengtheni
from 15 ft. to 60 ft., and 83.5 per cent. by lengthening it from 15 ft. to 120

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[SUBJECT TO REVISION.]

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# velopment of Blast-Furnace Construction at the Boston & Montana Smelter.

BY J. A. CHURCH, JR., GREAT FALLS, MONT.

(Butte Meeting, August, 1913.)

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## I. EARLY FURNACES.

R blast-furnace construction in America has long recognized l standard in the rectangular water-jacketed shaft with sep- rehearth. The details, however, and especially those relating dimensions, have had to meet a great variety of conditions, w no uniformity whatever. Selecting a few of the present stern plants, we find furnace lengths varying from 12.5 to 87 hs at tuyere level from 42 to 84 in., and heights above tuyeres t. 4 in. to 22 ft. 4 in.; and comparing present designs with the past, we meet contrasts no less striking. Unlike its se section, the length of a furnace has come to depend rather ize of unit desired than on the requirements of smelting; and uestion of length be ignored, several of the various existing can be grouped together on the basis of a common transverse This section, having a width at tuyere level of 56 in. and a bove tuyeres of 18 ft., was developed at Great Falls, and the its development contains some interesting features.

nd was broken for the Great Falls works in the spring of 1891. year later, the concentrator was able to begin operation, fol- ithin the next few months by the Brueckner, reverberatory, verter departments, and in April, 1893, by the first blast fur-

naces. These, with additions completed in 1894, included
having a total daily capacity of about 500 tons. Nos. 1 a
old furnaces, having already seen service at Butte; th
dimensions were 42 by 84 in. Nos. 3 and 4 were new.
120 in. long, 36 in. wide at the bottom of the jackets, 39⅜
the tuyeres, and 8 ft. 1 in. high from tuyeres to charge floo
1). The crucible formed part of the water-jacketed shaft
jackets extending 27 in. below the tuyere level and havi
cut at one end for the breast jacket. This general co
which was abandoned in the next furnace built, has reapp
17 years in the present No. 3, though the breast jacket, a
latter furnaces, is at the side. There were seven tuyere

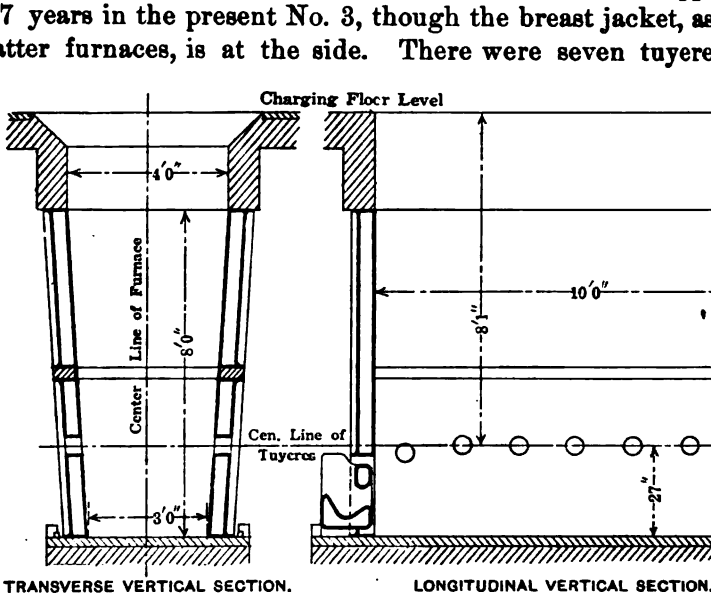


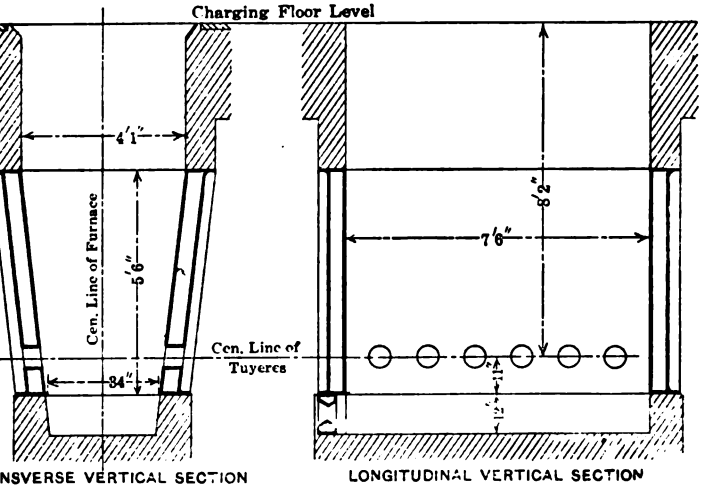

FIG. 1.—36 BY 180 IN. BLAST FURNACE. ERECTED AS NO. 3 O
ORIGINAL BLAST-FURNACE PLANT.

side, that nearest the discharge end being placed 2 in. low
others. No. 4 was 90 in. long, 34 in. wide at the bottom
ets, 36½ in. wide at the tuyeres, and 8 ft. 2 in. high from
charge floor (see Fig. 2). The jackets, extending 11 in.
tuyere level, rested on the walls of a brick crucible 12 in
breast jacket being set in the brick under one of the e
This became the standard type of crucible, from which n
was made until the year 1911. The tuyeres were 12 in nu
each side.

The forehearths of the furnaces were of the movable typ
at the time. That designed for No. 3 furnace consisted of
body lined with brickbats, and mounted on a wheeled tru

ensions being roughly 5 ft. length, 3 ft. width and 2 ft. depth. lining gave out, the whole affair was rolled out of the way laced by another, freshly lined.

is time the blast furnace was still something of an experiment tte ores, and the early practice was with small furnaces and t pressures. A majority of metallurgists in the Butte dis- ferred the reverberatory, and, with blast-furnace charges carry- uch as 90 per cent. of fines, this preference was not remark- n spite of low volumes and pressures of blast, the production lust was excessive; hand feeding was the rule, and the use of s looked upon with some suspicion. Nevertheless, the pro- raw-ore smelting had already made a beginning, and it only a gradual decline in the grade of ore to bring practice




G. 2.—34 BY 90 IN. BLAST FURNACE. ERECTED AS NO. 4 OF THE ORIGINAL BLAST-FURNACE PLANT.

present level. These were the conditions under which the alls furnacemen began their work, and out of which they pro- design of furnace that has contributed very largely to the nace production, not only of the district, but of the country. rst step in this development was the substitution of station- ers for the movable forehearths. This was tried in the spring , and proved an instant success. The first settler was built ace No. 1, with a diameter of about 7 ft., but within a short the furnaces were similarly equipped.

ord of the practice at this time will be found in Table I., ives the tonnage and composition of charge for each furnace

during August, 1894. It will be noted that Furnace No. 1
ning chiefly on re-treated material, 97 per cent. of its charg
ing of matte and converter slag. No assays appear for thi
but an assay of first-class ore, dated in October of the sa
shows a copper content of 25 per cent.; and another of co
centrates, of the same date, shows 20 per cent. of copper.

In April, 1895, a test was run on the Charles Allen proce
semerizing in the blast-furnace crucible. The crucible of fur
3 was equipped with a second set of tuyeres, connected
converter blast main, to raise the grade of the normal furna
as it came down from the smelting zone above. The experi
a failure, for it gave uncertain results and did a good deal o
to the jackets, which under these new conditions quickl
through; but it furnished some interesting figures, whic
found in Table II. The first two runs show the desired in
the grade of matte; and the second, in which slag and m
sampled together, shows a corresponding fall in the silicate de
a rise in the copper content of the slag. These changes
mark an approach to converter conditions. The third ru
other hand, gave no results whatever; blowing for 1.5 hr.
grade of matte only 0.3 per cent. Whatever the reason for
be, it is not shown by the records; but the fact itself he
plain why the work was dropped.

The regular practice in 1895 and 1896 is shown in Tabl
will be noted that furnace No. 1 was still re-treating larg
ties of matte and slag, and that its percentage of fuel is co
ingly low. Of the remaining three furnaces, No. 3 as the la
nace smelted the largest gross tonnage, and No. 4 showed
est tonnage per square foot of hearth area. The fuel burden
three furnaces were almost exactly the same, the weighted
for all months in which comparisons could be made being
2, 10.14 per cent.; for No. 3, 10.17 per cent.; and for No
per cent.

## II. Experiments with the High-Shaft Furnace.

In the spring of 1897 the original No. 1 furnace was re
an entirely new design. The high cost of using barren lim
flux the siliceous Butte ores had already made itself felt, an
was now made to replace more of this lime with ferrous oxid
from the ore itself. To increase the formation of ferrou
was proposed to expose the charge to furnace action for long
of time, and with this idea a very considerable increase wa

ight of shaft. As rebuilt, No. 1 was 24 ft. 5 in. high from level to charge floor. The jacketed portion of this height was ed by two tiers of jackets (see Fig. 3), of which the lower led 11 in. below the tuyeres and rested on the walls of a brick le, thus following the crucible construction of old No. 4. The side jackets were vertical, giving a shaft width of 48 in.; the tier was given a slight bosh, which reduced this width to 42 the bottom of the jackets and 42.5 at the tuyere level. The ckets were vertical, 180 in. apart.

furnace was blown in on July 16, 1897. The first difficulty ear was a lack of trap in the spout; but this was soon reme- and a number of campaigns were then made with fair results. ver, other and more serious troubles made their appearance,

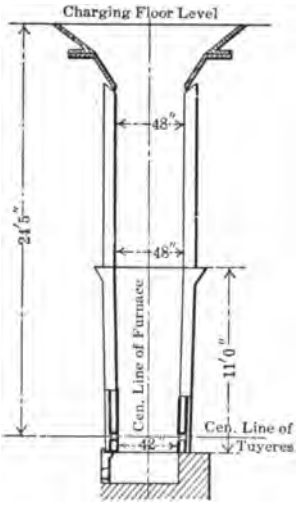



FIG. 3.—END ELEVATION OF NO. 1 BLAST FURNACE. ERECTED IN 1897.

ere so persistent that the type was finally abandoned. The most sting of these troubles, and the one most closely connected with onstruction of the furnace, was a decrease, instead of the expected se, in the degree of oxidation obtained. Any attempt to keep harge column up to the full height of the shaft always lowered rade of matte, and the furnace was usually run with a compara- low column. Although the blast pressures on No. 1 were much r than on the others, these difficulties were ascribed to insuffi- blast, and the high column was given a final test at the full ity of the blowing plant. For this purpose, the other furnaces temporarily shut down, and by shutting off some of the tuyeres . 1 the blast pressure was forced up to a maximum of 60 oz.

Nevertheless, the results were unsatisfactory, for the grad
decreased as before.

These difficulties of course indicate a lack of oxygen.
column was raised its resistance to blast was increased; and
pressure might rise, the volume of air fell off. This rule
not only in normal running, but also in the special test,
other furnaces were shut down; though in the latter case t
of blast was also decreased by reducing the tuyere area.
tuyeres had been left open enough blast might have been
secure the desired oxidization, although, with the type of bl
in use, an increase in height of column would always tend
pressure and lower the volume received by the furnace.
that the high column in No. 1 was never tested under adec
conditions, and that its results were inconclusive.

This does not mean that the design of No. 1 as a whole
even as an example of the high-shaft furnace. It was t
between upper jackets, and the formation of crust, whi
course, common enough, often led to bridging near the
column. Once formed, the crusts and bridges were hard
for the low columns and narrow shaft combined to mak
difficult.

There was another difficulty, due more to the experimen
of the furnace than to its actual design. All the furna
common charging floor; No. 1, therefore, was much lowe
others at the tapping level, and consequently the matte la
be set in a pit several feet below the normal level of the
floor. This pit collected water, and in case of a runaway
a source of danger.

When it was decided to remodel the furnace, the q
greatest width was being tested on Nos. 2 and 3 and no
results had been obtained. Moreover, it was desired to
quickly as possible, so the upper part of the shaft re
width of 48 in., and the width at the tuyeres was merely
from 42.5 to 44 in. The height was reduced to about 18 ft.,
with Nos. 2 and 3. No. 1 was remodeled on these lines
spring of 1899, and in its new form was blown in on Jun
same year. This type needs no further discussion, beca
more or less of a makeshift, and stands apart from the reg
opment of the Great Falls furnaces.

The record of No. 1 as the high-shaft furnace is sum
Tables IV. and V. For comparison, the records of Nos. 3
given in Table VI. for the same period as that covered b

l be seen that during the latter part of 1898 the matte from No. s slightly higher in grade than that from Nos. 3 and 4; but it be remembered that No. 1 was usually run with a compara- low column, so the results do not properly represent the high In fact, Table VI. shows that the height of column during eriod was about 15 ft., or 9 ft. less than the height of the shaft. lowing the usual practice at Great Falls, the oxygen ratios given ne slags in Tables IV., V., and VI. were calculated on the ption that alumina in the blast furnace acts as an acid. What- truth there may be in this assumption, it has been used in calcu- all the oxygen ratios given in the tables, and the results are fore fairly comparable. If it be true, most of the slags pro- l in these earlier years were abnormally acid, compared to nt practice; and those from No. 1, shown in Tables IV. and V., no exception.

### III. Experiments with the Wide Furnace.

perience with No. 1 during 1897-98 pointed towards a lower vider furnace. A reduction in height would remove what ed to be a useless part of the shaft, and would avoid tapping into below the converter floor level. An increase in width between jackets would avoid the sticking and bridging so common in arrow shaft of No. 1. Accordingly, during the year 1899 two designs were made, for furnaces 17 ft. 11⅝ in. high from tuyere to charging floor, 180 in. long between end jackets, and 72 in. between upper side jackets. Just why the foregoing height hosen does not appear; but it probably was governed chiefly by vailable head room, after allowing for the desired change in tap- level. The decision as to width was a matter of judgment, ed by experience with No. 1.

ere remained the question of width at the tuyeres. Furnaces 9, and 42 in. wide at this point had given good service, and there o reason to suppose that the limit had been reached. Accord- , the whole subject was tested by making the new units of dif- t widths at the tuyere level, both greater than anything tried up at time. No. 2 was built on its present lines, with a width at uyeres of 56 in. (See Fig. 4.) No. 3 was given a correspond- vidth of 72 in., the side jackets being vertical from top to bot- (See Fig. 5.)

is evident that No. 2 was a compromise between the extremes of 1 and 3, and that the more typical results for a wide furnace to be expected from No. 3. The record of No. 2 is therefore n up later, to show its survival as a standard type.

No. 3 was blown in with the 72 by 180 in. hearth section on Mar. 24, 1899. The furnace ran only about nine months, and in that time developed operating troubles which finally caused a complete change in design. One of these troubles was the production of sow. Another

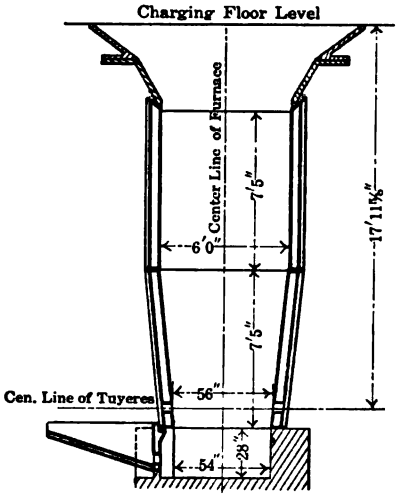



FIG. 4.—TRANSVERSE VERTICAL SECTION OF 56 BY 180 IN. BLAST FURNACE ERECTED IN 1899, AS NO. 2.

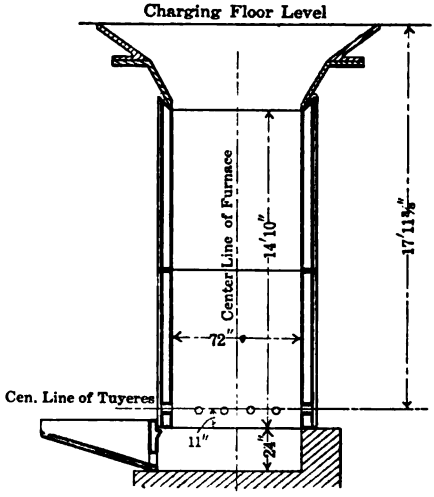



FIG. 5.—TRANSVERSE VERTICAL SECTION OF 72 BY 180 IN. BLAST FURNACE ERECTED IN 1899, AS NO. 3.

was the tendency of the blast to rise along the jackets, instead of spreading evenly through the charge column; and with this condition there appeared heavy, loose banks, or crusts, on the sides, which made the furnace hard to handle. A third difficulty, of less importance

se other two, was the relatively low temperature of the stream rom the spout. The worst of these troubles seems to have production of sow; at times it caused the freezing up of the and at others interfered with the running of the settler, clos- ap holes and making it necessary to tap at a higher level.

e of these operating difficulties, No. 3 was very little behind actual results. The detailed records of the two furnaces will in Tables VII. and IX., from which the following summary taken for purposes of comparison. It should be said that o suffered from uneven blast and production of sow, though uch as No. 3.

| Period. | April-June. | July-September. | October-December. |
|---|---|---|---|
| *No. 2 Furnace.* | | | |
| composition of charge: | | | |
| ter slag, | 23.9 | 23.6 | 20.3 |
| trates, | 24.2 | 18.2 | 20.7 |
| | 26.5 | 34.2 | 38.1 |
| one, | 24.9 | 23.4 | 25.0 |
| d per operating day, | 441 | 391 | 350 |
| erating day per sq. ft. hearth, | 6.3 | 5.6 | 5.0 |
| of fuel, | 8.3 | 9.3 | 9.5 |
| re, ounces, | 35 | 33 | 33 |
| *No. 3 Furnace.* | | | |
| composition of charge: | | | |
| ter slag, | 26.1 | 24,0 | 21.7 |
| trates, | 26.7 | 19.5 | 22.1 |
| | 23.3 | 33.2 | 31.1 |
| one, | 23.3 | 23.0 | 23.8 |
| d per operating day, | 350 | 420 | 312 |
| erating day per sq. ft. hearth, | 3.9 | 4.7 | 3.5 |
| of fuel, | 8,9 | 9.3 | 10.2 |
| re, ounces, | 33 | 33 | 32 |

figures show that the two furnaces ran on practically the rge. Since both were connected to a common blast main, the ssures are naturally almost identical. In gross tonnage per day, ceeded No. 3 in the first and third periods, but fell behind in the and for the three periods combined No. 2 is again slightly in

In tonnage smelted per square foot of hearth No. 2 always No. 3. In the first and third periods No. 2 was slightly more cal in fuel; but in the second, when No. 3 showed a better nage, the two furnaces in this respect were exactly alike.

pears, therefore, that the wide form was not abandoned on ac- its record, which was very nearly as good as that of the form

which superseded it. No figures are available to show the relative volumes of air received by Nos. 2 and 3, but their similarity in daily tonnage, in blast pressures, and, as far as recorded, in heights of column, suggests that these volumes were not very different. That is, the records of the two furnaces seem to have been controlled by some factor other than furnace design, and of all external factors, the blast conditions are usually the most important. There is some reason, therefore, to suppose that the daily tonnages of both No. 2 and No. 3 merely kept pace with the volumes of air supplied, and that on this account No. 3, as the larger furnace, showed a lower tonnage per square foot of hearth. The production of sow suggests that the action of the furnace was not strongly oxidizing; and with more air No. 3 might have shown improvement, not only in capacity, but also in ease of handling. The results of this form were therefore inconclusive, in so far as they related to a width of 72 in. at the tuyeres.

On the other hand, nothing seems to have been gained by making the sides vertical throughout. A cool stream at the spout implies a low temperature at the melting zone; and, though various factors may have helped to produce this effect, the experience of the plant has been that sharper smelting action usually follows a contraction of the lower jackets.

These conclusions agree fairly well with the results of a still wider furnace, the present No. 3, which is described farther on. This new design, which is 84 in. wide at the tuyeres, differs from the 72 by 180 in. type in almost every feature aside from width, having greater height, a sharp bosh in the lowest jackets, and a blast supply more than adequate to its needs. The good results it has given show that the 72 by 180 in. furnace had by no means reached the limit of width.

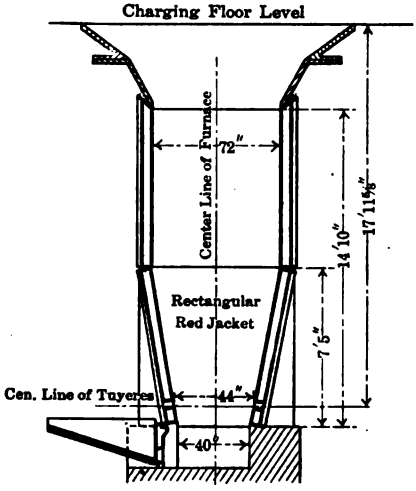
## IV. Experiments with Extreme Bosh.

In January, 1900, No. 3 was remodeled, with the idea of combining the good points of Nos. 1 and 2. No. 1, which was 44 in. wide at the tuyeres and 48 in. wide between upper jackets, was usually in good condition at the smelting zone, but gave a good deal of trouble by crusting and bridging over on top. No. 2, which was 56 in. wide at the tuyeres and 72 in. wide between upper jackets, was easier to keep open on top, but did not handle its blast as well in the smelting zone. The new furnace was therefore built 72 in. wide between upper jackets and 44 in. wide at the tuyeres, making the bosh very marked (see Fig. 6). The height was about 18 ft. and the length 180 in., as before.

On Jan. 18, 1900, the furnace was blown in. The campaign lasted

three months, and gave such poor results that it was never re-
ted. During February and March the ore supply carried a good
of fines, and at times all the furnaces suffered; but No. 3 was in
ble nearly all the time, and had the poorest record of all. On
. 24 the crucible broke out at the back, and advantage was taken
his accident to shut down the furnace and rebuild it on the lines
No. 2. In its new form, No. 3 was blown in again on Apr. 29,
0, and thereafter gave good results.

Vith the narrow hearth and sharp bosh, No. 3 developed troubles
ost exactly the opposite of those which marked the wide hearth
vertical sides. It discharged a hot stream at the spout, and made
ost no sow, but the crusts were so heavy and so hard that the fur-




IG. 6.—TRANSVERSE VERTICAL SECTION OF 44 BY 180 IN. BLAST FURNACE. ERECTED IN 1900, AS NO. 3.

could scarcely be kept open on both ends at once. Some of
e results, high smelting temperature and absence of sow, had been
cted, but the excessive crusting was a surprise, for the furnace
been designed to gain all the advantages of No. 1 at the hearth,
out its tendency to crust on top. As far as the hearth conditions
concerned, the design was a success. Smelting temperatures
high, and the absence of sow suggests no lack of oxidation.
ever, the extreme hardness of the crusts shows that the high
erature of the smelting zone was reflected in the upper part of
column, to an extent never found in either No. 1 or No. 2. Active
ting apparently began at a higher level in No. 3; and from a
parison of the three types, this seems to have been due to the

rapid convergence of the lower jackets, which tended towards an earlier concentration of smelting action. The flat slope of the jackets also gave some mechanical support to the crusts, making them harder to handle. As far as the records go, therefore, the design failed on account of inherent defects.

In the present No. 3, described later, the bosh of the lowest jacket is again very marked, but it extends a much shorter distance above the tuyeres, and represents a much smaller proportion of the total height. It seems merely to intensify smelting action at the proper level, without having much effect on the column above.

Table VIII. gives the record of No. 3 for January, February, and March, 1900. In per cent. of coke and tonnage per square foot of hearth, the furnace almost exactly equaled No. 2 for the same period. its gross tonnage, of course, was less.

## V. Survival of the 56 by 180 In. Furnace.

At the end of April, 1900, the blast-furnace department comprised three units. In hearth section, No. 1 was 44 by 180 in., and Nos. 2 and 3 were 56 by 180 in.; all three had a height of 17 ft. 11⅜ in. from tuyeres to charge floor. A fourth furnace was blown in on Oct. 2, 1900, and a fifth on Feb. 22, 1901, both conforming to the 56 by 180 in. type. Finally, in January, 1905, No. 1 was altered to the same lines, and started again on Aug. 24 of the same year.

The type first represented by No. 2 was thus the only one to survive. The causes of its survival may be found in the conditions, especially those of blast, under which the furnaces were operated. Until 1911, testing was confined almost wholly to furnace design, and the blast supply received less attention. The 56 by 180 in. type was therefore the product of a special set of conditions, in which a radical change might have developed a very different style of furnace. Lately steps have been taken towards new methods and new conceptions in the matter of blast supply, and one of the first results has been the successful operation of a furnace which differs in almost every essential from the older type.

This does not mean that the question of blast has been neglected. The early furnaces were run on 6 or 8 oz. pressure, in accordance with the practice of the time. The advent of larger units called for higher pressures, and No. 1, when it had the high shaft, was run on from 22 to 40 oz. (see Tables IV. and V.). During the same period pressures were also higher on the older furnaces, Nos. 3 and 4 being run on from 10 to 18 oz. (see Table VI.). During 1899 and 1900, when three large units were in operation, the pressures varied from

o 36 oz. Since then, in spite of the addition of two more units, have risen to 40 and 44 oz. These figures show that the blast ly has been gradually increased, but that a good deal of the ase has come since the 56 by 180 in. type became established. real advance in methods of supplying blast was made when the en of the air came to be regarded as part of the furnace charge, a definite volume of blast was maintained, irrespective of the ure involved.

e bad points of the other types show why No. 2 met its work-conditions so well. It was 6 ft. lower than the high-shaft furnace, thus offered less resistance to the passage of blast; it received air, therefore, at the pressure applied. It was 24 in. wider than ame furnace between upper jackets, and so avoided a good deal ouble in handling crusts. It was 16 in. narrower than the wide ace at the tuyeres, obtaining a better distribution of blast and r conditions in the smelting zone, and its reduction in width at uyeres was not enough to form a very pronounced bosh, so that roubles of No. 3 in its 44 by 180 in. form were not repeated. In parison with No. 2, all the other types were extreme in some par-ar feature, and No. 2 itself was a compromise among all of them. o. 2 improved upon the other types not only in ease of handling, also in capacity. During the period from April to December, sive, 1899, it treated 393 tons of charge per day, excluding fuel, st 364 tons treated by No. 3 in its 72 by 180 in. form. During ary, February and March, 1900, it treated 360 tons, against 273 treated by No. 3 in its 44 by 180 in. form. The high-shaft furnace torn down about the time No. 2 was started, but its tonnage g the last four months of 1898 was only 282 tons per day.

rious minor improvements have been made in the 56 by 180 in. ace since it was first designed, but only one of these is of much rtance. Up to 1911, the floor of the crucible was always built the breast opening, forming a shallow well at the bottom of the ace. Extra tap holes were left at the ends and back, entering vell just above the floor, and if tapping out was necessary, the nts of the crucibles were run into earth beds behind the furnace, broken up and wheeled away when cold. The furnace spout eased in depth towards the outer, or discharge, end, so that nothing l be run out through the breast opening unless there was blast gh to overcome the trap (see Fig. 6). When the present No. 3 designed, the crucible floor was sloped downward from all direc-towards the breast opening, which was thus at the lowest point e furnace. The spout increased in depth outward from the breast,

and its discharge end was covered by a thick plate, the lower portion of which was pierced near the bottom by a tap hole (see Fig. 7). To tap out the crucible, it was necessary simply to open this tap hole and run the molten material through the spout into the settler, where it belongs. This design, which was borrowed from another plant, not

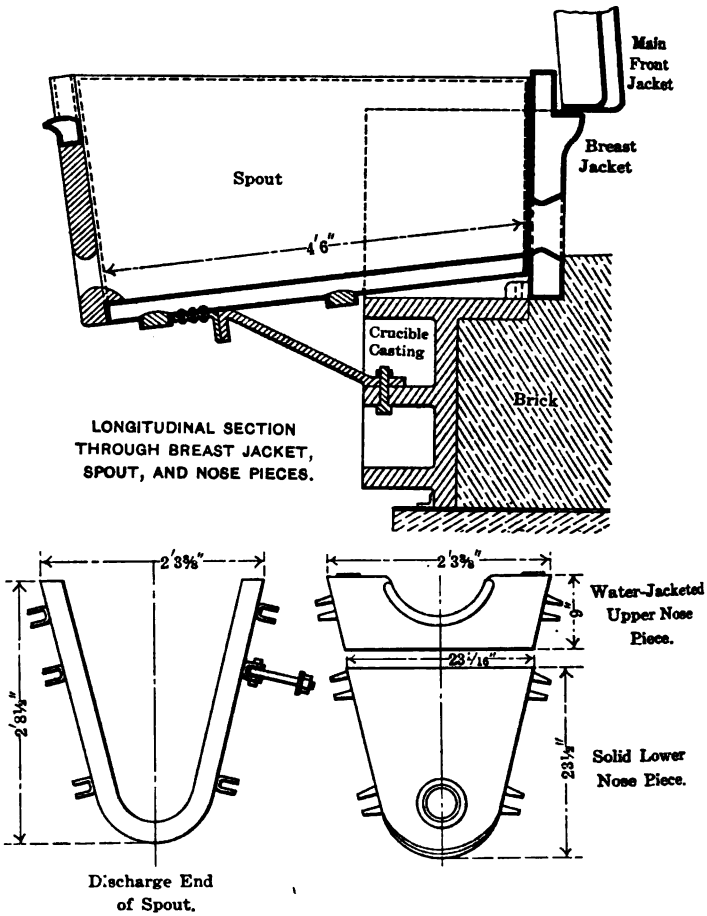


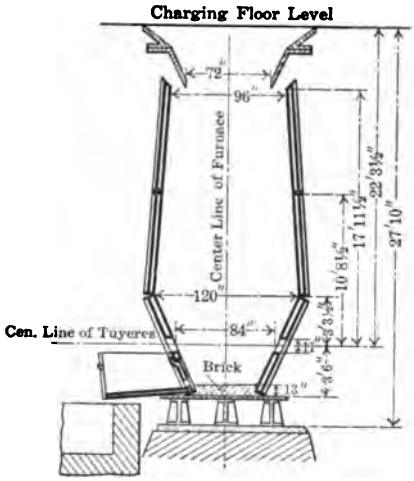

FIG. 7.—BLAST-FURNACE SPOUT INSTALLED ON FURNACE NO. 2 IN 1912.

only does away with bedding the slag, but increases the temperature of the stream running from the spout, for the molten material escapes from the crucible as fast as it formed, and undergoes no period of stagnation and cooling below the breast opening. Since its trial in No. 3, some of the 56 by 180 in. furnaces have been altered to the same design.

## VI. RECENT EXPERIMENTS.

Under the old conditions of blast supply, the 56 by 180 in. type became the standard of the plant, and from 1905 to 1911 it was the

style of furnace in use. In the latter year, however, experiments made with a new type of blower and a new design of furnace, the idea of developing a larger transverse section, which should e more air and smelt more material, without calling for an in- e in labor costs. The full discussion of this new design is ved for the future, but a brief description can be given here. rnace No. 3, which was selected for the experiment, was rebuilt 11. Its length is 180 in., as before, the novelty of the design ing entirely to the transverse section. Its height is 22 ft. 4 in. tuyeres to charge floor, and includes three tiers of jackets. It in. wide at the tuyeres, 120 in. wide at the top of the lowest tier ckets, and 96 in. wide at the top of the uppermost tier of jack-




. 8.—TRANSVERSE VERTICAL SECTION OF 84 BY 180 IN. BLAST FURNACE. ERECTED IN 1911, AS NO. 3.

ee Fig. 8). The sides are thus boshed in two directions, the or normal bosh being confined to the lowest jackets, and the sed bosh extending over the two upper tiers; the normal bosh is marked, the reversed bosh more gradual. The lowest jackets on cast iron plates, supported by short cast columns. These are covered within the furnace by a thin layer of brick, sloped ward towards the opening in the breast jacket, which is set in ess cut in one of the side jackets. The tuyeres are placed about ay up the lowest tier, the water-jacketed space underneath serv- s a crucible; it will be remembered that the essential features is construction were used in one of the early furnaces, the origi- o. 3. The tuyeres, 56 in number, are in the form of vertical 11 in. long and 3 in. wide.

The furnace is supplied with blast by a turbo-blower of high capacity, equipped with a constant-volume governor; the flow of air through the tuyeres is thus kept constant in volume, without regard to the pressure involved. There is a limit, of course, to the pressure at which the machine will work, but at the required volumes, from 17,000 to 20,000 cu. ft. of free air per minute, this limit is never approached.

The furnace spreads its blast evenly, shows some economy in coke, and under normal conditions discharges a hot stream at the spout. It has made some record runs, maintaining a capacity for short periods as high as 7 tons per square foot of hearth per day. The charge column, although sometimes raised nearly to the full height of the shaft, is usually held a few feet lower. One object of this practice is to obtain a better distribution of the charge, as it falls from the cars. All the furnaces are fed from the side, but in No. 3 the charging plates overhang the shaft for a distance of 12 in. (see Fig. 8), so that there is a narrow strip along each side which cannot be fed higher than the lower edge of the plate. At the height of column ordinarily adopted, however, this arrangement of the charging plates helps to keep the center level with the sides, and in fact is a step towards overhead charging.

## VII. Summary.

The results of these experiments in furnace construction are difficult to summarize, because they have developed no limiting dimensions. However, they point out certain tendencies in operation which must be remembered, if the size of furnace is to be much increased. They also show the importance of the blast, and open up a field for future research in methods of supplying and controlling the air.

In general, an increase in height of column calls for higher blast pressures, if the volume of blast is to remain unchanged. The cost of air compression rises with the pressure involved, and thus may possibly set a limit to economical height. Furthermore, in plants which have no system of maintaining a uniform charge, any trouble with poor slag at the spout is more serious with a high column than with a low one, because the high column does not allow corrective charges to descend so quickly to the breast, where they are needed. It also makes the handling of deep crusts more difficult. On the other hand, a higher column prolongs the period of gradual heating in the upper part of the furnace, and reduces the waste of heat in flue gases. Furthermore, it tends to prevent the escape of unconsumed oxygen, by exposing the charge for a longer time to oxidation. The column must be at least high enough to insure an even distribution

n the lower part of the furnace, and thus depends to some
the width at the tuyeres.

y one length of furnace, an increase in width at the tuyeres
increase in hearth area. If the volume of blast per square
earth is to remain unchanged, either the blast pressure must
or the tuyere area increased. The latter course, which is
economical, will sooner or later bring up some interesting
in jacket design. To some extent, the width at the tuyeres
s the width on top, to form the desired bosh. As the width
creases, the difficulties of side charging from cars, as prac-
Great Falls, will probably call for some system of overhead
, and thus introduce new mechanical features. Although
of this sort have been worked out at other plants, the unusual
hich seem possible in future furnaces may bring up problems
own.

, the oxygen of the blast is a part of the furnace charge,
nethod of blast supply should allow some sort of control over
ne, and hence the weight, of oxygen received by the furnace.

E I.—*Details of Operation of the Blast-Furnace Department during August, 1894.*

*Tonnages.*

| No. of Charges Smelted. | Total Tons Smelted, Exclusive Fuel. | Average Pounds Weight per Charge, Exclusive Fuel. | Pounds Fuel. | | Per Cent. of Fuel. |
|---|---|---|---|---|---|
| | | | Coke. | Anthracite Coal. | |
| 3,938 | 3,938.0 | 2,000 | 82,817.0 | 31,370 | 10.9 |
| 3,088 | 3,128.0 | 2,026 | 70,464.0 | . . . | 11.3 |
| 3,032 | 3,025.0 | 1,995 | 68,992.0 | . . . | 11.4 |
| 1,670 | 1,671.0 | 2,001 | 38,468.0 | . . . | 11.5 |

Nos. 1, 2, and 3 remained in operation throughout the month. No. 4 was the 14th.

lust was mixed with water to form a paste or mud, and charged wet into the

*Average Composition of Charge.*

| Per Cent. of | | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| Lime Rock. | Converter Slag. | Blast Furnace Slag and Chips. | Furnace Refuse Slag. | Concentrates. | First-Class Ore. | Flue Dust. | Matte. | Calcine. |
| 0.23 | 40.50 | 1.78 | 0.04 | 0.49 | 0.28 | 0.04 | 56.62 | 0.02 |
| 3.17 | 31.39 | 1.62 | 3.33 | 24.58 | 13.74 | 0.47 | 7.08 | 4.61 |
| 4.71 | 27.38 | 1.70 | 3.56 | 28.15 | 15.18 | 0.84 | 3.33 | 5.14 |
| 5.58 | 25.50 | 1.22 | 3.96 | 28.23 | 16.61 | . . | 0.51 | 8.38 |

TABLE II.—*Results of a Test on the Allen Process of Bessemerization in the Blast-Furnace Crucible, April, 1895.*

All Tests run on Blast Furnace No. 3.

*Test run on April 9.* [a]

| Time Sampled. | Matte Grade. | Corresponding Slag. | | | | | |
|---|---|---|---|---|---|---|---|
| | Per Cent. Cu. | Per Cent. Cu. | Per Cent. $SiO_2$. | Per Cent. $Al_2O_3$. | Per Cent. FeO. | Per Cent. CaO. | Oxygen Ratio. |
| Before Bessemerizing.. | 39.6 | . . . | . . . . . | . . . . . | . . . . . | . . . . . | . . . . |
| After Bessemerizing.... | 51.5 | . . . | . . . . . | . . . . . | . . . . . | . . . . . | . . . . |

*a.* The composition of the charge on this date was as follows:

| | Pounds. |
|---|---|
| Concentrates, | 800 |
| Ore, | 350 |
| Slag, | 350 |
| Lime rock, | 300 |
| Calcines, | 200 |
| Total, | 2,000 |

*Test run on April 11.*

| | | | | | | | |
|---|---|---|---|---|---|---|---|
| Before Bessemerizing.. | 37.5 | . . . | . . . . . | . . . . . | . . . . . | . . . . . | . . . . |
| After blowing 20 min. | 38.0 | 0.3 | 35.6 | 9.9 | 37.0 | 16.7 | 1.8099 |
| After blowing 1 hr. 40 m. | 45.3 | 1.0 | 33.8 | 8.2 | 38.7 | 17.5 | 1.6000 |
| After blowing 2 hr..... | 45.7 | 1.8 | . . . . | . . . . | . . . . . | . . . . . | . . . . |

*Test run on April 13.*

| | | | | | | | |
|---|---|---|---|---|---|---|---|
| Before Bessemerizing... | 39.0 | . . . | . . . . . | . . . . . | . . . . . | . . . . . | . . . . |
| After blowing 1 hr. 30 m. | 39.3 | . . . | 38.0 | 9.4 | 29.3 | 21.3 | 1.9492 |

BLE III.—*Details of Operation of the Blast-Furnace Department during 1895–96.*

| Blast Furnace No. | Composition of Charge. Tons. Total, Exclusive Fuel. | Ore, Concentrates, Calcine. | Matte. | Slags. | Lime Rock. | Per Cent. Ore, Concentrates, Calcine. | Matte. | Slags. | Lime Rock. | 24-Hour Days Operated. | Tons per Day, Exclusive Fuel. | Coke. Tons. | Per Cent. | Tons per Day per Sq. Ft. of Hearth Area. | Ounces Blast Pressure. |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | 4,227 | 216 | 2,923 | 1,022 | 66 | 5.1 | 69.2 | 24.2 | 1.5 | 30 | 140.9 | 412 | 9.7 | 5.75 | 4.8 |
| 2 | 2,816 | 1,402 | 466 | 549 | 399 | 49.9 | 16.6 | 19.2 | 14.3 | 25 | 112.6 | 312 | 11.1 | 4.60 | 8.8 |
| 3 | 3,764 | 2,189 | 42 | 870 | 663 | 58.2 | 1.1 | 23.1 | 17.6 | 28½ | 132.1 | 406 | 10.8 | 4.40 | 8.8 |
| 4 | 2,002 | 1,218 | 39 | 365 | 380 | 60.9 | 1.9 | 18.2 | 19.0 | 18½ | 108.2 | 222 | 11.1 | 5.09 | 8.6 |
| 3 | 3,911 | 2,281 | 99 | 789 | 742 | 58.3 | 2.5 | 20.2 | 19.0 | 31 | 126.2 | 434 | 11.1 | 4.21 | ..... |
| 3 | 4,208 | 2,707 | ..... | 621 | 880 | 64.3 | ..... | 14.8 | 20.9 | 27½ | 153.0 | 425 | 10.1 | 5.10 | ..... |
| 3 | 4,475 | 2,778 | ..... | 645 | 1,052 | 62.1 | ..... | 14.4 | 23.5 | 30 | 149.2 | 450 | 10.1 | 4.97 | ..... |
| 3 | 3,306 | 2,003 | ..... | 550 | 753 | 60.6 | ..... | 16.6 | 22.8 | 25½ | 129.6 | 335 | 10.1 | 4.32 | ..... |
| 1 | 1,951 | 513 | 578 | 673 | 192 | 26.3 | 29.3 | 34.5 | 9.9 | 15 | 130.1 | 188 | 9.5 | 5.31 | 9.6 |
| 2 | 3,429 | 1,237 | 794 | 946 | 452 | 36.0 | 23.2 | 27.6 | 13.2 | 26½ | 129.4 | 332 | 9.7 | 5.28 | 13.0 |
| 3 | 2,843 | 1,756 | 10 | 425 | 652 | 61.7 | 0.4 | 14.9 | 23.0 | 19 | 149.6 | 285 | 10.0 | 4.99 | 15.8 |
| 4 | 3,467 | 2,039 | 149 | 525 | 754 | 58.8 | 4.3 | 15.1 | 21.8 | 28½ | 121.6 | 348 | 10.0 | 4.72 | 12.8 |
| 1 | 3,843 | 1,192 | 982 | 1,292 | 377 | 31.0 | 25.6 | 33.6 | 9.8 | 30½ | 125.3 | 365 | 9.5 | 5.11 | ..... |
| 2 | 2,798 | 1,630 | 147 | 451 | 564 | 58.3 | 5.2 | 16.4 | 20.1 | 26⅔ | 104.9 | 281 | 10.0 | 4.28 | ..... |
| 3 | 3,637 | 2,211 | ..... | 553 | 873 | 60.8 | ..... | 15.2 | 24.0 | 24⅔ | 147.4 | 366 | 10.1 | 4.91 | ..... |
| 4 | 3,405 | 2,082 | 11 | 507 | 805 | 61.2 | 0.3 | 14.9 | 23.6 | 29⅔ | 114.8 | 342 | 10.0 | 5.40 | ..... |
| 1 | 3,845 | 615 | 1,548 | 1,545 | 137 | 15.9 | 40.3 | 40.3 | 3.5 | 29 | 132.6 | 353 | 9.2 | 5.41 | ..... |
| 2 | 2,123 | 1,143 | 38 | 521 | 421 | 53.8 | 1.8 | 24.5 | 19.9 | 18½ | 115.8 | 213 | 10.0 | 4.73 | ..... |
| 3 | 4,436 | 2,568 | 18 | 764 | 1,086 | 58.0 | 0.4 | 17.2 | 24.4 | 28½ | 154.7 | 443 | 10.0 | 5.16 | ..... |
| 4 | 3,288 | 1,904 | 10 | 571 | 803 | 57.9 | 0.3 | 17.4 | 24.4 | 28⅓ | 116.0 | 329 | 10.0 | 5.46 | ..... |
| 1 | 5,131 | 570 | 3,028 | 1,480 | 53 | 11.1 | 58.9 | 28.9 | 1.1 | 31 | 165.5 | 463 | 9.0 | 6.76 | ..... |
| 2 | 3,943 | 1,976 | 239 | 959 | 770 | 50.2 | 6.0 | 24.3 | 19.5 | 31 | 127.2 | 394 | 10.0 | 5.19 | ..... |
| 3 | 4,544 | 2,318 | ..... | 1,263 | 963 | 51.0 | ..... | 27.8 | 21.2 | 29 | 156.7 | 455 | 10.0 | 5.22 | ..... |
| 4 | 3,867 | 1,979 | ..... | 1,064 | 824 | 51.2 | ..... | 27.5 | 21.3 | 31 | 124.7 | 387 | 10.0 | 5.87 | ..... |

E IV.—*Details of Operation of Blast Furnace No. 1, July, 1897, to August, 1897, Inclusive.*

*Dimensions of Furnace.*

Hearth section.....42 by 180 in.
Height, center line of tuyeres to charging floor, 24 ft. 5 in.
Section at top of upper jackets..... 48 by 180 in.

average weight of charge is not recorded, but is given in another summary for Feb- 1898, as 2,000 lb. All the charges here recorded were probably of about this . On this assumption, the percentage of coke varied from 7.0 to 10.0 per cent. average number of charges per operating day, for the entire period covered by this ion, was 318.7.

| . | Number of Charges Smelted per Day. | Grade of Matte, Per Cent. Cu. | Slag. Per Cent. $SiO_2$. | Per Cent. FeO. | Per Cent. $Al_2O_3$. | Per Cent. CaO. | Oxygen Ratio. | Pounds Coke per Charge. | Number of Days Operated. | Ounces Blast Pressure. |
|---|---|---|---|---|---|---|---|---|---|---|
| ly | 229 | 47.5 | 40.2 | 30.4 | 12.5 | 13.2 | 2.5827 | 199.6 | 9½ | 35.3 |
| g. | 241.5 | 44.4 | 41.2 | 26.7 | 10.3 | 17.9 | 2.4217 | 200.0 | 17 | 34.0 |
| pt. | 266.7 | 47.4 | 40.5 | 27.1 | 11.9 | 18.2 | 2.4109 | 200.0 | 18 | 24.2 |
| t. | 323.6 | 47.6 | 39.5 | 25.3 | ..... | 20.0 | ..... | 192.0 | 23 | 40.7 |
| v. | 351.3 | 45.3 | 39.8 | 29.5 | 12.2 | 14.6 | 2.3019 | 173.4 | 18¼ | 32.0 |
| c. | 340.4 | 52.4 | 40.4 | 31.5 | 11.2 | 17.0 | 2.2479 | 152.0 | 27 | 24.3 |
| n. | 331.7 | 53.0 | 40.0 | 28.5 | 9.8 | 17.9 | 2.2524 | 141.8 | 29¾ | 27.7 |
| b. | 378.2 | 56.2 | 40.7 | 28.1 | 10.0 | 17.9 | 2.3116 | 140.6 | 25½ | 33.3 |
| ar. | 347.8 | 57.2 | 40.2 | 28.6 | 8.9 | 17.8 | 2.2273 | 158.6 | 23½ | 29.8 |
| r. | 300.0 | 57.8 | 40.1 | 27.2 | 8.9 | 20.3 | 2.1478 | 167.6 | 25 | 29.7 |
| ay | 326.0 | 58.7 | 39.0 | 33.3 | 10.5 | 15.8 | 2.1468 | 162.8 | 21 | 27.1 |
| ne | 314.9 | 56.0 | 39.8 | 31.2 | 10.2 | 14.4 | 2.3420 | 186.4 | 30 | 32.1 |
| ly | 299 | 56.2 | 42.7 | 25.5 | 9.8 | 18.8 | 2.4678 | 175.4 | 27¼ | 35.0 |
| g. | 273 | ..... | ..... | ..... | ..... | ..... | ..... | 190.2 | ..... | 30.6 |

## TABLE V.—*Details of Operation of Blast Furnace No. 1, September, October, November, December, 1898.*

All weights are Blast-Furnace Weights.

| Furnace. | No. 1. 42 by 180 in. 24 ft. 5 in. high. | | | |
|---|---|---|---|---|
| Months (1898). | September. | October. | November. | December. |
| **Composition of Charge.** | | | | |
| Total No. of Charges smelted, | 6,492 | 6,802 | 6,828 | 5,[illegible] |
| Coke, Tons. | 649.1 | 743.0 | 758.6 | 573.[illegible] |
| Lime rock, | 1,587.2 | 1,613.8 | 1,758.2 | 1,300.[illegible] |
| Conv. slag, | 1,785.2 | 1,727.4 | 1,830.8 | 1,662.[illegible] |
| Concentrates, | 273.1 | 536.8 | 604.1 | 782.[illegible] |
| First-class ore, | 2,885.8 | 2,749.6 | 2,702.4 | 1,957.[illegible] |
| Moose ore, | .... | .... | .... | .... |
| B. & M. matte, | 60.7 | 334.0 | 229.5 | 186.[illegible] |
| Colorado matte, | .... | .... | .... | .... |
| Refinery slag, | 58.4 | 72.8 | 84.1 | 92.[illegible] |
| B. F. slag and chips, | .... | .... | 9.5 | 1.[illegible] |
| Total tonnage, exclusive fuel, | 6,650.4 | 7,034.4 | 7,218.6 | 5,991.[illegible] |
| Total tonnage, exclusive fuel and flux, | 5,063.1 | 5,420.6 | 5,460.4 | 4,691.5 |
| Number of days operated, | 24¼ | 25 | 25½ | 20½ |
| Tons per operating day, | 274.2 | 281.4 | 283.1 | 292.[illegible] |
| Tons per operating day per square foot hearth, | 5.22 | 5.36 | 5.39 | 5.5[illegible] |
| Per cent. of coke, | 9.8 | 10.6 | 10.5 | 9.[illegible] |
| Height of charge col. in feet, | 13 | 15 | 15.1 | 15.[illegible] |
| Grade of matte, per cent. Cu, | 54.6 | 54.5 | 55.1 | 51.5 |
| **Slag.** | | | | |
| Per cent. $SiO_2$, | 41.2 | 41.8 | 41.2 | 40.5 |
| Per cent. FeO, | 25.4 | 23.4 | 23.5 | 24.[illegible] |
| Per cent. $Al_2O_3$, | 9.2 | 8.4 | 8.4 | 9.[illegible] |
| Per cent. CaO, | 20.5 | 22.0 | 22.8 | 22.0 |
| Oxygen ratio (base = 1, acid = ), | 2.2739 | 2.2735 | 2.1978 | 2.2235 |
| Blast pressure, Oz, | 24.7 | 24.5 | 27.5 | 22.[illegible] |

## TABLE VI.—*Details of Operation of Blast Furnaces Nos. 3 and 4, September, October, November, December, 1898.*

All Weights are Blast-Furnace Weights.

| Furnace. | Blast Furnace No. 3. 36 by 120 in. | | | | Blast Furnace No. 4. 34 by 40 in. | | | |
|---|---|---|---|---|---|---|---|---|
| Month (1898). | Sept. | Oct. | Nov. | Dec. | Sept. | Oct. | Nov. | Dec. |
| **Composition of Charge.** | | | | | | | | |
| Total number of charges smelted, | 3,943 | 4,116 | 4,523 | 4,276 | 3,717 | 3,333 | 3,751 | 3,[illegible] |
| Coke, Tons. | 396.0 | 444.8 | 444.3 | 434.3 | 372.4 | 368.3 | 376.5 | [illegible] |
| Lime rock, | 129.0 | 529.8 | 516.7 | 280.3 | 80.9 | 822.5 | 422.1 | [illegible] |
| Conv. slag, | 1,600.7 | 1,322.8 | 1,108.0 | 1,593.5 | 1,341.3 | 1,089.7 | 937.8 | 1,[illegible] |
| Concentrates, | 1,251.2 | 1,389.0 | 1,237.6 | 1,606.8 | 942.1 | 1,218.4 | 1,000.7 | 1,[illegible] |
| First-class ore, | 155.2 | 855.0 | 449.8 | ..... | 80.5 | 469.7 | 359.1 | .... |
| Moose ore, | ..... | 0.5 | 221.0 | 188.9 | ..... | 0.3 | 188.0 | [illegible] |
| B. & M. matte, | 757.7 | 237.7 | 572.1 | 561.1 | 1,150.9 | 373.4 | 658.0 | [illegible] |
| Colorado matte, | 156.2 | 2.0 | 466.0 | 326.0 | 204.1 | 2.4 | 424.9 | [illegible] |
| Refinery slag, | ..... | 1.4 | ..... | ..... | ..... | ..... | ..... | ..... |
| B. F. slag and chips, | ..... | ..... | 10.0 | 7.5 | ..... | ..... | 7.4 | [illegible] |
| Total tonnage, exclusive fuel, | 4,050.0 | 4,338.2 | 4,581.2 | 5,564.6 | 3,799.8 | 3,475.4 | 3,898.0 | 3,[illegible] |
| Total tonnage, excl. fuel and flux, | 3,921.0 | 3,808.4 | 4,064.5 | 4,283.8 | 3,718.8 | 3,153.9 | 3,475.9 | 3,[illegible] |
| Number of days operated, | 27¼ | 29½ | 27 | 28 | 26½ | 29½ | 30 | [illegible] |
| Tons per operating day, | 148.6 | 147.1 | 169.7 | 163.0 | 143.4 | 117.8 | 129.9 | [illegible] |
| Tons per op. day per sq. ft. hearth, | 4.95 | 4.90 | 5.66 | 5.43 | 6.75 | 5.54 | 6.11 | [illegible] |
| Per cent. of coke, | 9.8 | 10.3 | 9.7 | 9.5 | 9.8 | 10.6 | 9.7 | [illegible] |
| Grade of matte, per cent. Cu, | 51.8 | 53.0 | 53.9 | 51.0 | 52.1 | 51.5 | 55.1 | [illegible] |
| **Slag.** | | | | | | | | |
| Per cent. $SiO_2$, | 39.0 | 41.4 | 42.2 | 40.2 | 37.3 | 41.5 | 41.5 | [illegible] |
| Per cent. FeO, | 41.8 | 33.3 | 30.6 | 36.8 | 44.3 | 34.3 | 31.3 | [illegible] |
| Per cent. $Al_2O_3$, | 10.1 | 8.6 | 9.4 | 10.1 | 10.3 | 8.9 | 9.5 | [illegible] |
| Per cent. CaO, | 4.9 | 12.7 | 13.1 | 11.0 | 4.7 | 11.8 | 13.4 | [illegible] |
| Oxygen ratio (base = 1, acid = ), | 2.3738 | 2.3545 | 2.5374 | 2.2983 | 2.1964 | 2.3791 | 2.4513 | [illegible] |
| Blast pressure, Oz., | 16.7 | 18.6 | 17.2 | 14.4 | 12.8 | 13.7 | 12.6 | [illegible] |

TABLE VII.—*Details of Operation of Blast Furnace No. 3 (72 by 180 in.) for the Entire Period of its Use.*

All Weights are Blast-Furnace Weights.

| Months, 1899. | April. | May. | June. | July. | August. | Sept. | Oct. | Nov. | Dec. |
|---|---|---|---|---|---|---|---|---|---|
| number of charges | 5,242 | 10,557 | 1,743 | 11,737 | 13,464 | 11,714 | 8,100 | 9,560 | 8,180 |
| Coke. Tons. | 449.2 | 992.1 | 187.8 | 1,093.9 | 1,283.8 | 1,109.0 | 801.4 | 969.3 | 9,100 |
| Lime rock | 1,272.8 | 2,525.5 | 447.2 | 2,876.3 | 3,007.4 | 2,764.2 | 1,728.3 | 2,491.1 | 2,036.3 |
| Convertor slag | 1,282.5 | 3,021.1 | 559.9 | 3,135.7 | 3,398.2 | 2,512.3 | 1,727.0 | 2,107.2 | 1,852.6 |
| First-class ore | 1,397.4 | 2,387.1 | 318.5 | 3,879.1 | 4,128.8 | 3,979.9 | 2,285.8 | 3,366.4 | 2,450.5 |
| Moose ore | 19.0 | 115.2 | ..... | 306.7 | 207.6 | ..... | ..... | ..... | 73.3 |
| Coarse concentrates | 1,340.5 | 2,988.5 | 535.2 | 1,891.4 | 3,020.2 | 2,428.1 | 2,106.3 | 1,765.5 | 1,922.5 |
| B. and M. matte | ..... | ..... | ..... | ..... | ..... | ..... | 194.0 | 50.0 | 24.0 |
| Refinery slag | 32.0 | 62.8 | 10.8 | ..... | ..... | ..... | ..... | ..... | ..... |
| Calcine lump | 2.1 | 3.4 | ..... | ..... | ..... | ..... | ..... | 25.5 | 10.6 |
| Mud *a* | ..... | ..... | ..... | ..... | ..... | ..... | 8.7 | 20.0 | ..... |
| Reverb. chips | ..... | ..... | ..... | ..... | 18.8. | 115.4 | ..... | ..... | ..... |
| tonnage, exclusive fuel | 5,346.3 | 11,103.6 | 1,771.6 | 12,089.2 | 13,790.0 | 11,799.9 | 8,050.1 | 9,825.7 | 8,879.8 |
| tonn., excl. fuel and flux | 4,073.5 | 8,578.1 | 1,324.4 | 9,212.0 | 10,782.6 | 9,035.7 | 6,321.8 | 7,334.6 | 6,343.5 |
| er of days operated | 14 | 31 | 7 | 28¾ | 31 | 30 | 24¾ | 29¼ | 30 |
| per operating day | 381.9 | 358.2 | 253.1 | 420.5 | 444.8 | 393.3 | 325.3 | 333.1 | 279.3 |
| per op. day per sq. ft. rth | 4.24 | 39.8 | 2.81 | 4.67 | 4.94 | 4.37 | 3.61 | 3.70 | 3.10 |
| ent. of coke | 8.4 | 8.9 | 10.6 | 9.0 | 9.3 | 9.4 | 10.0 | 9.9 | 10.9 |
| t of chg. col. ab. tuyers | 12.7 ft. | 13.2 ft. | 13.4 ft. | 14 ft.*b* | 14 ft.*b* | 14.3 ft.*b* | No Rec. | No Rec. | No Rec. |
| of matte, per cent. Cu. | 48.6 | 48.4 | 53.0 | 51.1 | 48.8 | 49.5 | 49.6 | 52.7 | 54.2 |
| er cent. $SiO_2$ | 42.9 | 42.7 | 40.8 | 41.9 | 42.7 | 41.9 | 42.6 | 42.1 | 42.3 |
| er cent. FeO | 21.0 | 22.0 | 23.8 | 23.2 | 24.5 | 25.8 | 27.3 | 24.1 | 25.8 |
| er cent. $Al_2O_3$ | 10.1 | 10.1 | 10.0 | 10.0 | 10.1 | 10.0 | 10.1 | 10.1 | 10.1 |
| er cent. CaO | 22.4 | 22.3 | 22.8 | 22.1 | 20.3 | 19.8 | 18.0 | 22.5 | 21.1 |
| en ratio (base=1, acid=) | 2.4837 | 2.4298 | 2.2305 | 2.3473 | 2.4342 | 2.3617 | 2.4384 | 2.2954 | 2.3305 |
| pressure, Oz. | 28 | 35 | 34 | 30 | 36 | 34 | 31 | 31.6 | 33.3 |

Probably flue dust, specially prepared for blast-furnace treatment.
Record incomplete.

BLE VIII.—*Details of Operation of Blast Furnace No. 3 (44 in. by 180 in.), January, February, March, 1900.*

All Weights are Blast-Furnace Weights.

| Months—1900. | January. | February. | March. |
|---|---|---|---|
| l No. of charges smelted, | 4,540 | 6,410 | 8,504.0 |
| coal, | 48.7 | 46.7 | 74.0 |
| coke, | 384.5 | 602.1 | 786.8 |
| Lime rock, Tons | 1,209.5 | 1,223.0 | 2,191.5 |
| Converter slag. | 932.3 | 1,816.6 | 1,816.2 |
| Coarse concentrates, | 985.1 | 1,844.9 | 1,766.7 |
| First-class ore, | 1.496.7 | 1,550.7 | 2,830.9 |
| Moose ore, | 7.0 | 24.5 | 71.3 |
| B. and M. matte, | 21.7 | .... | 39.0 |
| Refinery slag, | 2.2 | .... | .... |
| Calcine lumps, | .... | .... | 3.5 |
| l tonnage, exclusive fuel, | 4,654.5 | 6,459.7 | 8,719.1 |
| l tonnage, exclusive fuel and flux, | 3,445.0 | 5,236.7 | 6,527.6 |
| ber of days operated, | 14 | 28 | 30¼ |
| per operating day, | 332.5 | 230.7 | 283.5 |
| per operating day per sq. ft. hearth, | 6.05 | 4.19 | 5.15 |
| ent. of fuel, | 9.3 | 10.0 | 99 |
| t pressure, ounces, | 29.6 | 33.4 | 56 |
| le of matte, per cent. Cu, | 52.6 | 51.1 | 493 |
| Per cent. $SiO_2$ | 40.8 | 41.4 | 42.2 |
| Per cent. FeO | 22.9 | 27.7 | 21.7 |
| Per cent. $Al_2O_3$ | 9.7 | 9.8 | 9.9 |
| Per cent. CaO | 22.3 | 17.7 | 20 9 |
| Oxygen ratio (base = 1; acid = ), | 2.2345 | 2.3666 | 2.5032 |

TABLE IX.—*Details of Operation of Blast Furnace No. 2 (56 by 180 in.) for each quarter-year, February, 1899, to September, 1900, inclusive.*

**All Weights are Blast-Furnace Weights.**

| | Period. | 1899 February-March. | 1899 April-June. | 1899 July-September. | 1899 October-December. | 1900 January-March. | 1900 April-June. | 1900 July-September. |
|---|---|---|---|---|---|---|---|---|
| | Total tons coke | 1,434.6 | 2,764.0 | 2,457.5 | 2,775.4 | 2,679.1 | 2,064.1 | 2,768.4 |
| | Total tons coal | ..... | ..... | ..... | ..... | 219.4 | 136.9 | ..... |
| | Per cent. fuel | 8.0 | 8.3 | 9.3 | 9.5 | 9.8 | 9.2 | 8.4 |
| Composition of Charge. | Total tons lime rock | 4,124.0 | 8,304.6 | 6,187.0 | 7,278.3 | 7,248.2 | 6,014.8 | 8,073.3 |
| | Total tons conv. slag | 4,387.5 | 7,993.2 | 6,245.6 | 5,904.2 | 6,413.2 | 4,585.2 | 7,196.7 |
| | Total tons first-class ore | 4,863.5 | 8,744.9 | 8,610.8 | 9,562.1 | 9,176.5 | 7,470.4 | 8,873.6 |
| | Total tons second-class ore | 113.9 | ..... | ..... | ..... | ..... | ..... | ..... |
| | Total tons Moose ore | ..... | 118.3 | 423.4 | 91.2 | 252.6 | 413.0 | 355.4 |
| | Total tons coarse concentrates | 4,122.9 | 8,096.2 | 4,819.0 | 6,032.5 | 6,554.3 | 4,770.0 | 7,141.3 |
| | Total tons B. & M. matte | ..... | ..... | 25.0 | 287.7 | 26.0 | 177.0 | 74.0 |
| | Total tons refinery slag | 231.0 | 144.1 | ..... | ..... | 29.9 | ..... | 14.2 |
| | Total tons blast furnace chips | 32.1 | ..... | ..... | ..... | ..... | ..... | ..... |
| | Total tons reverb. chips | ..... | ..... | 94.1 | ..... | ..... | ..... | ..... |
| | Total tons lump calcine | ..... | 8.1 | 8.5 | 26.2 | 0.9 | ..... | ..... |
| | Total tons refinery brick | ..... | ..... | ..... | ..... | ..... | 1.8 | ..... |
| | Total tons briquettes | ..... | ..... | ..... | ..... | ..... | 523.1 | 1,379.8 |
| | Total tons mud *a* | ..... | ..... | ..... | 22.1 | ..... | ..... | ..... |
| | Number of operating days | 49¼ | 75¾ | 67½ | 83¼ | 82½ | 63 | 83 |
| | Tons per operating day, exclusive fuel | 363.3 | 441.0 | 391.3 | 350.1 | 360.0 | 380.2 | 398.8 |
| | Tons per operating day, excl. fuel and flux | 279.6 | 331.4 | 299.7 | 262.8 | 272.1 | 284.8 | 301.6 |
| | Tons per operating day, per square ft. hearth | 5.19 | 6.30 | 5.59 | 5.00 | 5.14 | 5.43 | 5.70 |
| | Total number of charges | 17,197 | 32,038 | 25,989 | 28,657 | 28,991 | 23,287 | 31,368 |
| | Average weight per charge | 2,081 | 2,085 | 2,033 | 2,034 | 2,049 | 2,057 | 2,111 |
| | Blast pressure, oz | 25.7 | 34.7 | 33.3 | 32.8 | 36.5 | 33.9 | 30.8 |
| | Height of charge column, feet | 12.9 | 13.6 | ..... | ..... | ..... | ..... | ..... |
| | Grade of matte, per cent | 51.3 | 50.3 | 50.1 | 51.9 | 51.7 | 52.7 | 53.9 |
| Slag. | Per cent. $SiO_2$ | 40.4 | 52.4 | 42.3 | 42.4 | 41.6 | 42.2 | 41.5 |
| | Per cent. FeO | 24.2 | 21.1 | 23.8 | 24.8 | 23.7 | 21.1 | 22.4 |
| | Per cent. $Al_2O_3$ | 10.0 | 10.1 | 10.1 | 10.1 | 10.0 | 9.5 | 8.7 |
| | Per cent. CaO | 21.9 | 23.6 | 21.4 | 21.5 | 20.8 | 22.8 | 22.9 |
| | Oxygen ratio (base = 1, acid =) | 2.2442 | 2.3797 | 2.3848 | 2.3348 | 2.3875 | 2.3946 | 2.2659 |

*a* Probably flue dust, specially prepared for blast-furnace treatment.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.
[SUBJECT TO REVISION.]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the e meeting, Aug. 18 to 21, 1913, when an abstract of the paper will be read. If this is impossible, discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 t 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its ior. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1913, when XLVI. of the *Transactions* will go to press. Any discussion offered thereafter should preferably n the form of a new paper for publication in Vol. XLVII. (with suitable cross references in both imes).

# creasing the Efficiency of MacDougall Roasters at the Great Falls Smelter of the Anaconda Copper Mining Co.

BY FRANK R. CORWIN AND SELDEN S. RODGERS, GREAT FALLS, MONT.

(Butte Meeting, August, 1913.)

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## I. INTRODUCTION.

SINCE the first installation of MacDougall roasters at the Great Falls elter of the Anaconda Copper Mining Co., the capacity of the furces has been more than doubled. During the first nine or ten years the operation of the MacDougall department the tonnage treated by e furnaces remained practically unchanged. Experimental work rried out in 1906, however, showed that increasing the draft of the rnaces would bring about an increase in capacity. In 1909 the acDougall department was connected to the new smelter stack and

flue system, and the stronger draft caused by this change increased the capacity of the furnaces. The change in draft conditions made it necessary to operate differently, and systematic experimental work was started to adapt the furnaces to the new conditions. This experimental work extended over a period of several years, and resulted in very largely increasing the capacity of the MacDougall furnaces, decreasing the percentage of flue dust, and also brought about other changes and improvements tending to raise the efficiency of the MacDougall department. These changes, of course, resulted in reducing operating expenses. The improvement was brought about without building any new MacDougalls or enlarging the old ones, and was the result of the thorough study of the MacDougalls and numerous tests.

It is the purpose of this article to give a brief account of the experimental work and to show how the increase in efficiency was brought about.

Part of the experimental work described in this paper was carried out by the writers, while in the employ of the Anaconda Copper Mining Co., under the direction of M. W. Krejci, Metallurgist, and Arthur Crowfoot, Chief Sampler. Complete records of all of the MacDougall testing work have been placed at our disposal through the courtesy of A. E. Wheeler, Superintendent.

## II. Brief History of Present MacDougall Plant.

The first change from the Brueckner cylinders used for roasting concentrates at Great Falls was the construction of a MacDougall roaster of the Herreshoff type. This furnace was first started on June 15, 1898. It had eight hearths, was lined with fire brick, and had an air-cooled central shaft, but no provision was made for cooling the rabble arms. It was operated for a while, and accomplished the work of roasting, but ran so hot that the rabble arms became twisted and distorted out of shape. To prevent the furnace from becoming too hot a device was then installed to draw the heated gases away from the hearths. This consisted of a pipe connected to alternate hearths of the MacDougall furnace and also connected to the flue. It was equipped with dampers, so that the hot gases could be drawn away from any one of the alternate hearths directly into the flue.

This first furnace not proving altogether a success, the cooling device was eliminated and the furnace shut down to be reconstructed. When rebuilt it was what is known as the Evans-Klepetko type of MacDougall furnace, having six hearths and water-cooled central shaft and rabble arms. Another MacDougall furnace was built at the same

These two furnaces are still in use, and are Nos. 8 and 9, re-
vely, of the present 22 MacDougall furnaces. Except for a few
changes, such as substituting solid arms for the water-cooled
on the first hearth, the removal of the branch pipes at first used
of the rabble arms to distribute the cooling water, and changes
ign of the rakes, rake frames and rabble arms, they still remain
ially the same as when built in 1899. Six more furnaces, now
n as Battery III., were built and started in November, 1899. In
pring of 1900 four more MacDougall furnaces were constructed,
ng with Nos. 8 and 9 the present Battery II. of six furnaces.
01, Battery IV., consisting of six furnaces, was built, and in
ary, 1903, the four furnaces comprising Battery I., these last
urnaces completing the equipment of 22 MacDougall furnaces.

## I. First Experimental Work on Increasing Capacity of Furnaces.

e composition of the concentrate treated when the MacDougall
was first started was approximately as follows:

| u. Cent. | $SiO_2$. Per Cent. | Fe. Per Cent. | $Al_2O_3$ Per Cent. | Zn. Per Cent. | S. Per Cent. |
|---|---|---|---|---|---|
| 3.3 | 24.2 | 25.6 | 4.6 | 0.9 | 30.0 |

hen first built, the MacDougall furnaces had a rated capacity of
ns of raw wet concentrate containing about 35 per cent. of sul-
roasted down to 7 per cent. of sulphur, per furnace per day.
tual practice, however, the capacity of the furnaces fell some-
below this, averaging roughly around 35 tons of total cupreous
ial per furnace per day. The first experimental work looking
ds an increase in the capacity of the furnaces was carried out by
Klepinger, in June and July, 1906, the primary object of the
made by Mr. Klepinger being to determine whether an increase
draft of the MacDougalls would increase their capacity. These
were made previous to the building of the new smelter stack.
cDougall furnace No. 2 was selected for the test, on account of
vorable location. To increase the draft on No. 2 furnace, a
Sturtevant exhauster was connected to the top of the furnace,
own in the accompanying sketch from Mr. Klepinger's report,
1), the gas being discharged into the main MacDougall cross
When the exhauster was in use the slides in the regular draft
were closed. When for any reason the exhauster was stopped
slides were drawn out, allowing the furnace to operate under

Sketches of the old and new flue system are shown in Figs. 2 and 3, respectively. The change in draft conditions for the MacDougall department is indicated by the following figures:

*Draft in Inches of Water, MacDougall Furnace.*

| | Flue System. | |
|---|---|---|
| | Old. | New. |
| MacDougall dust chamber, . . . . . | 0.255 | 0.9 |
| Flue necks from MacDougall furnaces to cross flue, . . . . . | 0.139 | 0.93 |

Using the same feed as under the old draft conditions, the first effect of the stronger draft was to put out all of the MacDougall furnaces. The reason for this was that, treating the same feed as formerly, the amount of heat generated by the oxidation of the sulphur remained practically unchanged; whereas, due to the increased draft, more air was drawn through the furnaces, more heat lost in the escaping gases, and consequently the furnaces became cold and went out. The first remedy tried was decreasing the draft by partly closing the dampers in the crosstake flue. After doing this the MacDougall department ran satisfactorily, but partly closing the dampers in the crosstake flue also decreased the draft for all the other smelter departments. After a few days, therefore, the dampers in the crosstake flue were again opened wide and the MacDougall draft decreased by partly closing the slides in the flue necks from the individual MacDougall furnaces. Also, the resistance to the passage of air through the furnaces was increased by covering one or two of the side drop holes on the second and fourth hearths with iron plates, reducing the drop hole area on these floors. As the foremen and furnacemen became familiar with operating the furnaces under the new conditions, the feed was gradually increased and more use made of the available draft. The increased capacity of the furnace under an increased draft, predicted by Mr. Klepinger's experimental work in 1906, was more than realized, as is shown by the following figures, taken from the monthly smelter reports. These figures show, for the MacDougall department, the tons treated per furnace day, tons treated per square foot of hearth area, and the average per cent. of sulphur in the calcine, for the 10 months immediately preceding and the 10 months immediately following the change to the new flue system.

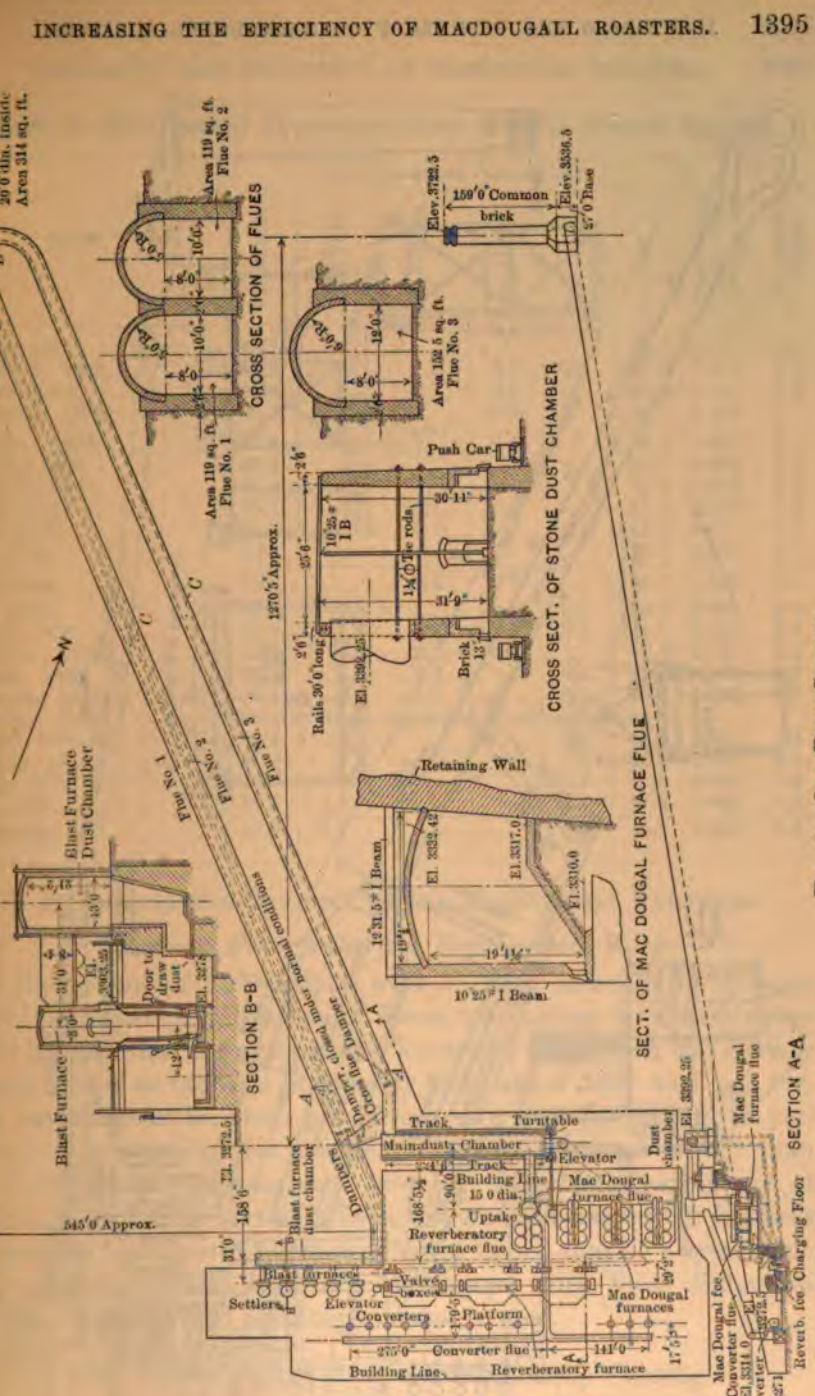



FIG. 2.—OLD FLUE SYSTEM.

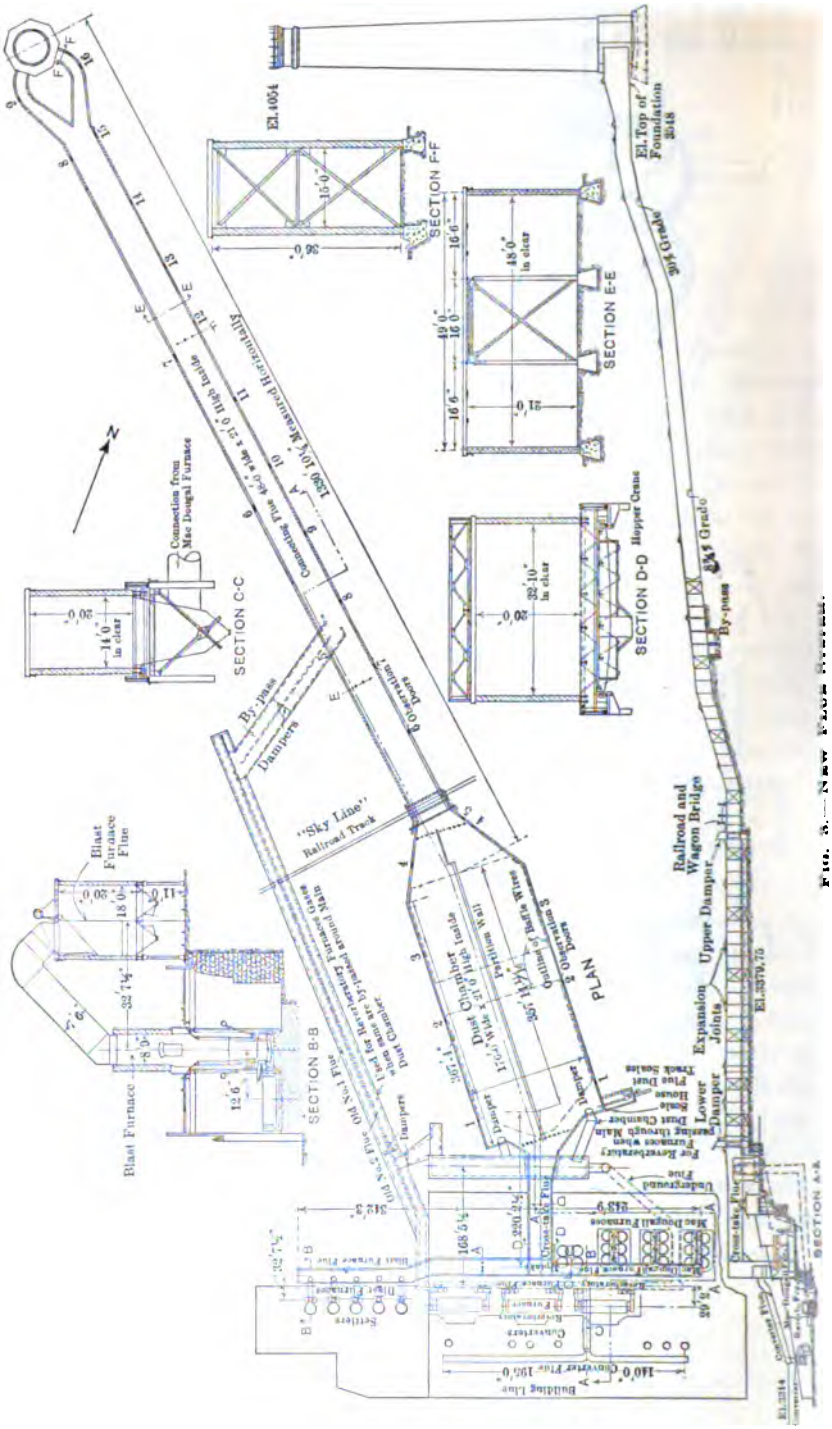



FIG. 3.—NEW FLUE SYSTEM.

*Data on MacDougall Department from Monthly Smelter Reports.*

| Month. | Tons Treated per Furnace Day. | | | | | | Total Tons per Square Foot of Hearth Area. | Per Cent S in Calcine. |
|---|---|---|---|---|---|---|---|---|
| | Concentrate. | Screenings. | Dried Slimes. | Flue Dust. | Limestone. | Total. | | |
| st, 1908, | 28.7 | ... | ... | ... | ... | 28.7 | 0.032 | 9.1 |
| nber, 1908, | 43.4 | ... | ... | 0.1 | 0.1 | 43.6 | 0.046 | 9.1 |
| er, 1908, | 41.4 | ... | 0.1 | 0.9 | ... | 42.4 | 0.045 | 8.9 |
| nber, 1908, | 36.2 | ... | ... | 0.3 | ... | 36.5 | 0.039 | 7.7 |
| nber, 1908, | 37.8 | ... | (Copper Precip. 0.3) | 0.1 | ... | 38.2 | 0.041 | 8.1 |
| ry, 1909, | 33.1 | ... | ... | ... | ... | 33.1 | 0.036 | 8.8 |
| ary, 1909, | 35.4 | ... | ... | ... | ... | 35.4 | 0.038 | 8.2 |
| 1, 1909, | 35.1 | ... | 0.1 | ... | ... | 35.2 | 0.038 | 8.4 |
| 1909, | 34.3 | ... | 0.2 | ... | 0.01 | 34.5 | 0.037 | 9.2 |
| 1909, | 3.33 | ... | ... | ... | ... | 33.3 | 0.036 | 9.1 |
| verage, | 35.9 | ... | 0.2 | 0.1 | ... | 36.1 | 0.039 | 8.7 |
| 1909, | 49.7 | ... | ... | ... | 0.5 | 50.2 | 0.053 | 9.7 |
| st, 1909, | 40.8 | ... | 0.7 | ... | 0.4 | 41.9 | 0.046 | 8.6 |
| nber, 1909, | 41.6 | ... | 2.1 | 0.1 | ... | 43.8 | 0.047 | 8.3 |
| er, 1909, | 40.2 | ... | 2.4 | ... | 0.3 | 42.9 | 0.046 | 8.2 |
| nber, 1909, | 37.3 | ... | 2.0 | ... | 0.2 | 39.5 | 0.042 | 7.7 |
| nber, 1909, | Smelter down on account of railroad strike. | | | | | | | |
| ry, 1910, | 49.1 | ... | ... | ... | 0.5 | 49.6 | 0.053 | 8.2 |
| ary, 1910, | 51.9 | ... | ... | ... | 1.7 | 53.6 | 0.057 | 8.7 |
| 1, 1910, | 40.5 | ... | ... | ... | 2.3 | 42.8 | 0.046 | 7.6 |
| 1910, | 44.8 | 1.2 | ... | ... | 1.2 | 47.2 | 0.050 | 7.9 |
| verage, | 44.0 | 0.1 | 0.8 | ... | 0.8 | 45.7 | 0.049 | 8.3 |

## V. Increasing the Capacity of the MacDougall Furnaces.

### *Conditions Prior to Starting Experimental Work.*

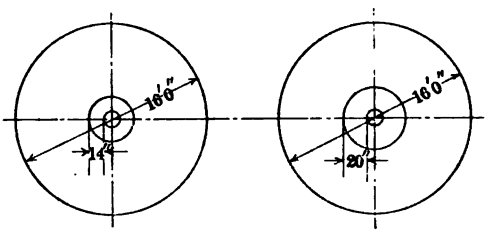
the spring of 1910 efforts were begun to increase the tonnage ed per MacDougall furnace. From the figures tabulated above ll be seen that the new smelter stack and flue system had already ght about an increase in capacity from about 36.1 tons per fur- day to about 45.7 tons per furnace day, without raising the per- age of sulphur in the calcine. Treating tonnages higher than t 50 tons per furnace day, however, raised the percentage of hur in the calcine an undesirable amount and caused the furnaces rust up heavily. Nevertheless it was hoped that by a systematic y of the conditions governing MacDougal work these difficulties d be overcome and a higher tonnage treated.

### (*A*). *Test of Arthur Crowfoot on Increasing Capacity.*

). *Description of Test.*—The first step towards increasing the capac- f the furnaces was a test carried out under the direction of Arthur wfoot, Chief Sampler, during the period from May 3 to May 7,

inclusive, 1910. Four furnaces were selected for the test work, Nos. 1 and 3 operating as a unit and Nos. 18 and 19 operating as a unit. The essential differences between the two units were as follows:

1. The central drop hole connecting the third hearth with the fourth hearth in furnaces Nos. 1 and 3 measured approximately 20 in. from the shaft to the edge of the drop hole, while in furnaces Nos. 18 and 19 this opening measured approximately 14 in. The standard size for this third hearth drop hole had always been 14 in.; but a short time previous to starting this test the third hearth drop holes on furnaces Nos. 1, 2, 3, and 4 had been increased to 20 in., adopted from the practice at the Washoe Smelter at Anaconda.




2. Speed of Rabble Arms. The rabble arms in furnaces Nos. 1 and 3 made one revolution in 50 sec., while the rabble arms in furnaces Nos. 18 and 19 made one revolution in 37 sec. The regular speed at which furnaces Nos. 18 and 19 ran, before starting the test, was one revolution in 53 sec., but this was increased to one revolution in 37 sec. about two weeks before starting the test.

The instructions for carrying out the tests were to start all furnaces on a fine cencentrate feed of about 55 tons wet weight per 24 hr. per furnace, continuing this feed for 24 hr. on each furnace. After the first 24 hr., the rate of feed to each furnace was to be increased 5 tons per 24 hr. and this rate of feed kept up for 24 hr., and so on, increasing the rate of feed by 5-ton increments until reaching a rate of feed of 75 tons of wet concentrate per furnace per 24 hr.

The information obtained from the test was of considerable interest. A fractional part of the total data obtained is tabulated below.

*Tonnage Test on MacDougall Furnaces.*

(*A*). *Average results obtained from Nos.* 1 *and* 3 *Furnace Unit.*

| Date of Test, 1910. | May 3. | May 4. | May 5. | May 6. | May 7. |
|---|---|---|---|---|---|
| inal tons wet conct. to be fed per 24 hr., | 55 | 60 | 65 | 70 | 75 |
| l tons wet conct. fed per 24 hr., | 52.72 | 59.05 | 64.54 | 69.17 | 59.18 |
| al rate of feed, tons per 24 hr., | 58.80 | 66.13 | 71.85 | 80.18 | 94.54 |
| rs furnace in operation, | 21.51 | 21.30 | 21,71 | 20.70 | 15.79 |
| rs furnace down, | 2.49 | 2.7 | 2.29 | 3.30 | 8.21 |
| cent. of 24 hr. furnace down, | 10.37 | 11.25 | 9.54 | 13.75 | 34.20 |
| t reading on flue necks, inches water, | 1.28 | 1.26 | 1.23 | 1.3 | 1.36 |
| Concentrate Fed. | | | | | |
| cent. moisture, | 7 5 | 8.0 | 6.7 | 7.6 | 7.1 |
| y per cent. Cu, | 8.80 | 7.55 | 8.20 | 8.4 | 8.45 |
| y per cent. S, | 36.0 | 32.8 | 34.6 | 34.4 | 35.8 |
| Calcine Produced. | | | | | |
| y per cent. Cu, | 10.55 | 9.75 | 10.35 | 10.1 | 10.25 |
| y per cent. S, | 8.2 | 9.2 | 11.3 | 12.2 | 12.6 |

(*B*). *Average results obtained from Nos.* 18 *and* 19 *Furnace Unit.*

| | | | | | |
|---|---|---|---|---|---|
| inal tons wet conct. to be fed per 24 hr., | 55 | 60 | 65 | 70 | 75 |
| l tons wet conct. fed per 24 hr., | 52.45 | 58.31 | 60.91 | 59.41 | 63.01 |
| al rate of feed, tons per 24 hr., | 64.20 | 72.72 | 71.40 | 74.16 | 79.84 |
| rs furnace in operation, | 19.58 | 19.24 | 20.49 | 19.25 | 18.92 |
| rs furnace down, | 4.42 | 4.76 | 3.51 | 4.75 | 5.08 |
| cent. of 24 hr. furnace down, | 18.42 | 19.83 | 14.60 | 19.79 | 21.17 |
| t reading on flue necks, inches water, | 1.16 | 1.09 | 1.09 | 1.12 | 1.16 |
| Concentrate Fed. | | | | | |
| cent. moisture, | 8.4 | 6.6 | 6.6 | 7.1 | 7.1 |
| y per cent. Cu, | 6.40 | 8.10 | 8.25 | 8.65 | 8.0 |
| y per cent. S, | 32.8 | 33.4 | 34.8 | 35.0 | 35.4 |
| Calcine Produced. | | | | | |
| y per cent. Cu, | 10.35 | 9.65 | 9.85 | 9.95 | 10.1 |
| y per cent. S, | 8.4 | 7.8 | 8 8 | 9.6 | 10.4 |

*b*). *Discussion of Results Obtained from Test.*—The most interesting clearly brought out by this test was that increasing the feed to furnaces did not increase the percentage of sulphur in the calcine much as expected. In fact, the 18-19 furnace unit with the smaller rd hearth drop hole and greater speed of revolution of the cen- l shaft produced very satisfactory calcine. Previously it had been ught that, under a given draft, the number of pounds of sulphur 24 hr. which a furnace would eliminate was, very approximately, constant, and that on increasing the feed, and consequently the unds of sulphur per 24 hr. delivered to a furnace, the excess of phur fed would nearly all go into the calcine. The test showed that increasing the feed of concentrate more pounds of sulphur were

burned per minute and per square foot of hearth area, and the furnaces ran much hotter. In fact, the main difficulties experienced were on account of too high a temperature being developed. The unusually large amount of heat generated caused the drop holes to become built up, and heavy accretions to form on the roofs. The conclusions drawn from the test were that a high tonnage feed of straight concentrate, under the conditions existing at that time, caused the furnaces to become too hot for practical operation.

(1). Third Hearth Center Drop Hole.—Below are tabulated data on the amount of time lost by the two furnace units:

| Nominal Tons Treated. | Percentage of Time Down in 24 Hr. | | |
|---|---|---|---|
| | Furnaces 1 and 3. | Furnaces 18 and 19. | Differences. |
| 55, . . . . . | 10.37 | 18.42 | 8.05 |
| 60, . . . . . | 11.25 | 19.83 | 8.58 |
| 65, . . . . . | 9.54 | 14.60 | 5.06 |
| 70, . . . . . | 13.75 | 19.79 | 6.04 |
| 75, . . . . . | 34.20[a] | 21.17 | ..... |

[a] About 23 per cent. due to mechanical breakdown of furnace No. 3.

From the above tabulation it will be seen that the 18–19 furnace unit with the 14-in. third hearth central drop lost on an average about 7 per cent. more time than the 1–3 furnace unit with the larger third hearth central drop hole. The reason for this was that the smaller third hearth drop hole, on account of the higher velocity of the gases and the concentration of heat at this point, caused heavy accretions to build up on the roof above this drop hole. Practically all of the additional time lost by the 18–19 furnace unit over the 1–3 furnace unit was due to barring accretions from the roof of the third hearth.

(2). Speed of Rabble Arms.—The observations showed that in treating high tonnages, increased speed was advantageous, for the reason that it kept shallower beds on the furnace hearths, and there was less banking up of material in front of the rabble arms.

*General Discussion of Data Obtained from Above Test.*—The data obtained from this test outlined the way for future work. It was necessary to have the furnaces run cooler, which could be accomplished by either drawing more air through the furnaces and consequently discharging more heat in the waste gases, or else by decreasing the sulphur content (fuel value) of the feed. Also, for treating high tonnages, the test indicated that the larger size of drop hole and increased rate of speed were in all probability desirable.

*(B). Investigation Carried Out by John A. Church, Jr.*

mediately following the above test, John A. Church, Jr., was led to make a special investigation looking towards improving vork of the MacDougall department and increasing the tonnage ed per furnace. At this point it should be noted that credit for xcellent results which were obtained from this investigation is both to Mr. Church and to George S. Crouse, calcine foreman. Crouse has been employed in the Calcine Department since De- er, 1893, working on the Brueckner cylinders before the first llation of MacDougall furnaces; and has been foreman since . He has always gladly co-operated with the Testing Depart- in its MacDougall work. The writers in particular are indebted r. Crouse, who assisted them in their later work and generously d at their disposal the information obtained from his long expe- e and keen interest in MacDougall work.

r. Church began his investigation by observations on MacDougall ace No. 19. Acting on the information obtained from Mr. Crow- s test, this furnace was operated on a high tonnage, but the fuel e of the feed was lowered by mixing screenings and lime rock the concentrate. The feed treated was made up of 20 tons of entrate and 6 tons of foreign material per shift. By "foreign rial" is meant any constituent of the feed lower in fuel value the concentrate, such as slime or first-class ore screenings. Shifts ll 8 hr. The furnace had a small third hearth drop hole, meas- 14 in. from the shaft to the edge of the drop hole, and ran at high l, the rabble arms making one revolution in 38 sec. The draft regulated by dampers in the flue necks, and by closing all doors ot one on the sixth hearth. The furnace handled the 20 tons of entrate per shift satisfactorily and the foreign material mixed the concentrate kept the temperature of the furnace from becom- oo high. It was found, however, that the work of keeping the clean above the third hearth drop hole was excessive for normal ing.

n May 17, therefore, MacDougall furnace No. 18 was equipped an enlarged third hearth drop hole and started for parallel com- on with furnace No. 19. The conditions under which the two aces ran were as shown below:

| Conditions. | Furnace No. 18. | Furnace No. 19. |
|---|---|---|
| of one revolution, . . . . . | 38 sec. | 38 sec. |
| f third hearth drop hole (shaft to e of drop hole), . . . . . | 20 in. | 14 in. |
| inches water, . . . . . | 0.8 to 0.9 | 0.8 to 0.9 |
| ge of concentrate, . . . . . | 20 per shift. | 20 per shift. |
| ge of foreign material, . . . . | As much as furnace would take. | 7–8 tons per shift. |

Both furnaces had two of the six drop holes covered on the second hearth, and two closed on the fourth hearth. Two doors were left open on the sixth hearth.

A few days of operation under the above conditions showed that furnace No. 18 with the larger third hearth drop hole ran considerably cooler and crusted up less than furnace No. 19. Also furnace No. 18 would not take as large a tonnage of foreign material without becoming too cold. Although by means of feeding foreign material along with the concentrate the temperature of No. 19 furnace as a whole could be maintained at about the right point, on account of the concentration of heat at the third hearth drop hole and the high velocity of the gases at this point, the work of keeping the roof above the third hearth clean still continued to be excessive. This test therefore corroborated the earlier work of Mr. Crowfoot, and demonstrated for high tonnages the advantages of the larger third hearth drop hole. Accordingly, when on May 24 a leak in the second hearth made it necessary to shut down furnace No. 19 for repairs, advantage was taken of the shut down to enlarge the third hearth drop hole to 20 in.

While enlarging the third hearth drop hole of a furnace greatly reduces the work of keeping the furnace clean, it also has the undesirable effect of making the calcine run higher in sulphur. A year or two later a means was found of retaining the smaller drop hole, and yet avoiding excessive incrusting on the third hearth roof. Two extra drop holes were cut in the third hearth close to the edge of the center drop hole. The inner rakes on the third hearth rabble arms were set so as to push the material away from the center, and therefore all material passing from the third to the fourth hearths was forced to pass through the two extra drop holes. This meant dropping a large bulk of material four times in every revolution of the central shaft, instead of having the material showering down continuously over the edge of the center drop hole. It reduced the amount of incrusting on the third hearth roof and localized what incrustations formed over the extra drop holes, making the work much easier for the furnacemen. The 14-in. center drop hole combined with the two extra drop holes is the present practice for the third hearth. It is mentioned here because, although during the work in the spring of 1910 to increase the tonnage per furnace it was found that the 20-in. was an improvement over the 14-in. third hearth drop hole, later work as noted above has made it desirable to again change back to the smaller drop hole.

Returning to Mr. Church's investigation, while furnace No. 19 was

n for repairs and to have the third hearth drop hole enlarged, ob-
ations were kept up on furnace No. 18. More doors were gradu-
opened on the sixth hearth until finally the furnace was run with all
of the sixth hearth doors open. (The doors on the upper
ths are always kept closed except when they must be opened for
ning, or pulling lumps out of the furnace.) The opening of each
essive door brought about an improvement in the condition of the
ace, showing that previously not enough oxygen had been sup-
l. Similarly, the dampers in the flue necks were opened wider,
roving the work of the furnace, until in time the full available
t of the lateral flue was being used, about 1.2 in. of water.

he chief difficulties experienced with No. 18 were due to the sen-
eness of the furnace. Changes in the quality of the feed, irreg-
feeding of foreign material, and other adverse conditions would
n cool the furnace too far, necessitating the use of the oil machine
restore a normal working temperature. The first improvement
aintaining the furnace at a steady temperature was made by clos-
one additional second hearth drop hole. This made three second
th drop holes closed in all. At first it was thought that the in-
sed amount of heat obtained by decreasing the second hearth drop
area was due to increasing the frictional resistance to the passage
as through the furnace, allowing less air to enter the furnace
less heat to be carried away in the waste gases. The improve-
t which had been made in the work of the furnace, however, by
ning more doors on the sixth hearth and opening wide the dampers
he flue necks, showed that this was not the case. The drop hole
on the second hearth, even with three drop holes covered, was
e enough not to throttle to any appreciable extent the gases trav-
g through the furnace. The effect, therefore, of closing one extra
p hole was to concentrate the ascending hot gases, which had pre-
sly been passing through four drop holes, into three, thus inten-
ng the heat at these three points. Near the outer edge of the
nd hearth is where the concentrate should first begin to burn, and
ertain temperature is required for its ignition. As furnace No.
ad a tendency to run cold, closing an extra second hearth drop
, therefore, benefited the furnace by helping to secure proper
tion for the incoming concentrate. Decreasing the second hearth
p hole area probably also helped by lessening the amount of heat lost
adiation from the third hearth.

nother improvement made in maintaining furnace No. 18 at a good
king temperature was increasing the amount of concentrate fed.
feed of concentrate was raised first to 22 and later to 24 tons per

shift. With the higher tonnage the furnace developed more heat and it was found advisable to remove one of the second hearth drop hole covers, again returning to the condition of four open and two closed. On June 2, MacDougall furnace No. 19 was again started, the third hearth drop hole having been enlarged. The furnace was operated in every respect similar to No. 18. It was found that for some unexplained reason No. 19 normally ran hotter than No. 18, and as the tonnage of concentrate fed was increased to 24 tons per shift it was found that No. 19 ran best with al of the second hearth drop holes uncovered. No. 18 throughout the testing work was found to give the best results with two of these drop holes covered. It will be seen, therefore, that the proper second hearth drop hole area is a variable quantity, depending on the temperature conditions of a furnace, quantity of feed treated, and so on.

On June 17 it was decided to run furnace No. 22 against furnaces Nos. 18 and 19 to determine the question of the relative advantages of high and low speed, all other conditions being the same. Accordingly a course of brick was knocked out around the third hearth drop hole of No. 22, and instructions were given to gradually raise the tonnage of concentrate to 24 tons per shift. Nos. 18 and 19 continued to revolve once in 38 sec. and No. 22 was left unchanged at one revolution in 53 sec. The operation of all three furnaces during about a 10-day period showed that the high-speed furnaces gave less trouble from material banking up in front of the arms and in general produced calcine assaying lower in sulphur. A discussion of the relative advantages of high-speed, condensed from the report of Mr. Church, is presented herewith.

(*a*). *Advantages of High Speed.*—1. Effect of Increased Speed on Depth of Bed.—The experimental work showed that for high tonnages the increased speed gave better results, in that the depth of bed on the hearths was kept lower and there was less tendency for the feed to bank up in front of the arms. The reason for the shallower bed in a high-speed furnace is as follows: Assume that in a MacDougall furnace the depth of bed remains constant, then every revolution of the central shaft moves a constant amount of material at any one point on the hearth over a radial distance equal to the width of two furrows. If the speed is increased and the depth of bed remains constant, every revolution moves the same amount of material the same distance as before, but in a given time the greater number of revolutions will move a greater tonnage. If, however, the speed is increased and the tonnage remains constant, every revolution will move a less quantity of material than before, that is, the depth of bed will be reduced.

Effect of Increased Speed on Degree of Calcination.—Increasing speed of a MacDougall furnace, means not only a decrease in the h of bed with constant tonnage, as shown above, but also means the bed is stirred more frequently. Every passage of a rabble by stirring the bed, presents a freshly exposed surface to the n of the oxygen-bearing gases. The best speed is that which ex- s a fresh surface just as rapid oxidation of the sulphur is begin- to die in the old surface. The time required for this rapid oxida- of the surface to cease varies with the character of the material, amount of oxygen in the gases, the temperature of the bed, tem- ture of the gases, and many other factors. In general, however, it be said that the period during which the surface of the bed under rapid oxidation is shorter for the fifth and sixth hearths than for third and fourth hearths. Up to the point at which a surface is ed while still undergoing rapid oxidation of its sulphur, increas- the speed tends to increase the rate of elimination of sulphur.

n the third floor, the speed of one revolution in 38 sec. undoubt- made the stirring too rapid. On the fourth floor observations cated that the increased speed was probably about right, while for ifth and sixth hearths a still higher speed might have been used dvantage.

*Effect on Regular Operation of Department by Applying Results Obtained from Experimental Work.*

y July 1, 1910, the advantages of increased speed had been dem- rated to the extent that it was deemed advisable to change all of furnaces in Battery IV. to the speed of one revolution in 38 sec.; , all the furnaces in the MacDougall department were increased peed, the speeds in use at present (May, 1913) being as shown w:

| *ner Speed of Revolution of Shaft.* | *Present Speed of Revolution of Shaft.* |
|---|---|
| revolution in 53 sec. | Battery I. (4 furnaces), one revolution in 45 sec. |
| | Batteries II., III., and IV. (18 furnaces), one revolution in 38 sec. |

t the same time that all of the furnaces in Battery IV. were ged to the increased speed (July, 1910), the furnaces in operation at battery were also brought up to the increased tonnage of 24 of concentrate per shift. Later, all of the MacDougall furnaces raised to the new tonnages. The effect on the MacDougall de-

partment, as the improved methods of operation developed experimental work were gradually applied to the regular of the furnaces, is shown by the figures tabulated below. ures are taken from the Monthly Smelter Reports.

*MacDougall Department.*

*Average Tonnage Treated per Furnace Day.*

| Month. | Concentrate. | First-Class Ore Screenings. | Miscellaneous. | Total Cupreous Material. | Limestone. | Total. | Total Tons per |
|---|---|---|---|---|---|---|---|
| April, 1910... | 44.8 | 1.2 | ..... | 46.0 | 1.2 | 47.2 | 0 |
| May, 1910... | 46.9 | 3.9 | 0.1 | 50.9 | 2.6 | 53.5 | 0 |
| June, 1910... | 47.2 | 3.6 | 0.6 | 51.4 | 3.6 | 55.0 | 0 |
| July, 1910... | 55.2 | 6.7 | 0.1 | 62.0 | 4.6 | 66.6 | 0 |
| August, 1910.. | 52.4 | 4.0 | ..... | 56.4 | 3.9 | 60.3 | 0 |
| September, 1910... | 67.5 | 3.9 | ..... | 71.4 | 6.5 | 77.9 | 6 |
| October, 1910... | 68.1 | 5.3 | ..... | 73.4 | 8.2 | 81.6 | 0. |
| November, 1910... | 73.8 | 4.6 | ..... | 78.4 | 9.8 | 88.2 | 0. |
| December, 1910... | 73.1 | 4.2 | ..... | 77.3 | 11.0 | 88.3 | 0. |
| January, 1911... | 67.6 | 2.4 | ..... | 70.0 | 11.9 | 81.9 | 0. |
| February, 1911... | 70.5 | 4.0 | ..... | 74.5 | 14.3 | 88.8 | 0. |
| March, 1911... | 75.1 | 0.3 | ..... | 75.4 | 10.3 | 85.7 | 0. |

(*D*). *Fundamental Principles Involved in Increasing the To*

Before the experimental work to increase the capacity of Dougall furnaces was started, the reasons for not treating about 14 tons of concentrate per furnace per shift were that the tonnage raised the percentage of sulphur in the calci the furnaces to become too hot, especially locally on the the 14-in. third hearth drop hole, and made the furnaces heavily, necessitating an excessive amount of cleaning. lem of treating increased tonnages therefore involved the factors :

(*a*). Supplying sufficient oxygen and stirring the hearths enough to burn the increased amount of sulphur.

(*b*). Regulating the heat from the increased amount of sulp burned per square foot of hearth area so that the furnaces become too hot.

(*c*). Regulating the drop hole area so as to avoid too gr centration of heat and too high a velocity of the gases th drop holes, thus preventing the furnaces from building up incrustations, particularly on the roof above the third hea drop hole.

The specific changes made, through which the treating tonnages became possible, were as follows :

(*a*). The furnaces were operated under the full available d

n. of water), with no dampers in the flue necks and with four s open on the sixth hearth, so as to draw enough air through the ace to supply the necessary amount of oxygen.

). The speed of revolution of the central shafts was increased one revolution in 53 sec. to one revolution in 45 sec. on the first ery (four furnaces), and to one revolution in 38 sec. on all the r batteries (18) furnaces. This increase of speed had two effects:

It permitted an increase in tonnage without any particular in-se in the depth of bed on the hearths, thus helping the furnaces hanically.

It stirred the beds more frequently and aided in the more rapid ation of sulphur by bringing the sulphur in contact with sufficient gen. In a given time, to obtain the same degree of calcination, a ace treating a higher tonnage has to eliminate a larger quantity lphur.

). First-class ore screenings and lime rock were mixed and fed the increased tonnage of concentrate to reduce and regulate the perature of the furnaces.

). The area of the third hearth center drop hole was increased. reduced the amount of heat which had formerly been concen-d at this point and decreased the velocity of gases through this ing, resulting in permitting the use of a higher tonnage without xcessive amount of crusting on the third hearth roof.

). The second hearth drop hole area was regulated so as to obtain cient heat at the edge of the second hearth to ignite the incoming entrate.

## I. Treatment of an Increased Amount of Ore Screenings.

he experimental work described above had resulted in increasing efficiency of the MacDougall department by increasing the capac-f the individual furnaces. That is, the total output of the de-ment remained about the same, but the work was done by a fewer ber of furnaces, resulting, as will be shown later, in a considera-reduction in labor cost. A short time later it also became desir-to increase the total output of the MacDougall plant to supply feed for an additional reverberatory furnace. The MacDougall aces were already handling practically the entire output of fine entrate from the concentrator, and no particular increase in the ply of fine concentrate could be looked for. To increase the unt of material produced by the MacDougall department, there-, it was desired to treat a larger amount of first-class ore screen-, of which a sufficient supply could be obtained.

In the previous work, the ore screenings, being lower in
than the concentrate, had been used as a cooling agent to k
the temperature of the furnaces. The problem now becam
of running the MacDougalls so as to treat the best possible t
concentrate, using screenings only as a cooling agent, one o
the MacDougalls so as to treat the best possible tonnage
ings by utilizing the available heat in the most efficient w
means used for treating a large tonnage of screenings were a

1. Operation of the furnaces on a high tonnage of concen
2. Proper regulation of the second hearth drop hole area.
3. Withdrawal of lime rock from the MacDougall feed
being taken by screenings.

1. Operating Furnaces on a High Tonnage of Conc
Screenings being low in fuel value and having a cooling effe
viously stated, it was necessary to keep the furnace as hot a
Increasing the feed of concentrate meant burning more
sulphur per square foot of hearth area per minute and develo
heat. The concentrate feed to each individual furnace was
maintained as high as possible without unduly raising the p
of sulphur in the calcine.

2. Proper Regulation of Second Hearth Drop Hole
order to secure proper ignition for the feed, the second he
hole area was regulated so as to obtain as high a temperat
outer edge of the second hearth as possible without too mu
tling of the draft.

3. Replacement of Lime Rock by Screenings.—Lime ro
no fuel value, it was found that its place could be taken by
quantity of screenings, the screenings carrying an average
to 20 per cent. of sulphur and therefore having a fuel va
construction of the MacDougall furnaces was therefore c
that on two furnaces in each battery lime rock could be fe
to the outer edge of the sixth hearths. All the lime rock t
the MacDougall plant was fed to the sixth hearths of these

In addition to the possibility of treating a larger tonnage
ings, there were a number of other advantages in favor of thi
ment. One advantage was that the amount of lime rock
with the calcine was not limited by the temperature conditi
furnaces, for feeding lime rock directly to the edge of the si
had very little cooling effect. Another advantage was that
rock being greater in size than any other constituent of the M
feed, its removal from the main body of the furnace lessene
ger of breaking rakes by the wedging of large fragments

blades. Furthermore, there had been no particular advantage in sending the lime rock completely through the furnaces. It contained no sulphur to be eliminated, and, as the temperature in the furnaces was not great enough to burn lime rock to lime, it underwent no chemical changes. By sending it directly to the outer edge of the sixth hearths it was warmed as much as if it had passed entirely through the furnaces and was nearly as well mixed with the calcine.

## VII. Maximum Percentage of Ore Screenings in a MacDougall Feed.

At this point, the experimental work on increasing the efficiency of the MacDougall department was taken up by Mr. Corwin, who carried out the tests to determine the maximum percentage of screenings which could be treated in a MacDougall feed, and to further increase the tonnage of concentrate treated per furnace, Tests on the Repath-Marcy furnace, described later, were made by both of the writers, and the experimental work on the Crouse furnace was taken up by Mr. Rodgers. All of the testing was done under the direction of Mr. Krejci and Mr. Crowfoot, and in co-operation with George S. Crouse, MacDougall foreman, to whom a great amount of credit for the steady improvement in MacDougall work is due.

The test to determine the maximum percentage of screenings, even up to 100 per cent., which could be treated in a MacDougall feed was started in December, 1910. The results of this test are of unusual interest as showing the wide range in sulphur content of the material which can be successfully roasted in a MacDougall furnace.

At the time of starting the test about 8,820 lb. of screenings could be treated per furnace per shift, mixed with about 50,000 lb. of concentrate. The ratio was therefore about 25 tons of concentrate (wet weight) to 4.41 tons of screenings per shift, or 85 per cent. of concentrate to 15 per cent. of screenings. The screenings were fed intermittently during a shift, and, as they always had a cooling effect, they were used only when the furnaces were running particularly hot. This led the furnacemen to believe that the continuous treatment of a 100 per cent. screenings feed would be impossible, and in carrying out the test it was necessary to increase the percentage of screenings by slow degrees. The successful solution of the problem depended on keeping the furnace sufficiently hot.

MacDougall furnace No. 16 was used for the test work, and whenever necessary the temperature of the furnace was raised by means of either or both of the two following:

1. Compressed air.
2. Coal and oil.

1. Compressed Air.—The advantage of compressed ai ing a MacDougall furnace hot was discovered by noticing produced by air blown into the furnace through an oil ma the oil in the machine had been exhausted. It was observ jet of air on the fourth hearth caused more rapid oxida sulphur, developing more heat and increasing the tempera furnace. The increase in rapidity of the oxidation was the air impinging on the hearth, bringing the calcine int contact with fresh oxygen.

In the test to determine the maximum percentage of ore which could be used in the feed to a MacDougall furna No. 16 was first equipped for compressed air with three 0. inserted through the observation holes of the fourth he It was found that air blown through the sepipes aided gr creasing the temperature of the furnace, and many time furnace showed signs of becoming cold a recovery was means of compressed air without the use of oil or coal. vantage, however, was that there was an increased amount ing on the rabble arms, due to sparks produced by the a on the bed. A large amount of this incrusting was done and better results obtained from the air, by replacing t pipes by 0.75-in pipes. This change made it possible to crease the velocity of the air entering the furnace, at the increasing the volume. Thus the amount of sparking w and the hearth area affected by the air increased.

A further improvement was made in the method of i the compressed air into the furnace before the completion On the fourth hearth, three 0.75-in. pipes each 4.5 ft. lo serted through holes drilled in the shell and brickwork nace. These pipes were run towards the center of the fur the fourth hearth roof, leaving just enough room for the r to pass beneath them. Just outside of the shell of the f pipes were each connected to a rubber hose delivering air from the converter air main. Inside of the furnace the had two series of 0.25-in. holes drilled in them, extendin brick wall of the furnace to the ends of the pipes. The drilled 1.5 in. apart, and the two series were each 0.5 in. a ing the air at nearly right angles down on the bed. T caused the compressed air to affect a large area of th because the air was directed against the bed at nearly ri

'king was practically eliminated. Pipes were also used to intro-
e air to the fifth hearth, the arrangement being exactly the same
or the fourth. It should be remarked here that, although the
ipment described above was found highly satisfactory during the
't period of the test, it would not be suitable as a permanent in-
lation, for the reason that the pipes are liable to become bent by
heat and interfere with the arms, and also because they make it
cult to clean the roofs of the hearths where they are placed.

he experiment was also tried of blowing air in on the third hearth,
the results were unsatisfactory. The temperature of the third
rth is too variable, and unless the air is blown against a hot bed it
s not increase the rate of oxidation of the sulphur, having simply
oling effect.

. Coal and Oil.—As the percentage of ore screenings in the feed
MacDougall furnace No. 16 was increased, the furnace could not
kept hot enough by the use of air only, and it became necessary
use a small amount of slack coal. This coal was fed through a
erent hopper from the screenings and concentrate, a fairly even
l being obtained by poking the coal through the hopper a little at
me and allowing it to mix with the screenings and concentrate on
first hearth. Whenever the furnace received a slight set back
was also used to keep it hot, the oil being blown in on the fourth
rth in the usual manner.

n night shift of Feb. 7, 1911, a 100 per cent. ore screenings feed
first treated in the furnace and from then until Feb. 12, the fur-
e was maintained on this feed. Coal and air were used continu-
y to keep the furnace hot, and oil when necessary. To show the
fuel value of ore screenings as compared with concentrate,
assays on monthly samples of screenings and fine concentrate
ted in the MacDougall department during January, 1911, are pre-
ted below:

*Assay Results on Monthly Samples, January, 1911.*

| To MacDougalls. | Electrolytic Cu. | $SiO_2$. | FeO. | $Al_2O_3$. | CaO. | S. |
|---|---|---|---|---|---|---|
| | Per Cent. | Per Cent. | Per Cent. | Per Cent. | Per Cent. | Per Cent. |
| -class ore screenings, . . . | 7.10 | 46.9 | 16.6 | 9.7 | 0.1 | 17.4 |
| concentrate, . . . . . | 7.45 | 24.0 | 32.8 | 6.1 | 0.1 | 33.4 |

record of MacDougall furnace No. 16 for the last 19 days of the
is given herewith.

*Record of MacDougall Furnace No. 16 During Test on Tr of Ore Screenings.*

| Date, 1911. | Tons of Material Charged. | | | Per Cent. of Feed. | | Fuel Used. | | Fuel per of Feed |
|---|---|---|---|---|---|---|---|---|
| | Concentrates. | Screenings. | Total. | Concentrates. | Screenings. | Coal. | Oil. | Coal. |
| | | | | | | Pounds. | Gallons. | Pounds. |
| January 25..... | 65.16 | 17.75 | 82.91 | 78.59 | 21.41 | ..... | ..... | ..... |
| January 26..... | 48.71 | 26.97 | 75.68 | 64.36 | 35.64 | 3,425 | 63 | 45.26 |
| January 27..... | 45.68 | 31.40 | 77.08 | 59.26 | 40.74 | 3,500 | 74 | 45.41 |
| January 28..... | 34.64 | 34.42 | 69.06 | 50.16 | 49.84 | 4,100 | 43 | 59.37 |
| January 29..... | 30.51 | 36.11 | 66.62 | 45.80 | 54.20 | 3,050 | 35 | 45.78 |
| January 30..... | 24.37 | 40.15 | 64.52 | 37.77 | 62.23 | 2,025 | 35 | 31.39 |
| January 31..... | 24.39 | 46.64 | 71.03 | 34.34 | 65.66 | 3,400 | 41 | 47.87 |
| February 1..... | 18.30 | 48.21 | 66.51 | 27.51 | 72.49 | 4,350 | 56 | 65.40 |
| February 2..... | 14.21 | 66.95 | 81.16 | 17.51 | 82.49 | 10,605 | 17 | 130.67 |
| February 3.. | No scre | enings | used fr | om Feb. | 3 to 6, | inclusive. | | |
| February 4.. | | | | | | | | |
| February 5.. | | | | | | | | |
| February 6.. | | | | | | | | |
| February 7..... | 7.80 | 56.32 | 64.12 | 12.16 | 87.84 | 9,255 | 24 | 144.34 |
| February 8..... | ..... | 74.13 | 74.13 | ..... | 100 | 13,745 | 8 | 185.42 |
| February 9..... | ..... | 74.35 | 74.35 | ..... | 100 | 11,430 | 46 | 153.73 |
| February 10.... | ..... | 69.80 | 69.80 | ..... | 100 | 10,260 | 47 | 146.99 |
| February 11.... | ..... | 65.80 | 65.80 | ..... | 100 | 8,335 | 24 | 126.67 |
| February 12.... | ..... | 66.52 | 66.52 | ..... | 100 | 7,955 | 5 | 119.59 |

## VIII. Tests to Further Increase Capacity of Fur

Immediately on the completion of the test to determine mum percentage of ore screenings which could be treated Dougall feed, experimental work was again resumed on the tonnages. MacDougall furnace No. 16, which had just on the screenings test, was used also for the tonnage test. in view was to treat in this furnace the highest tonnage with good calcination. As already stated, this furnace wa with pipes for blowing air on the beds of the fourth and fi In the screenings test the air had been used to increase oxidation, in order to increase the temperature of the furna high tonnage test compressed air was used to increase the dation, but not to make the furnace run hotter. With th nage of concentrate, and therefore the large amount of sulp per square foot of hearth area, the furnace ran so hot always necessary to feed screenings along with the concent the furnace and prevent the material on the third hearth f The object of increasing the rate of oxidation was not, th raise the temperature of the furnace, but to improve the calcination. The speed of revolution of the central sha changed, remaining at one revolution in 38 sec.

The high-tonnage test was carried out during the period 13 to Mar. 3, inclusive, 1911. The maximum tonnage tre

24-hr. period, 105.08 tons, was put through the furnace on Feb. Below is given a record of the furnace for the three shifts of . 23:

*Record of MacDougall Furnace No. 16, Feb. 23, 1913.*

| Shift. | Tons Material Charged. | | Total Feed. Tons. | Per Cent. of Feed. | | Fuel. | | Per Cent. in Calcine. |
|---|---|---|---|---|---|---|---|---|
| | Concentrate. | Screenings. | | Concentrate. | Screenings. | Coal. Pounds. | Oil. Gallons. | |
| . . . . . | 32.48 | 1.60 | 34.08 | 95.31 | 4.69 | . . . | . . . | 12.2 |
| noon, . . . . . | 32.14 | 3.16 | 35.30 | 91.05 | 8.95 | . . . | . . . | 13.1 |
| t, . . . . . | 32.47 | 3.23 | 35.70 | 90.95 | 9.05 | . . . | . . . | 11.6 |
| Total 24 hr., . . . . . | 97.09 | 7.99 | 105.08 | . . . | . . . | . . . | . . . | . . . |

lthough it was found that on some days tonnages of around 100 per 24 hr. could be treated, producing calcine assaying about per cent. in sulphur, the test showed that these tonnages could be treated continuously on account of trouble with the driving hinery. Sometimes the feed would become so heavy on the first rth that the machinery would refuse to carry it, and it would be essary to rabble the material by hand on to the second hearth. akages of the driving machinery were of frequent occurrence, ne-itating shutdowns for repairs. During these shutdowns the fur-e would lose not only the tonnage which would have been treated it been possible to continue the running of the furnace, but also, account of the cooling off of the furnace during the shutdown, it ld lose tonnage on account of the necessity of reducing the feed tarting up, until again reaching a normal working temperature. he heavy load on the driving machinery was the result of increas-the tonnage treated without increasing the speed of revolution of central shaft. As explained earlier in this report, increased ton-e with constant speed means deeper beds on all the hearths. The showed that with the central shaft making one revolution in 38 sec. tons per 24 hr. could not be treated continuously in the furnace, on unt of overloading the driving machinery. Increasing the speed of central shaft was not tried and might or might not have caused too uent stirring of the bed. Whether or not this tonnage could have n treated successfully by means of increased speed is still an open stion.

## IX. Tests to Decrease Percentage of Flue Dust Ma MacDougall Furnaces.

The month following the completion of the high-tonnage tes 1911, experimental work was undertaken to try to reduce centage of the flue dust made by the MacDougall furnaces. dust produced by a MacDougall furnace is caused by the dropping from hearth to hearth through the rising current With a strong draft small particles of partly roasted concen picked up by the gases and carried along into the flue. S on the lower floors the incrustations around drop holes are of the material falling through the rising gases. Small pa calcine are picked up by the draft, and as during the fall thr gases they undergo very rapid oxidation and become part when they are thrown up against the roof they stick there, accretions. The experimental work to reduce the amount of was therefore based on eliminating the fall of material thr upward current of gases.

### (*A*). *Repath-Marcy Furnace.*

The first tests were made with the method invented and by Repath and Marcy. This method provides drop holes fo sage of the material from hearth to hearth, which are seal material itself against the gases, and also provides separate ways from hearth to hearth for the gases. A furnace fulfill requirements was designed by the engineers at the Great Fall and MacDougall Furnace No. 17 was remodeled to conform design. This furnace will be referred to as the Repath-Marc The construction of the furnace is shown in the accompanyin Fig. 4.

The Repath-Marcy furnace was started on Mar. 15, 1911. there were a number of mechanical difficulties encounter interfered with the regular discharge of the material from hearth. At times the drop holes would allow the materi through too fast and would become open for the passage At other times they would not allow the material to pass thr enough and would become choked. Another difficulty met the excessive loss of heat by radiation from the gas pass The furnace was therefore shut down after a few days' run defects in the drop hole construction were remedied, and th sageways lined with brick. On starting up again after mak changes, with the furnace running satisfactorily mechanical nearly as possible in accordance with the ideas of the designe

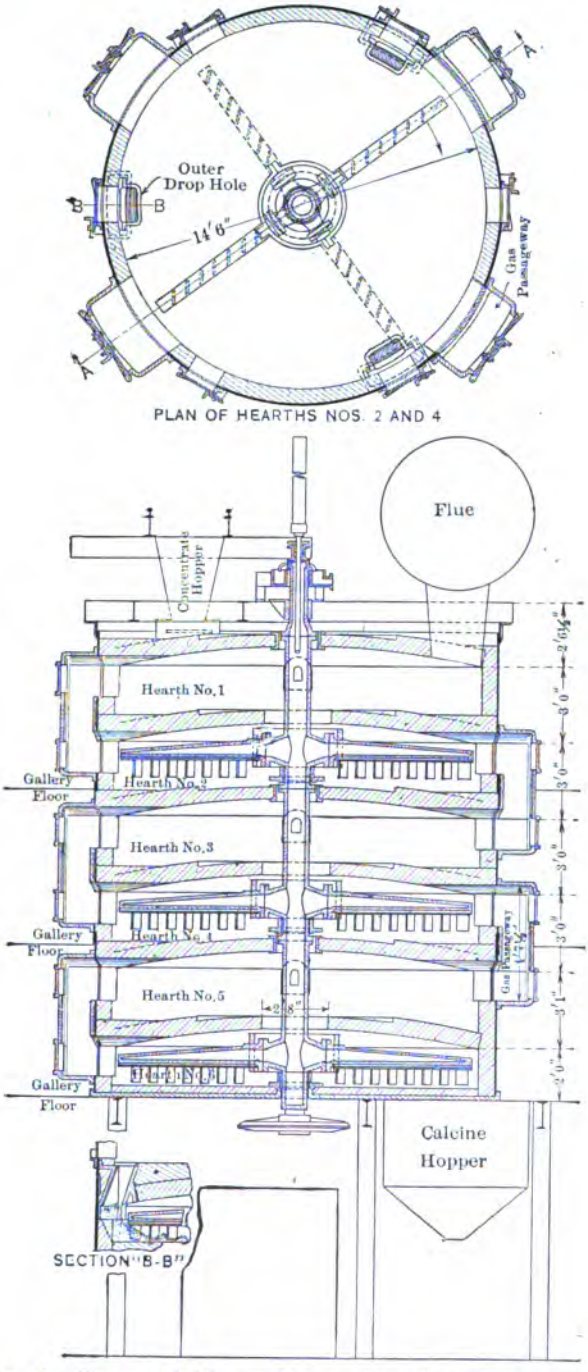



**Fig. 4.**—Repath & Marcy Method for Dust Prevention in MacDougall Furnace.

found impossible to operate the furnace without the use of oil. Different conditions of tonnage and draft were tried, result was always the same: The furnace would not run with Even when the fourth hearth was kept hot with coal and oil cine produced would assay from 18 to 20 per cent. in sulph test showed, therefore, that the furnace as first constructed w success.

From the above it will be seen that the furnace was unsat on account of its reduced power to eliminate sulphur. Not sulphur was burned to either keep the furnace hot or produce calcine. The reason for the low elimination of sulphur was of the principal means of oxidizing the sulphur in the Evans-l type of MacDougall furnace—namely, the showering down of through the rising current of hot oxygen-bearing gases—h removed without anything else being substituted to take i In the regular type of MacDougall furnace a large amount o is burned and a considerable proportion of the total heat is g during the fall of material from hearth to hearth. Durin each particle is exposed on all sides to the action of oxygen ation goes on much more rapidly than when the particle is the surface of the hearth. Eliminating this fall therefore red amount of flue dust made by the furnace at the expense of power.

To make sure, however, that the reduced amount of oxida not caused by an insfficient amount of air being drawn thr furnace, a test was carried out under the direction of Mr. Cr determine the relative amount of gas being discharged by the Marcy furnace in comparison with two of the regular Ma furnaces, Nos. 18 and 1. The test showed that there was abo cent. as much gas by weight being discharged from the Repa furnace as from furnace No. 18, and about 81 per cent. as from furnace No. 1. The lesser amount of gas discharge Repath-Marcy furnace was probably caused by a greater resistance to the passage of air through the furnace in compar the regular MacDougall type. The gas passageways, after be with brick, had a smaller area than the drop holes in the MacDougall furnace. Also, the path of the gases through th Marcy furnace was longer. Inasmuch, however, as the Repa furnace was treating less than two-thirds as much material as furnaces, it was evident that sufficient oxygen was being draw the furnace for the tonnage treated, and that the decrease to oxidize sulphur was caused by eliminating the fall of mate hearth to hearth through the rising current of hot gases.

### (*B*). *Modified Repath-Marcy Furnace.*

bservations on the MacDougall furnaces had tended to show that of the dust carried into the flue was produced in the fall through center drop hole from the first hearth to the second, and that a of the sparks formed in the fall through the side drop holes from the nd hearth to the third were also carried into the flue. The test work ng indicated that, as then constructed, a certain amount of shower-down of material through the rising gases was necessary in order to in a good elimination of sulphur, it was decided to modify the de- of the Repath-Marcy furnace. The furnace was accordingly shut n and the following changes made: Between the fourth and fifth, the fifth and sixth hearths the sealed drop holes and separate gas ageways were eliminated, and the construction was changed back at of the regular MacDougall furnace, allowing the material trav- from the fourth to the fifth and the fifth to the sixth hearths to through the rising current of gases as before. Also, pipes were lled to blow compressed air on the fourth and fifth hearths, the pment being exactly the same as was used in the test to treat a per cent. ore screenings feed.

fter the above changes had been made there was no trouble expe- ed in running the furnace on a feed not exceeding 20 tons per , although to maintain the furnace at a working temperature it necessary to use compressed air continuously. Whenever the air shut off for a while a cooling action would soon be observed on hird hearth, and in from 2.5 to 3 hr. the calcine would run high ulphur. Also, it was found impossible to increase the feed over ns per shift without cooling the furnace and producing high sul- s. The furnace was operated in this way for about a month n May 16 to June 15, 1911), treating on an average about 18 tons oncentrate, wet weight, per shift and producing calcine averaging t 7.8 per cent. of sulphur.

uring the regular operation of this furnace it was desired to com- it with the Evans-Klepetko type of MacDougall furnace, to deter- e whether or not any improvement had been made in reducing the unt of flue dust produced. As all of the furnaces in each battery harge their gases into a common flue, any method involving actu- collecting and weighing the flue dust would have been out of the tion. A method was therefore devised as follows: The two fur- s to be compared were operated on a feed of concentrate only. ing the test period all the concentrate charged to each furnace was ghed and sampled, and also all the calcine, calcine barrings, and ine clean-up produced by each furnace. At the start and at the

end of the test the concentrate hoppers of both furnaces were
and the calcine hoppers empty. From the weights and ass
the pounds of copper charged to each furnace and the pounds
recovered in the furnace products by each furnace were c
For each furnace, the difference between the weight of coppe
and the weight of copper recovered in the furnace pro
assumed to be the weight of copper which had gone into
Applying an average assay for copper in MacDougall flue d
above weights of copper assumed as going into the flue, the
flue dust produced by each furnace was calculated as show

$$\frac{\text{Pounds copper fed} - \text{pounds copper recovered in furnace products}}{\text{Average assay per cent. copper in MacDougall flue dust.}} \times 100 = \text{Po du}$$

As a check on the above, the weight of flue dust pro
also calculated as follows:

$$\frac{\text{Pounds } SiO_2 \text{ fed} - \text{pounds } SiO_2 \text{ recovered in furnace products}}{\text{Assay per cent. } SiO_2 \text{ in MacDougall flue dust.}} \times 100 = \text{Pounds produce}$$

Using the above-described method, a comparative test w
out on the modified Repath-Marcy furnace No. 17 and o
No. 18 during the 48-hr. period from 9 a.m., May 25, to 9
27, 1911. A summary of the principal data obtained from
is presented herewith:

*Summary of Test Data.*

| Average. | Furnace No. 18. Evans-Klepetko type. |
|---|---|
| Tons of concentrate treated per 24 hr. (wet weight), | 84.96 |
| Per cent. of moisture in concentrate treated, | 9.5 |
| Tons of concentrate treated per 24 hr. (dry weight), | 76.85 |
| Tons of calcine produced per 24 hr., | 46.83 |
| Assay per cent. sulphur in calcine, | 10.7 |
| Per cent. of dry weight of feed recovered in calcine, | 60.93 |
| Per cent. of copper fed recovered in calcine, | 74.3 |
| Per cent. of sulphur fed eliminated (from calcine), | 81.5 |
| Calculated by Cu and $SiO_2$ methods. | |
| Pounds of flue dust produced per 24 hr., | 27,000 |
| Pounds of flue dust per dry ton fed, | 351 |
| Flue dust, per cent. of dry weight of charge, | 17.6 |

Although the test figures showed that the modified Rep
furnace as constructed at the Great Falls Smelter made co
less flue dust, from the fact that the furnace as arranged by us
much reduced capacity and could not be operated withou
tinuous use of compressed air, it was not thought advisabl
this type of furnace for regular work.

should be remarked in passing that the figures for flue dust above include all of the flue dust actually passing out of the ces. Part of this dust settles in the MacDougall cross flues, and there passes through hoppers and chutes to the calcine hoppers v. Part settles in the main MacDougall flue and is drawn off cars by the calcine trammers. Practically all of the remainder covered in the uptake and crosstake flues and the main dust ber. Although the MacDougall furnaces produce a large nt of flue dust, therefore, a considerable proportion of the dust covered at about the same cost as is involved in handling ne.

(*C*). *Crouse Furnace Designed for Dust Prevention.*

ortly after the completion of the test on the modified Repath-y furnace, a new test was started on a furnace equipped ust prevention designed by George S. Crouse. The furnace de-d by Mr. Crouse was similar to the modified Repath-Marcy fur- in that it was arranged to carry the material from the first h to the second and from the second hearth to the third without ping it through the rising gases, but the method of doing this entirely different. In the Repath-Marcy furnace drop holes provided for the descending material and separate gas passage- for the rising gases. In the Crouse furnace the gases traveled rds through the drop holes exactly the same as in the Evans-etko type of MacDougall furnace. Concentrate passing from irst to the second hearth was discharged through an extra drop which, by an ingenious arrangement, was sealed against the g gases only during the brief period when material was falling gh it. From the second to the third hearth the concentrate fell gh the regular side drop holes, these drop holes being open for rising gases except when concentrate was being discharged gh them.

brief description of the equipment of the Crouse furnace is as ws: The extra first hearth drop hole measured 7 by 8 in. in plan was located about 4 in. from the edge of the regular center drop All the rakes on the first hearth rabble arms pushed the con-rate towards the center of the furnace except the two inner rakes, on each arm. These rakes pushed the material away from the er. The concentrate, therefore, instead of showering continu- over the edge of the first hearth center drop hole, as in the lar type of MacDougall furnace, was forced to pass through the 7 by 8 in. drop hole, falling twice in every complete revolution

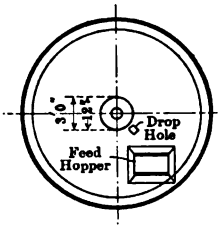
of the central shaft. The rabble arms on the first, second,
hearths were built so as to be always in line one above the
the second hearth rabble arms, towards the center of th
were attached two iron vessels open at the top and bottom,
from the bed of the second hearth up to nearly flush with
When one of the first hearth rabble arms would be droppin
trate through the extra 7 by 8 in. drop hole, the receiving
one of the second hearth rabble arms would be directly
drop hole, sealing it against the rising gases. Similarly, a
vessel was attached near the outer end of each of the thi
rabble arms, sealing the regular second hearth side drop h
material was falling through them.

In dropping the material from the third to the fourth h
extra drop holes were used. This drop hole construction fo
hearth was referred to earlier in this report in connectio
question of enlarging the third hearth center drop hole f
to 20 in. The extra third hearth drop holes were not pecu
Crouse furnace, being in use in several of the other furna
time. They were not at any time sealed against the rising
by dropping a large bulk of material four times in every
of the central shaft, instead of having it showering continu
the edges of the center drop hole, the amount of incrustati
on the third hearth roof was greatly reduced. Also, the in
formed were easier to remove, because they were localize
two extra drop holes. These extra drop holes permitted t
14-in. third hearth center drop hole.

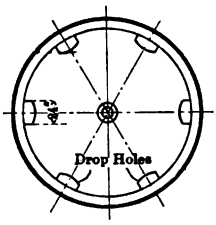
One further feature in the equipment of the Crouse fu
the use of "spark catchers" on the ends of the fourth hea
arms. These "spark catchers" were plates extending h
in front of and in back of the ends of the arms. The ob
plates was to catch the sparks formed while the rabble
pushing material through the fourth hearth side drop h
preventing the building up of incrustations on the roofs.
could be cleaned through one door, and could be cleaned n
than the roof above the drop holes.

The drop hole arrangement and general equipment of
furnace is shown in the two accompanying sketches, *A*
Fig. 5.

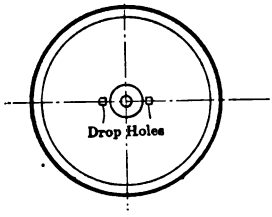
The Crouse furnace, then, eliminated the dropping of c
through the rising gases between the first and second and
and third hearths without providing any separate gas pa
One result of this was that the Crouse furnace was much
pact than the modified Repath-Marcy furnace. At this p



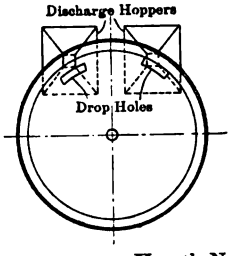

Hearth No. 1.
center drop hole is now used as a gas
eway only, and extra drop hole for
trates.



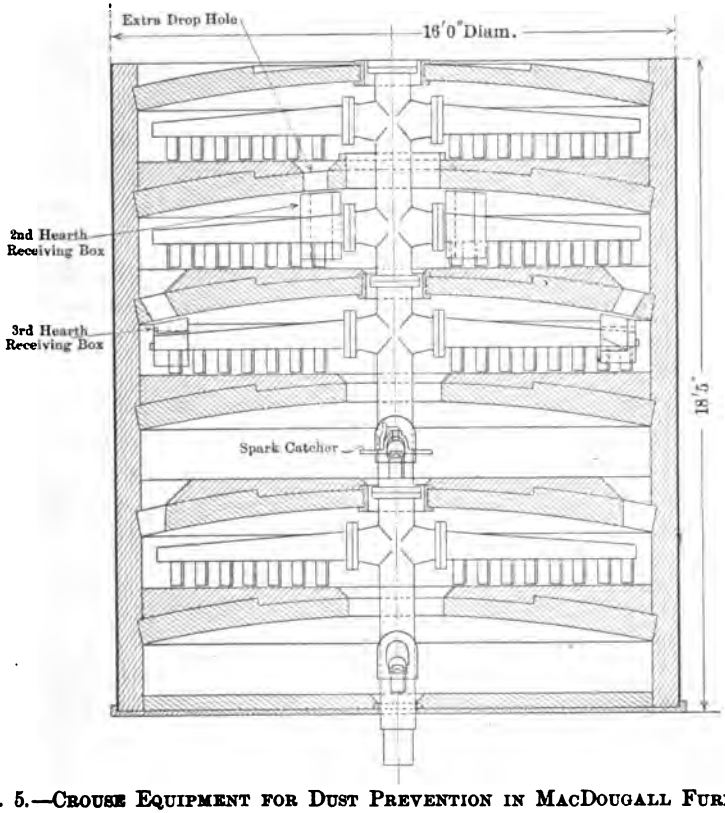

Hearths Nos. 2 and 4.
Usually one or more of these side drop holes are covered to damp the air current.




Hearth No. 3.
two extra drop holes through which
cines fall to the hearth below, localize
duce incrustations.




Hearth No. 6.
The discharge drop holes are 8 in. wide and 28 in. long on outer arc.




. 5.—Crouse Equipment for Dust Prevention in MacDougall Furnace.

be remarked that the dropping of material through the rising gas current may be eliminated between the first and second and the second and third hearths with much less injury to the capacity of the furnace than would be produced by eliminating this drop between the lower hearths. For example, in falling through the rising gases from the first to the second hearths, the concentrate may be dried and heated to a certain extent, but no sulphur is eliminated and no heat generated. In dropping through the gas current between the lower floors, a certain amount of silver is eliminated from the calcine and heat is evolved. Dispensing with this drop between the lower floors would therefore not only raise the percentage of sulphur in the calcine, but would also take away part of the means on which the furnace depends for the supply of heat to keep it running. In a furnace where the heat is all produced from the oxidation of the sulphur without the help of any outside fuel, any factor tending to cut down the total amount of heat produced is a serious detriment to the furnace.

(*a*). *Tests on the Crouse Furnace.*—MacDougall furnace No. 3 was remodeled to conform with Mr. Crouse's design, and was first started on Dec. 18, 1911. From the very beginning this furnace demonstrated that it would run satisfactorily without the use of coal, oil, or compressed air and would produce calcine assaying the same percentage in sulphur as the average of the other furnaces, although at first it was necessary to feed it a slightly lower tonnage than the average of the other furnaces. A number of minor changes were made in the equipment from time to time, such as rebuilding the second hearth drop holes to make them come down more nearly to the tops of the receiving vessels on the third hearth rabble arms, and adding two extensions to the third hearth receiving vessels. In general, these changes were designed to seal the drop holes more securely against the ascending gases during the period of the falling of the material from hearth to hearth.

The first test to compare the Crouse furnace with one of the regular MacDougall furnaces as to the relative amounts of flue dust produced, was carried out in April, 1912. The results of this test showed that the Crouse furnace was producing considerably less flue dust, but at the expense of a certain reduction in calcining capacity. With both furnaces being fed practically the same tonnage of concentrate, 64.5 tons, wet weight, per 24 hr., the Crouse furnace produced calcine assaying 14.2 per cent. of sulphur, as against 10.5 per cent. of sulphur in the calcine from the regular MacDougall furnace. Temperature readings taken to compare the Crouse furnace with the regular MacDougall furnaces gave the following results:

*Temperature in °F. on Hearths of MacDougalls.*

| | Hearth No. 1. | Hearth No. 2. | Hearth No. 3. | Hearth No. 4. | Hearth No. 5. | Hearth No. 6. |
|---|---|---|---|---|---|---|
| Average of six regular MacDougall furnaces, . | 429 | 1,017 | 1,142 | 1,216 | 1,162 | 1,106 |
| Crouse furnace, . . . . . | 560 | 880 | 960 | 1,100 | 1,135 | 1,150 |

From the above figures it will be seen that the Crouse furnace ran considerably cooler than the regular furnaces, as would be inferred from the fact that in the comparative test, treating the same feed as the regular furnace with which it was compared, the Crouse furnace produced calcines higher in sulphur; that is, it burned a lesser amount of sulphur per minute and per square foot of hearth area.[1]

As a result of this test the Crouse furnace was shut down on Apr. 13 and the following changes were made in the equipment:

*Changes Made in Equipment of MacDougall Furnace No. 3, Apr. 19 to Apr. 24, 1912.*

*Hearth.* *Changes.*

First.—Center drop hole increased in size. Formerly measured 11 in. from shaft to edge of drop hole. Changed to 14 in. from shaft to edge of drop hole.

Second.—Side drop holes previously measured approximately 5 by 9 by 20 in. Enlarged by chipping out brick between edge of drop holes and shell of furnace to approximately 12 by 15 by 21.5 in.

Third.—Previously center drop hole measured approximately 20 in. from center shaft to edge of drop hole. Was originally made 11 in. with two extra drop holes, but had become enlarged to 20 in. by the falling out of two center courses of brick, converting the center drop hole and extra calcine drop holes into one large drop hole. Rebuilt, making edge of center drop hole 14 in. from shaft, and two extra calcine drop holes 6 by 7 in.

Fourth.—Side drop holes enlarged from approximately 5 by 9 by 20 in. to approximately 10 by 12.5 by 25 in. Brick chipped out between edge of drop hole and shell of furnace.

Fifth.—No changes. Center drop hole measured 14 in. from shaft to edge of drop hole.

Sixth.—No changes.

[1] For further data regarding temperatures on the hearths of MacDougall furnaces, see *Practice of Copper Smelting*, by E. D. Peters, pp. 93 to 95, and also paper by L. S. Austin, The Washoe Plant of the Anaconda Copper Mining Co. in 1905, *Trans.*, xxxvii., 431 (1906).

The above changes in general increased the drop hole area, making it possible for a larger volume of air to be drawn through the furnace. Also, the increase in drop hole area was made as far as possible by chipping out the brick between the outer edges of the side drop holes and the shell of the furnace, thus increasing the drop hole area with as little reduction in hearth area as possible. In the regular MacDougall furnaces the standard size of drop hole for the second and fourth hearths is 5 by 9 by 24 in. In comparison with the regular MacDougall furnaces, therefore, the Crouse furnace, after the above changes had been made, had a larger second and fourth hearth drop hole area.

When the Crouse furnace was again started it was found that the changes in construction had brought about a considerable increase in calcining capacity, and that it would handle satisfactorily as large a tonnage as the average of the other furnaces. A comparative test was made between the Crouse furnace and McDougall furnace No. 1, lasting for eight days during the period from May 21 to May 29, 1912. Data obtained from this test are tabulated below:

*Summary of Test Data.*

*Test period:* 12 *m.*, *May* 24, *to* 9 *a.m.*, *May* 29, 1912.

| Average. | Furnace No. 1. | Crouse Furnace. (No. 3.) |
|---|---|---|
| Tons concentrate treated per 24 hr. (wet weight), | 67.157 | 68.493 |
| Per cent. of moisture in concentrate treated, | 8.0 | 7.9 |
| Tons of concentrate treated per 24 hr. (dry weight), | 61.78 | 63.079 |
| Tons of calcine produced per 24 hr., | 39.372 | 42.706 |
| Assay per cent. sulphur in calcine, | 10.3 | 9.6 |
| Tons of total products, excluding flue dust, per 24 hr., | 40.531 | 44.334 |
| Per cent. of dry weight of feed recovered in above products, | 65.6 | 70.3 |
| Per cent. of copper fed recovered in above products, | 78.6 | 87.0 |
| Per cent. of sulphur fed eliminated from above products, | 81.3 | 81.2 |
| Calculated by Cu Method. | | |
| Pounds of flue dust per 24 hr., | 21,049 | 13,422 |
| Pounds of flue dust per dry ton fed, | 341 | 213 |
| Flue dust, per cent. of dry weight of charge, | 17.0 | 10.7 |

From the above figures it will be seen that the Crouse furnace as finally constructed was a complete success. It had a slightly greater calcining capacity than the regular furnace with which it was compared, and was a big improvement over the regular furnace in that it converted a larger percentage of the feed into furnace products and made less flue dust. Enlarging the drop hole area on the regular furnace to correspond with the drop hole area of the Crouse furnace might or

might not have again given the regular furnace a slightly greater calcining capacity, but the Crouse furnace would still have had the big advantage in its favor of making less flue dust. It may also be remarked here that although on account of the rebuilding of the smelter, and possible changes in the MacDougall plant, the old MacDougall furnaces have not been remodeled to conform with the Crouse design, the original Crouse furnace, No. 3, has continued in successful operation up to the present date.

MacDougall furnace No. 18, with which the modified Repath-Marcy furnace was compared, in the test figures shows a greater calcining capacity than MacDougall furnace No. 1, with which the Crouse furnace was compared. These two MacDougall furnaces, Nos. 18 and 1, are exactly similar in construction, and with the same character of feed have practically equal calcining capacities. A change in the grade of concentrate between the dates of the two tests accounts for the larger tonnage treated by MacDougall furnace No. 18.

## X. Concrete Hearths.

A comparatively recent change made in the construction of the MacDougall furnaces, which has tended to decrease the operating cost of the department, has been the substitution of reinforced concrete hearths in place of the old brick hearths. The brick hearths were 9 in. thick, 14 ft. 6 in. in diameter, and were made of regular size bricks laid in regular courses. Due to the expansion and contraction caused by changes in temperature when the furnaces were started and shut down, these bricks would often become loosened and would fall out around the drop holes. Also, bricks would become loosened by barring heavy incrustations from the roofs, and holes would occasionally be formed in the hearths. The falling out of bricks from either of the two above-mentioned causes would decrease the area of the hearths and would allow material to fall through to the hearths below before it had been sufficiently calcined. On this account furnaces would sometimes have to be shut down for repairs after running for only a few days. The hearths would generally last a great deal longer than this, but it was never possible to tell when it would be necessary to close down for repairs.

To avoid the above disadvantages of brick hearths, the substitution of concrete hearths was suggested by Mr. Corwin. As an experiment, the third and fourth brick hearths were removed from MacDougall furnace No. 6 and concrete hearths substituted in their place. The third hearth was selected because it undergoes probably the greatest variation in temperature during regular operation, and the fourth

hearth because it is probably subjected to the greatest variation in temperature on starting up and shutting down the furnace. The material from which the concrete for the hearths was made was as follows:

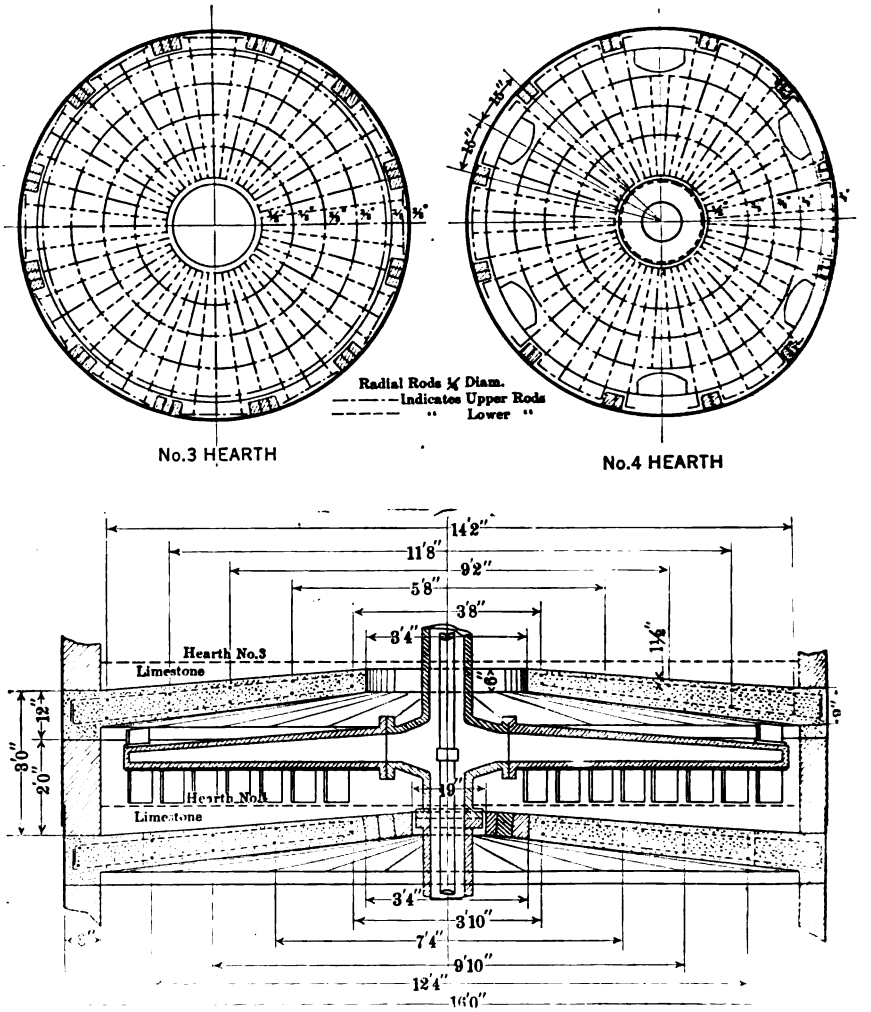



FIG. 6.—CONCRETE CONSTRUCTION OF HEARTH FOR MACDOUGALL FURNACE.

| Material. | Amount. |
|---|---|
| Portland cement, | 1 part. |
| Tailings sand, | 2 parts. |
| Crushed slag, | 4 parts. |

The hearths were installed by first building wooden forms in the furnace and then putting the concrete in on the forms. These hearths were 9 in. thick where they joined the shell of the furnace, decreasing to 6 in. in thickness near the center shaft. The concrete was reinforced by two series of concentric iron rings tied together by radiating iron rods. One series of concentric rings and radiating rods was buried in the concrete about 1.5 in. above the bottom of the hearths, the other about 1.5 in. below the top. The inner one of the concentric iron rings in each series was made of 0.5-in. iron rod, the two ends fastened together, and the outer rings of $\frac{3}{8}$ in. iron rod. The radiating iron rods were all 0.25 in. in thickness, 24 in each series. The method of reinforcing the concrete is shown in detail in Fig. 6.

After putting in the third and fourth hearths of concrete, MacDougall furnace No. 6 was started on July 24, 1912. In November the furnace was shut down and the second and fifth hearths changed from brick to concrete, the furnace being started again on Nov. 23, 1912. From November, 1912, to the present date (May, 1913), the furnace has been in continuous operation except for short shutdowns to repair the machinery, etc. The hearths are still in as good condition as when first installed, showing no cracks or wear of any kind. The heavy incrustations that form on the roofs of the hearths are removed more easily, as they do not adhere as tightly to the smooth surface of the concrete as to the rough surface of the brick hearths. The hearths give every appearance of being practically indestructible, and the indications are that the MacDougall furnaces, when constructed throughout with concrete hearths, will run practically independent of masonry repairs.

## XI. Conclusion.

As has been stated, systematic experimental work to increase the efficiency of the MacDougall furnaces was first started in May, 1910. Below are presented in parallel comparison data on tonnage treated in the MacDougall department for the months of April, 1910, and April, 1913. Some figures are given showing the reduction in labor required for operating the department. The figures on labor do not include foremen, trammers, oilers, samplers, and so on, as the number of men employed in such positions is practically independent of the capacity of the furnaces. The figures include only the occupations directly connected with the running of the furnaces where a decrease in the number of furnaces in operation means a decrease in the number of men required.

| | April, 1910. | April, 1913. |
|---|---|---|
| Number of days McDougall department in operation, | 30 | 30 |
| Total tons cupreous material treated during month, | 17,778.4 | 18,369.4 |
| Number of furnace days, | 386.9 | 238.4 |
| Average number of furnaces in operation per day, | 12.9 | 7.95 |
| Tons cupreous material treated per furnace day, | 46.0 | 77.1 |
| Average per cent. S in calcines, | 7.9 | 8.9 |
| Partial List of Labor used in Operating the McDougall Department. | | |
| Total number of furnacemen during month, | 270 | 180 |
| Average number of furnacemen per 24 hr, | 9 | 6 |
| Average number of furnacemen per 100 tons cupreous material treated, | 1.52 | 0.93 |
| Total number of feeders during month, | 105 | 80¾ |
| Average number of feeders per 24 hr., | 3½ | 2¾ |
| Average number of feeders per 100 tons cupreous material treated, | 0.59 | 0.44 |
| Total number of laborers pulling lumps during month, | 90 | ..... |
| Average number of laborers pulling lumps per 24 hr, | 3 | ..... |
| Average number of laborers pulling lumps per 100 tons cupreous material treated, | 0.51 | ..... |

From the above figures it will be seen that the MacDougall department treated a larger tonnage of cupreous material in April, 1913, with an average of 7.9 furnaces operating per day, than in April, 1910, with an average of 12.9 furnaces operating per day. Due to the fewer furnaces running in April, 1913, there was a saving in labor over April, 1910, of three furnacemen per 24 hr., three laborers pulling lumps per 24 hr., and $\frac{5}{6}$ feeder per 24 hr.

The experimental work in connection with the MacDougall department resulted not only in increasing the capacity of the furnaces, but also in increasing the percentage in first-class ore screenings which could be handled in the feed. It brought about the development of a furnace producing a much smaller percentage of flue dust, and the substitution of reinforced concrete in place of brick hearths. All of the above changes tended to decrease the daily operating expense of the department. These improvements were made without building any new furnaces or enlarging the old ones, and are interesting as showing what can be accomplished by systematically studying the conditions governing a smelting department.

Valuable general information about MacDougalls and about MacDougall practice at the Great Falls and Washoe smelters will be found in the *Practice of Copper Smelting*, by E. D. Peters, and in the following papers:

The Washoe Plant of the Anaconda Copper Mining Co. in 1905, by L. S. Austin, *Trans.*, xxxvii., 431 (1906).

Progress in the Metallurgy of Copper, by L. S. Austin, *Mineral Industry*, vol. xv. (1906).

Notes on the Metallurgy of Copper in Montana, by H. O. Hoffman, *Trans.*, xxxiv., 258 (1903).

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.
[SUBJECT TO REVISION.]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Butte meeting, Aug. 18 to 21, 1913, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1913, when Vol. XLVI. of the *Transactions* will go to press. Any discussion offered thereafter should preferably be in the form of a new paper for publication in Vol. XLVII. (with suitable cross references in both volumes).

# Notes on the Metallography of Refined Copper.

BY EARL S. BARDWELL, GREAT FALLS, MONT.

(Butte Meeting, August, 1913.)

THE structural relations existing between cuprous oxide and copper were first systematically studied by Heyn[1], who suggested that a study of the microstructure of refined copper might be substituted for the analytical methods in vogue for determining the oxygen content of copper coming from the refining furnace. Hofman, Green and Yerxa,[2] in their paper A Laboratory Study of the Stages in the Refining of Copper, working along the lines suggested by Heyn, showed that a study of the microstructure could be made to yield a correct estimate of the oxygen content of the specimen of refined copper under examination.

In order to determine the oxygen content of the specimen of refined copper under examination a photomicrograph was first made. From the relative areas of copper and eutectic, the percentage of cuprous oxide and the corresponding percentage of oxygen were then readily calculated. It was found that the values for oxygen thus obtained corresponded closely with the analytical results.

With the idea of utilizing this method of determining the oxygen content of each charge dipped from the refining furnaces at the Boston & Montana Reduction Plant, the physical testing laboratory was equipped with suitable apparatus for carrying out the work. It may be of interest to describe certain of the methods which have been worked out in this connection and give such data as have been obtained bearing on the effect of the oxygen content of refined copper upon its conductivity.

*Apparatus.*—For the benefit of those who may be unfamiliar with metallographical apparatus and methods and may be interested to know what equipment is required, a few words with regard to the apparatus is in place at this point.

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[1] *Mittheilungen aus den Königlichen Versuchsanstalten zu Berlin*, vol. xviii., p. 315 (1900). See also *The Metallographist*, vol. vi., p. 49 (1903).

[2] *Trans.*, xxxiv., 671 (1903).

The polishing head shown in Fig. 1 is provided with two polishing disks, one on either side. These polishing disks are driven by a direct-current variable-speed motor, direct connected. The optical apparatus shown in Fig. 2 consists of a Bausch & Lomb compound microscope with suitable combinations of eyepieces and objectives, a Bausch & Lomb direct-current self-regulating arc, and a camera stand and camera.

*Samples.*—From each charge dipped from the refining furnaces two test bars are dipped, one when dipping is started and one at or near the end. In the case of wire-bar charges two intermediate bars are dipped. These test bars have the same cross-section as the wire bars, but are shorter. From these test bars are sawed the samples which are drawn into wire for the physical tests; *i. e.*, tensile strength, torsion, and conductivity. At the same time a piece about 0.5 in. in cross-section and 4 in. in length is sawed out for the metallographical laboratory. A piece about 0.5 in. cube is taken for microscopical examination. One specimen from each charge is examined, except in case of a charge which shows a low conductivity, in which case the last test bar is also examined and the oxygen content determined. In this way we know the best and the worst about every charge dipped from the furnaces and are able to retain a permanent record of each charge in the form of photomicrographs.

*Polishing.*—The specimen is polished in the usual way. If the polishing has been done carefully the structure can be seen faintly with the naked eye. If upon examination under the microscope the specimen appears satisfactory it may be etched in such a way as to bring out the structure very distinctly.

*Etching.*—To bring out the structure sufficiently well for the taking of a satisfactory photomicrograph is a difficult matter. A suitable method for etching seemed, therefore, desirable. Nitric acid and ammonia were tried; but while occasionally good results were obtained, neither reagent seemed to be suited to our work. Finally the experiment of heating a polished specimen in a current of dry hydrogen gas was tried. The results obtained were startling. This reagent offers the advantage, unusual among etching reagents, of attacking the cuprous oxide without in any way affecting the copper. The reaction upon which the method depends is $Cu_2O + H_2 \leftrightharpoons 2\,Cu + H_2O$. This reaction is reversible, and the exact conditions necessary for obtaining the best results have not as yet been worked out. The indications are that to obtain the best results the specimen should be heated at about 300° C. and allowed to cool in a current of hydrogen. Excellent results have been obtained by heating in this way for 4 or

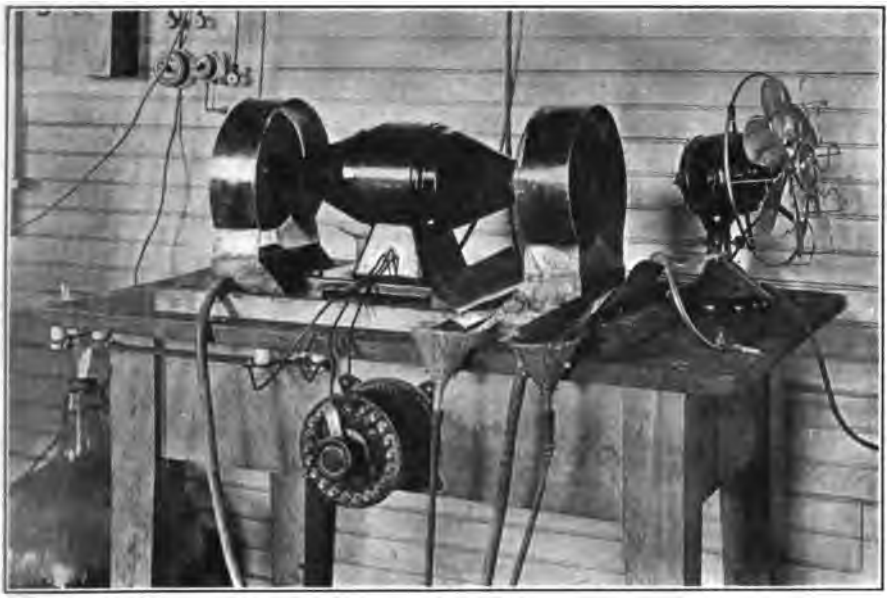

FIG. 1.—POLISHING WHEEL.

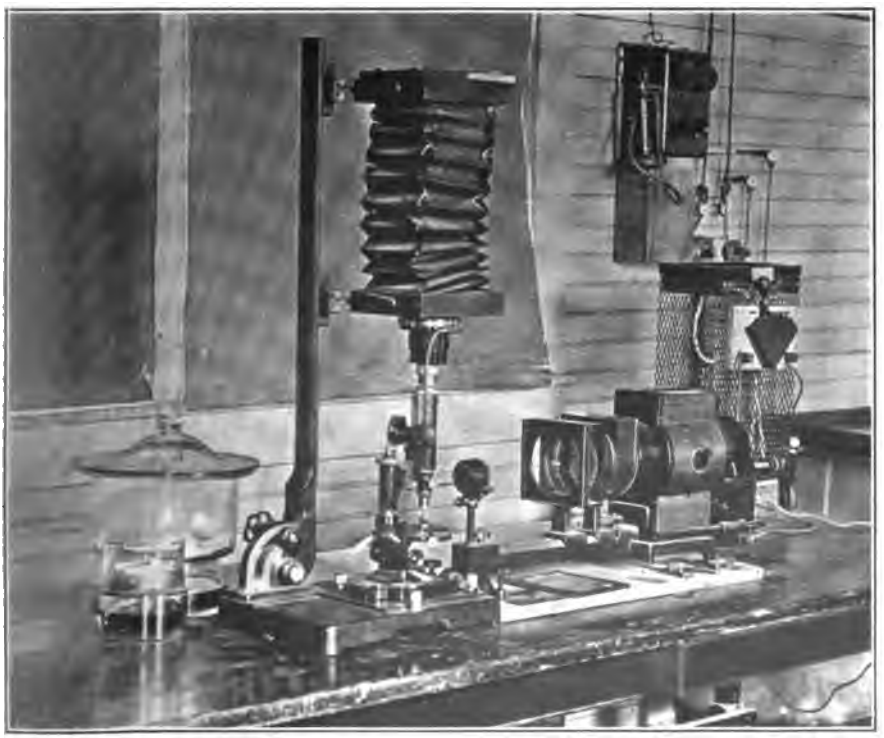

FIG. 2.—METALLOGRAPHICAL APPARATUS.

5 min. Heating to too high a temperature or cooling too suddenly tends to cause cracks in the polished surface, varying from cracks visible to the unaided eye to those visible as fine hair lines under the microscope. The latter are not serious if they are not so numerous as to obscure the structure. The structure in the case of the etched specimen is clearly visible to the naked eye, the network of eutectic appearing as if depressed below the surface of the rest of the specimen, while the copper areas stand out in relief. Under the microscope the eutectic network stands out black on a very nearly white background, and may be photographed with results that are extremely gratifying.

The apparatus required for the etching consists of a Kipp hydrogen generator, a wash bottle containing a concentrated solution of caustic soda, a drying tower containing stick caustic soda, a U-tube containing calcium chloride, a silica tube in which the specimen is placed for heating, and another U-tube containing calcium chloride.

Where the specimen is to be etched with hydrogen it has been found possible to simplify the polishing somewhat, as the etching may be depended upon to bring out the structure. When the specimen shows the structure under the microscope reasonably well it is ready to be etched. The silica tube is disconnected, and the polished specimen, which should be clean and dry, is inserted. The silica tube is then reconnected and hydrogen passed through the apparatus at the rate of 12 to 15 bubbles per minute for 10 min., in order to displace any air that may be contained in the apparatus. The silica tube is then heated by means of a Bunsen flame directed against the tube at the point where the specimen is located. The tube should not be allowed to get hotter than a very low red heat. Heating is continued for about 3 or 4 min., at the end of which time the flame is removed and the tube allowed to cool to room temperature with the hydrogen still passing. The apparatus must not be disconnected until the tube has become quite cool, otherwise there is danger of the specimen becoming oxidized or tarnished. The presence of moisture in the silica tube when the specimen is cooling also seems to cause a tarnish. This is due to the reversible nature of the reaction upon which the operation is dependent, and is obviated by having a sufficiently rapid circulation of dry hydrogen gas to sweep the water resulting from the primary reaction over into the calcium chloride tube.

The specimen upon being removed from the tube will be found to have retained its polish, but the structure will now be clearly visible to the unaided eye. The little cuprous oxide areas in the eutectic have been robbed of their oxygen so as to cause innumerable minute

pits, and it is the shadows which are cast in these pits under reflected light that bring out the detail when examined under the microscope. The eutectic areas thus appear black when illuminated by reflected light, while actually they are no different in color than the rest of the specimen. The polished surface under the microscope is all of one color; *i. e.*, the color of pure copper.

The photomicrograph is now taken in the usual manner, which I do not need to describe in detail.

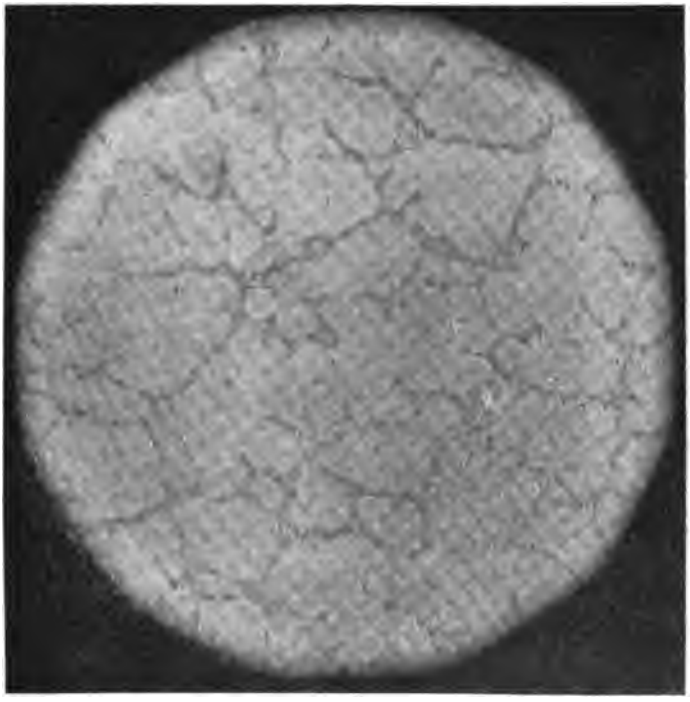

FIG. 3.—REFINED COPPER. UNETCHED. 0.071 PER CENT. OF OXYGEN. 87 DIAMETERS.

Fig. 3 represents one of the best photomicrographs which we were able to obtain without etching. This specimen contained 0.071 per cent. of oxygen. Fig. 4 shows the photomicrograph of a specimen containing 0.0715 per cent. of oxygen which has been etched by means of hydrogen. Fig. 5 is the photomicrograph of a specimen similarly etched but containing 0.1409 per cent. of oxygen.

*Estimation of Oxygen Content.*—The measurement of the eutectic after the manner outlined by Hofman, Green, and Yerxa is a tedious process, requiring from 1.5 to 2 hr. Eye estimation, on the other hand, while a valuable aid in controlling the operation of a furnace, is unsatisfactory from the point of view of accuracy and the making

of permanent records. One can very readily get a relative estimate of the amount of oxygen present in the specimen under examination, but two observers are liable to differ considerably in their respective estimates. In order to eliminate the personal equation as far as possible, a method that is capable of yielding an accurate estimate is desirable. In order to be available for practical purposes the method must, moreover, be capable of giving not only fairly accurate results, but results that are quickly obtainable.

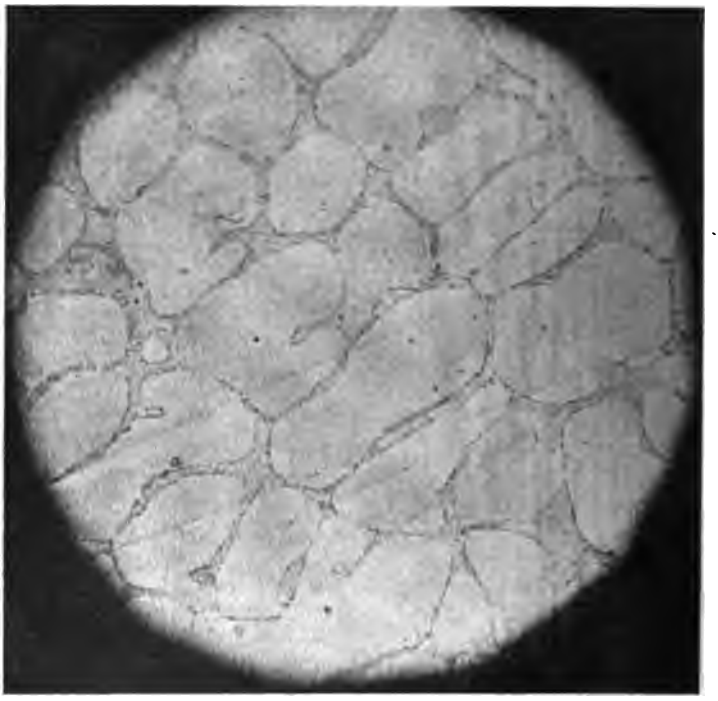

FIG. 4.—REFINED COPPER. ETCHED BY HYDROGEN. 0.0715 PER CENT. OF OXYGEN. 87 DIAMETERS.

The method at present in use consists in projecting the image, microscopic field, directly upon a piece of "Duplex" paper so as to cover a circle 15 or 16 in. in diameter. When the image has been sharply focused on the paper the outlines are traced lightly with a hard pencil. This gives directly an enlarged reproduction of the microscopic field. The copper areas are next cut out with a sharp knife. We now have a network of paper representing eutectic and some odd shaped pieces of paper representing copper. The two lots of paper are then weighed on chemical balances, and as the weights obtained are directly proportional to the areas, we have the equivalent of the planimetric method, with very little expenditure of

time. The oxygen is then computed in the same manner as that outlined by Hofman, Green, and Yerxa. Working in this way, it is possible to make four or five determinations in an hour. Separate determinations made on the same test bar will ordinarily be found to give results checking within 6 or 7 per cent.

The experiment was tried of taking a button sample at the same time that the test bar was dipped, and determining the oxygen in both the button sample and the test bar. Two instances of this sort gave results as follows:

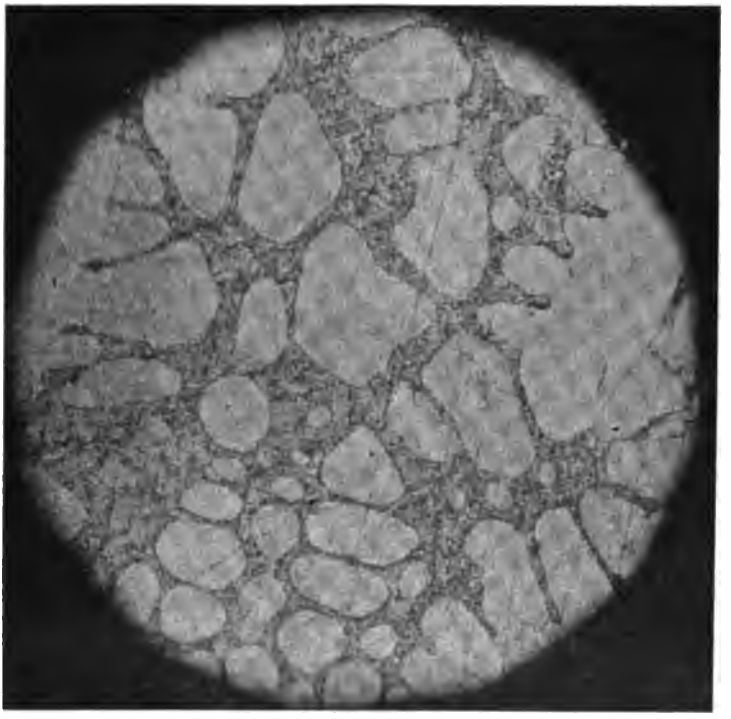

FIG. 5.—REFINED COPPER. ETCHED BY HYDROGEN. 0.1409 PER CENT. OF OXYGEN. 87 DIAMETERS.

*Oxygen Content by Measurement Method.*

| Charge No. | Button Sample. Per Cent. | Test Bar. Per Cent. |
|---|---|---|
| 2-132-4 | 0.071 | 0.076 |
| 2-133-4 | 0.092 | 0.087 |

Crude as the method appears at first sight, it is fully as accurate as the planimetric method and is infinitely simpler to work than the standard analytical method. Two examples will serve as a comparison between the results obtained by the measurement method and the results obtained by the usual hydrogen combustion method.

*Oxygen Content.*

| Charge No. | Measurement. Per Cent. | Hydrogen Combustion. Per Cent. |
|---|---|---|
| 2-192-2 | 0.0446 | 0.0492 |
| 2-188-2 | 0.0715 | 0.0675 |

A photomicrograph of the charge numbered 2–188–2 is given in Fig. 4.

The hydrogen combustion method for determining oxygen in refined copper is a difference method and is liable to all the inaccuracies of methods of that type. The measurement method is a direct method and should yield more accurate results.

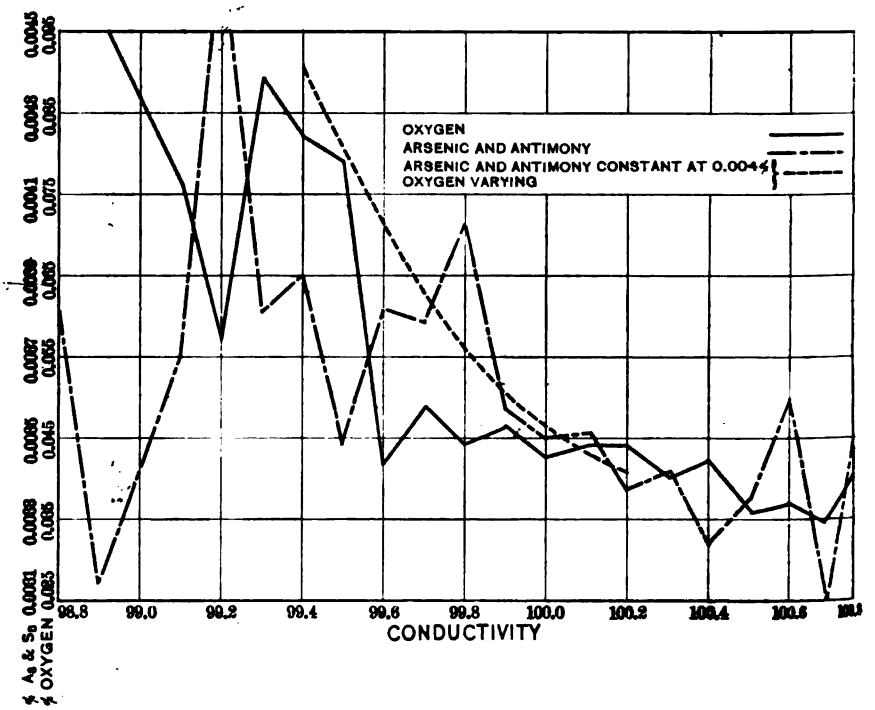


FIG. 6.—CONDUCTIVITY CURVE OF REFINED COPPER.

The image may be projected on the paper in either of two ways. First, with a sufficiently powerful arc the image may be projected directly through the microscope eyepiece. This necessitates special arrangements for darkening the room and for shielding the screen from light reflected from the arc. Second, the photographic negative may be used as a lantern slide and the image projected through it upon the screen. The latter method is the one usually adopted. The room need not be darkened, aside from pulling down shades in the portion of the room where the work is going on.

*Effect of Oxygen on the Conductivity of Refined Copper.*—By means of the measurement method the oxygen has been determined in about 300 furnace charges, and as we know the conductivity corresponding to each oxygen determination, we are able to present data which are of considerable interest. The curve, Fig. 6, is plotted with conductivities as abscissæ and oxygen content as ordinates. The content in combined arsenic and antimony is here similarly plotted against the conductivity. In making this curve the oxygen content of all samples having the same conductivity has been averaged, and similarly the arsenic and antimony content. The total number of results entering into each average is shown in the following table.

| Conductivity. Per Cent. | Number of Determinations. Oxygen. | Arsenic and Antimony. |
|---|---|---|
| 98.9 | 3 | 3 |
| 99.0 | 0 | 0 |
| 99.1 | 2 | 2 |
| 99.2 | 2 | 2 |
| 99.3 | 8 | 8 |
| 99.4 | 7 | 7 |
| 99.5 | 9 | 9 |
| 99.6 | 4 | 4 |
| 99.7 | 10 | 10 |
| 99.8 | 14 | 14 |
| 99.9 | 21 | 21 |
| 100.0 | 31 | 31 |
| 100.1 | 33 | 33 |
| 100.2 | 35 | 36 |
| 100.3 | 44 | 43 |
| 100.4 | 32 | 32 |
| 100.5 | 23 | 22 |
| 100.6 | 12 | 12 |
| 100.7 | 3 | 3 |
| 100.8 | 3 | 3 |

Looking at the curve, we see that while a decrease in oxygen content coincides with an increase in conductivity, the curve is very irregular. It is evident that the conductivity depends upon the combined effect of the impurities and not on the oxygen content alone. As the percentages of the several impurities vary with each charge, some method of studying the effect of oxygen alone seemed to be required. While attempting to ascertain the effect of casting temperature on the physical properties of refined copper a ladleful of the molten metal was held until a portion of the metal had solidified, accompanied, naturally, by an increase of the oxygen content of the mother metal. It was found, however, that the other impurities in the mother metal remained constant, the oxygen alone varying. Thus the phenomenon of selective freezing seemed to offer a method of ap-

proaching the problem. Pressure of other work has thus far prevented following this line out. I shall, however, venture to present the results that were obtained, in hope that they may prove suggestive to some one working along similar lines.

A ladleful of metal was dipped from the furnace and a test bar cast from it. After casting this first bar the ladle was allowed to stand for about 1.5 min., at the end of which time a second bar was cast. After 1.75 min. a third bar was cast. Photomicrographs of specimens from each of these bars, 1, 2, and 3, are shown in Figs. 7, 8, and 9. In the photomicrographs the gradual increase in oxygen will be noted. The following table gives complete data as to physical tests and chemical composition:

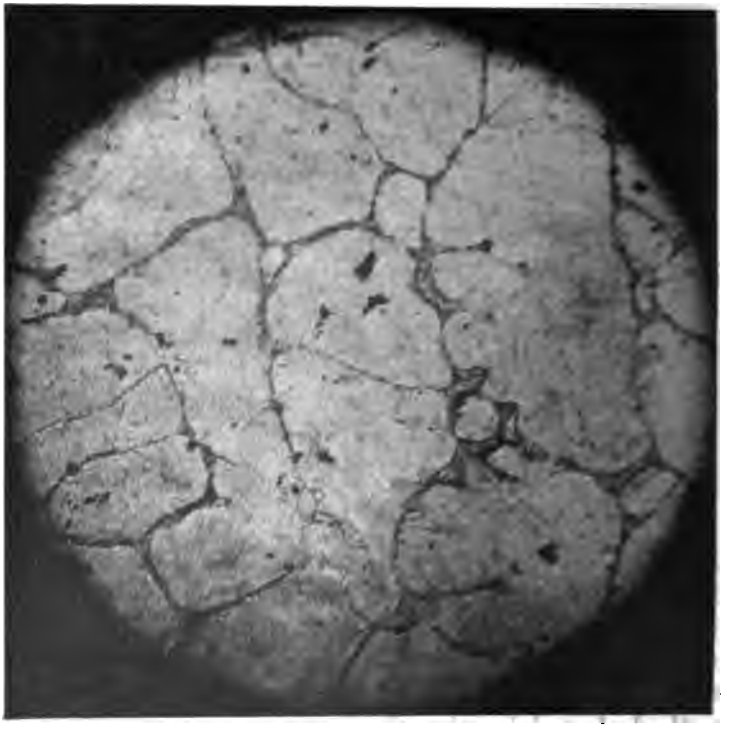

FIG. 7.—BAR NO. 1.

*Physical Properties.*

| | Bar 1. | Bar 2. | Bar 3. |
|---|---|---|---|
| Tensile strength, lb. per sq. in..... | 66,280 | 65,540 | 65,880 |
| Per cent. elongation in 5 ft..... | 1.2 | 1.0 | 1.1 |
| Conductivity: | | | |
| Hard drawn..... | 97.2 | 96.8 | 96.4 |
| Annealed..... | 100.2 | 99.8 | 99.4 |

*Chemical Composition.*

| | | | |
|---|---|---|---|
| Arsenic and antimony, per cent... | 0.0043 | 0.0037 | 0.0036 |
| Silver, ounces per ton..... | 1.00 | 1.00 | 1.00 |
| Cuprous oxide, per cent..... | 0.365 | 0.495 | 0.815 |
| Equivalent oxygen, per cent..... | 0.0408 | 0.0555 | 0.0912 |

The question will probably arise as to how much of the increase in oxygen is caused by the oxygen in the atmosphere. Owing to the considerable amount of metal in the ladle and the fact that the metal was freezing, I shall assume that atmospheric oxygen played a negligible part in the experiment.

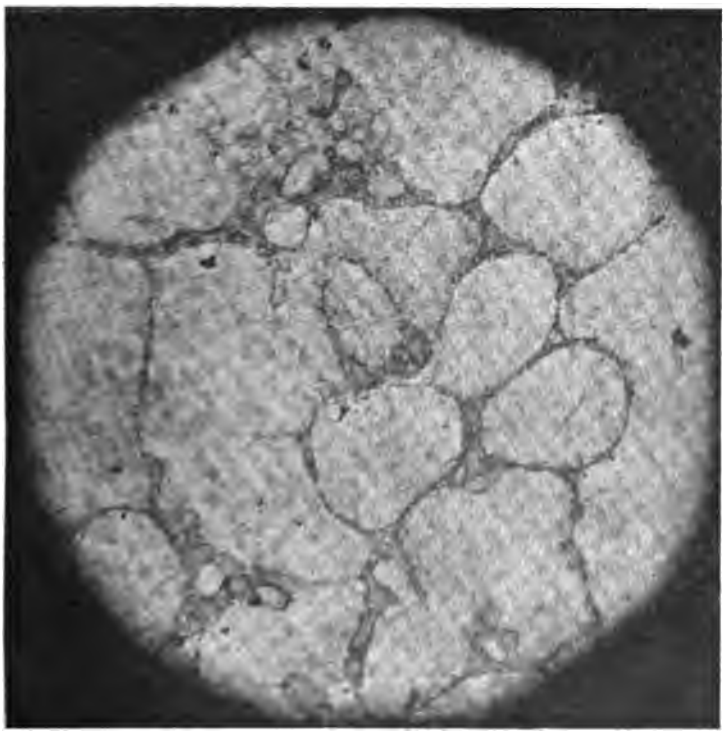

FIG. 8.—BAR No. 2.

Fig. 10 gives the freezing point curve of copper-cuprous oxide alloys plotted from data obtained by Heyn in 1900. Only a portion of the curve is shown, as alloys containing more than 8 per cent. of cuprous oxide are of no practical importance and do not concern us here. The line *a a'*, Fig. 10, represents the cooling of Bar 1 (see Fig. 7). When temperature *a'* is reached copper separates out, and the composition of the mother metal varies along the branch of the curve marked "Freezing Point of Copper" until it reaches a composition represented by 3.45 per cent. of cuprous oxide, whereupon it

solidifies as the eutectic mixture of cuprous oxide and copper, the dark network shown in the photomicrographs. Now, very shortly after the first bar was dipped the metal in the ladle reached the temperature at which it began to freeze in a like manner to that described above in the case of the test bar.

At the time of dipping the second bar the mother metal in the ladle had reached a composition represented by *b′ b*. The oxygen in the second bar is 0.0555 per cent., as against 0.0408 per cent. in the case of the first bar, and the conductivity has dropped from 100.2 to

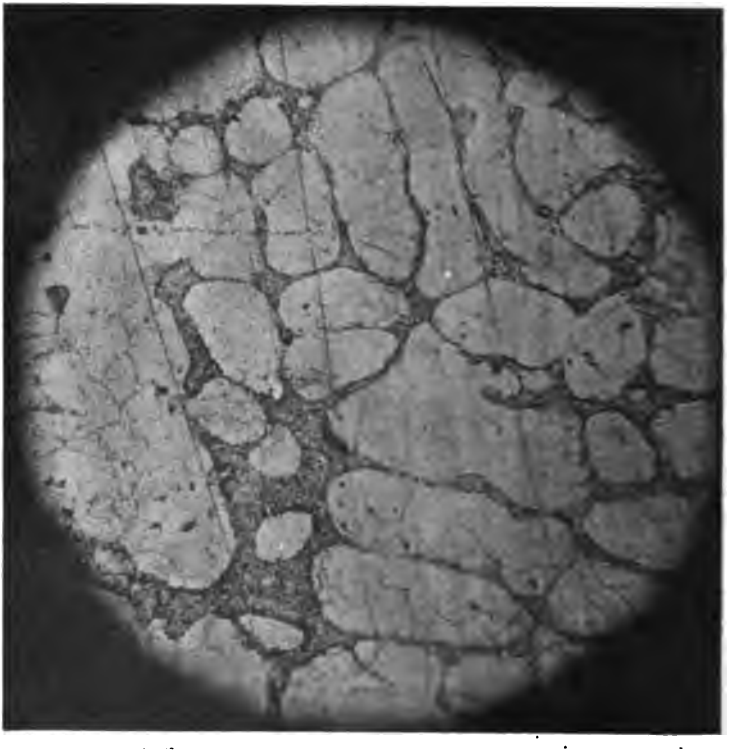

FIG. 9.—BAR NO. 3.

99.8 per cent. The mother metal in the ladle continued to freeze selectively until finally when the mother metal had reached the composition *c′ c* the third bar was dipped. This bar contained 0.0912 per cent. of oxygen and had a conductivity of 99.4 per cent. The silver, as will be noted, remains the same in each of the three bars. This is probably due to the fact that silver is slightly soluble in copper in the solid state and is capable of forming mixed crystals with copper to a limited extent. Under the conditions of the test silver would, therefore, separate out with the copper. The solubility of

silver in copper is evidently such that the mother metal becomes neither richer nor poorer in silver as the metal freezes. The arsenic and antimony varies from 0.0043 per cent. in the first bar to 0.0036 per cent. in the third. The accuracy of this determination is such that the arsenic and antimony might be said to be the same in each of the three bars. This again seems to be due to the fact that the compounds which arsenic and antimony form with copper are to a certain extent soluble in copper in the solid condition. With the other impurities remaining constant, or practically so, the falling off in conductivity as we pass from the first bar to the third seems to be due to oxygen alone.

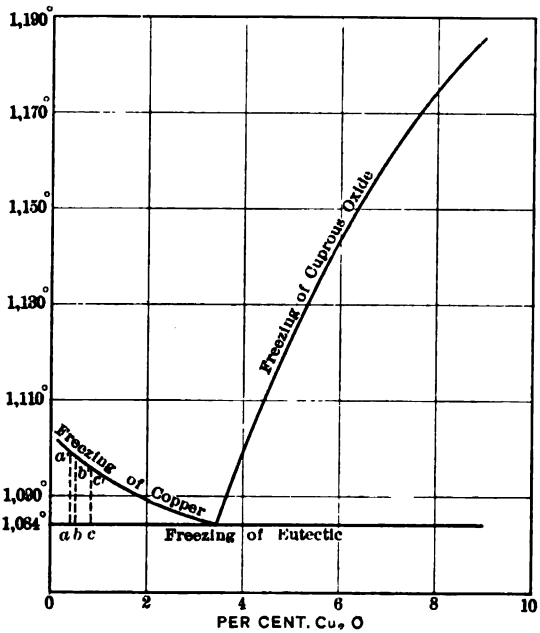


FIG. 10.—FREEZING-POINT CURVE OF COPPER-CUPROUS OXIDE ALLOYS.

These results have been plotted on the curve Fig. 6 and, as will be seen, the points lie along a smooth curve. By comparing a series of tests of this kind made upon different furnace charges with varying amounts of impurities it should be possible to obtain considerable information concerning the effect of oxygen on conductivity in the presence of varying amounts of other impurities, as well as perhaps valuable information concerning the interrelation of oxygen and the other common impurities with regard to their effect on conductivity.

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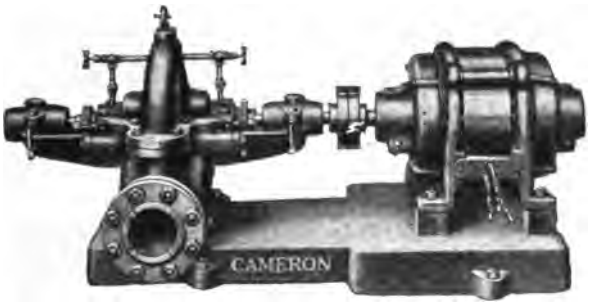


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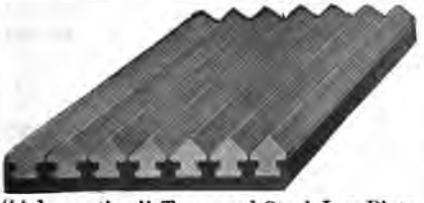

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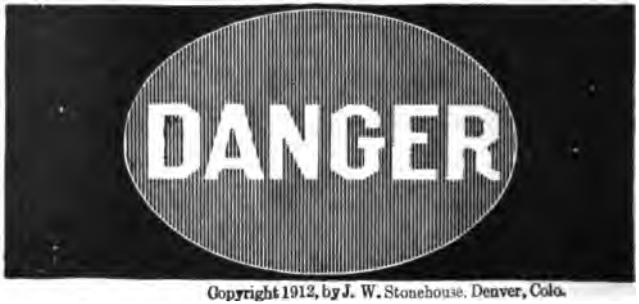


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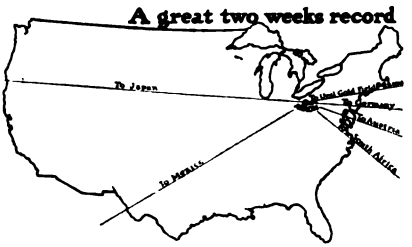


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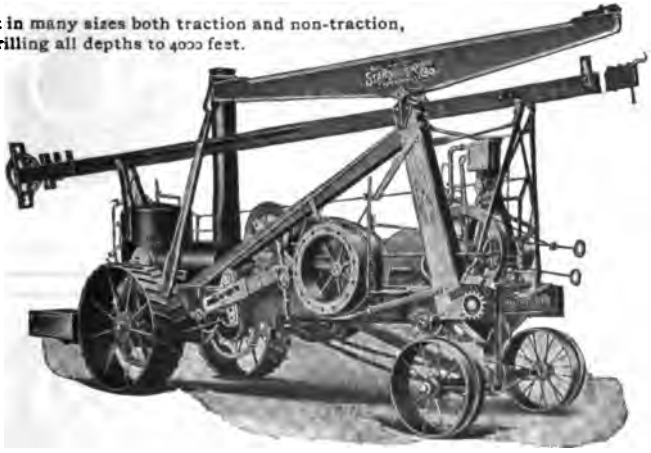

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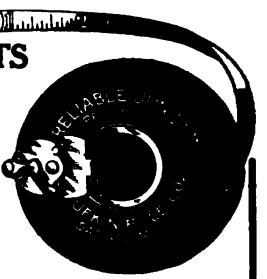

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REFERENCE
LIST

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**ALDRIDGE, WALTER H.**
Consulting Mining and Metallurgical Engineer
14 Wall Street NEW YORK

**ARMSTEAD, Henry Howell**
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AMERICAN INSTITUTE OF MINING ENGINEERS
29 West 39th Street, New York, N. Y.

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_____
} Signatures of three Members or Associates.

*Place of birth* _____ *Year of birth* _____

*Education, general and technical, when, where and how acquired, with degrees, if any.*

| *Dates* | |
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*Record of experience. Briefly, the past and present employment, with names of employers, companies and associates. (Proper names, names of companies, etc., should be written without abbreviations.)*

| *Dates* | |
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*Present position*_____

_____ *Signature*

*Dated*_____ *191*

## EXTRACTS FROM THE CONSTITUTION.

### ARTICLE II.—MEMBERS.

SEC. 1. The membership of the Institute shall comprise four classes, namely: 1. Members; 2. Honorary Members; 3. Associates; 4. Junior Members. * * *

SEC. 2. The following classes of persons shall be eligible for membership in the Institute, namely: as Members, all professional mining engineers, geologists, metallurgists, or chemists, and all persons actively engaged in mining and metallurgical engineering, geology, or chemistry; as Associates, all persons desirous of being connected with the Institute who in the opinion of the Board of Directors are suitable.

As Junior Members, all students in good standing in engineering schools who have not taken their degrees and who are nominated by at least two of their instructors. * * *

Every candidate for election as a Member, Associate, or Junior Member must be proposed for election by at least three Members or Associates, must be approved by the Committee on Membership, as prescribed in the By-Laws, and must be elected by the Board of Directors.

No. 80 AUGUST 1913

# Bulletin of the American Institute of Mining Engineers.

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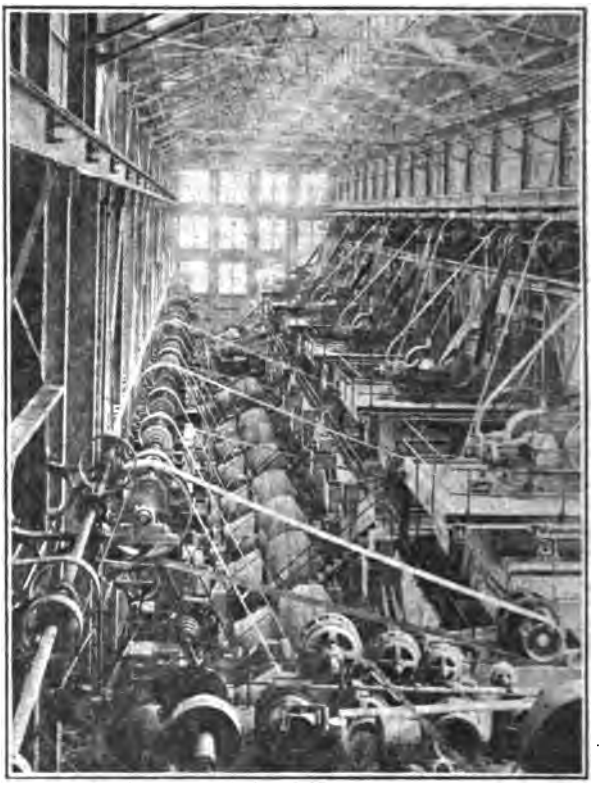
TECHNICAL JOURNALS AND OTHERS DESIRING TO REPUBLISH ARTICLES CONTAINED IN THIS *BULLETIN* SHOULD APPLY FOR PERMISSION TO THE EDITOR, AT 29 WEST 39TH STREET, NEW YORK, N. Y.



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# BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.

No. 80 AUGUST 1913

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Entered as second class matter, October 16, 1911, at the post office at Philadelphia, Pa., under the Act of March 3, 1879.

One of the results of the work of the Committee on Precious and Base Metals is that the present *Bulletin* is as large as some of the annual volumes of *Transactions* of the Institute, and we extend our compliments and thanks to the Committee and express to the members thereof and to the authors, who have so bounteously given of their time and information, the sincere gratitude of the Institute. In order to get all these papers printed and distributed to the membership in advance of the meeting, it was necessary to divide the work between two printers. This accounts for the discrepancy in the pagination of the *Bulletin;* and we also ask the indulgence of our readers in case errors have crept into this preliminary edition during the period of unusual pressure.

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**Program of the Butte Meeting.**—The program of the Butte meeting will include six technical sessions for the presentation and discussion of papers and five half-day visits to mines and metallurgical works. The following program is practically in final form, with slight additions which may be necessary later. Every effort will be made to have papers presented and discussed on scheduled time, so that members who are unable to attend the entire meeting may so plan their trip as to hear and discuss papers in which they are especially interested.

*Tentative Program of Butte Meeting.*

**Saturday, August 16.** Arrive Great Falls, 8.30 a.m.
Arrive Giant Springs, 11.00 a.m.
Luncheon at Rainbow Hotel.
Leave for Boston and Montana Smelter, 2.30 p.m.

*Technical Session, 8.30 p.m.*

Max Hebgen, Hydro-Electric Development in Montana.[3]
W. T. Burns, Notes on the Great Falls Electrolytic Plant.[3]
Charles W. Goodale and John H. Klepinger, The Great Falls Flue System and Chimney.
E. E. Brownson, The Determination of Arsenic and Antimony in Converter and Electrolytic Copper.[3]
A. E. Wheeler and M. W. Krejci, Great Falls Converter Practice.[3]
Dorsey A. Lyon and Robert M. Keeney, The Smelting of Copper Ores in the Electric Furnace.[3]
S. E. Bretherton, Preparation of Ore Containing Zinc for the Recovery of Other Metals such as Silver, Gold, Copper, and Lead.[3] Discussion by F. L. Wilson.
Edward Keller, Some Recent American Progress in the Assay of Copper-Bullion.[3]

*Note:* Members will please take the distributed copies of the August *Bulletin* with them to Butte, and, on Tuesday, to Anaconda. The supply of *Bulletins* in Montana is limited.

**Sunday, August 17.** *En route* to Butte. Optional stop at Helena.

**Monday, August 18.**

*Technical Session, 9.30 a.m.*

B. H. Dunshee, Timbering in the Butte Mines.[3]
Norman P. Braly, Shaft Sinking Methods at Butte.[3]
Reno H. Sales, Ore Deposits at Butte, Montana.[3]
D. C. Bard and M. H. Gidel, Mineral Associations at Butte.[3]
John Gillie, The Use of Electricity in Mining in the Butte District.[3]
Paul Klopstock, The Kennedy Mining District, Nevada.[1]
E. G. Woodruff, Topographic Maps for the Mining Engineer.[1] Discussion by Mortimer A. Sears.
Anton Eilers, Notes on the Occurrence of Some of the Rarer Metals in Blister Copper.[1]
E. J. Hall and C. W. Drury, Assay of Gold and Silver by the Iron-Nail Method.[1] Discussion by A. M. Smoot.
W. Spencer Hutchinson, An Assay for Corundum by Mechanical Analysis.[1]
Henry S. Drinker and Rossiter W. Raymond, Biographical Notice of John Fritz.[1]

Leave for visits to mines, 2.00 p.m.

*Technical Session, 8.00 p.m.*

F. W. Apgar, The Use of the Microscope in Mining Engineering.[1]
Edward P. Mathewson, Development of the Basic-Lined Converter for Copper Matte.[1] Discussion by R. Baggaley.
H. T. Van Ells, Mining Cost Accounts of the Anaconda Copper Mining Co.[1]
Frederick Laist, Roasting and Leaching Tailings at Anaconda, Mont.[2]
Theodore Simons, The Evolution of the Round Table for the Treatment of Metalliferous Slimes.[2]
J. A. Church, Jr., The Development of Blast-Furnace Construction at the Boston & Montana Smelter.[2]
H. M. Chance, Valuation of Coal Land.[2]
M. H. deHora, The Gold Placers of Antioquia, Republic of Colombia, South America.[1]
N. H. Thomson and L. T. Sicka, The Tooele Plant of the International Smelting & Refining Co.[2]
Robert Franke, Hardinge Mills vs. Chilean Mills.[2]
George F. Kunz, The New International Diamond Carat of 200 Milligrams.[2]
Blamey Stevens, The Laws of Jointing.[2]
Charles A. Mitke, Method of Testing Draeger Oxygen Helmets at the Copper Queen Mine.[2]
Bruno Nordberg, Electrically Generated Compressed Air in Butte Mines.

**Tuesday, August 19.** (Each member will please take a copy of the August *Bulletin* to Anaconda).

Leave for Anaconda Works, 8.00 a.m.

---

[1] Papers published in June *Bulletin*.
[2] Papers published in July *Bulletin*.
[3] Papers published in August *Bulletin*.

*Technical Session, 2.00 p.m.* (At Anaconda.)

Ralph Hayden, Concentration of Slimes at Anaconda, Mont.[3]
Albert E. Wiggin, The Great Falls System of Concentration.[3]
Robert Ammon, The Anaconda Classifier.[3]
E. S. Bardwell, Application of Hindered Settling to Hydraulic Classifiers.[3]
Edgar M. Dunn, Determination of Gases in Smelter Flues; and notes on the Determination of Dust Losses at the Washoe Reduction Works, Anaconda, Mont.[3]
James O. Elton, Arsenic Trioxide from Flue Dust.[3]
Robert H. Richards, Ore Dressing Improvements. (If time permits.)

Social evening in Butte, 8.00 p.m.

**Wednesday, August 20.**

*Technical Session, 9.30 a.m.*

W. McA. Johnson, The Reducibility of Metallic Oxides as Affected by Heat Treatment.[2]
William L. Saunders, Rock Drill Economics.
Robert P. Roberts, Thermal Effects of Blast Furnace Jackets.[2]
Paul Billingsley, The Southern Cross Mine, Georgetown, Mont.
C. Colcock Jones, The Discovery and Opening of A New Phosphate Field in the United States.
R. S. Handy, Milling vs. Hand Sorting of Lead Ore.

Leave for visits to mines, 2.00 p.m.

*Technical Session, 8.00 p.m.*

John C. Febles, The Precipitation of Copper from the Mine Waters of the Butte District.[2]
W. T. Burns, Notes on the Electrolytic Refining of Copper Precipitate Anodes.[2]
Frank R. Corwin and Selden S. Rodgers, Increasing the Efficiency of MacDougall Roasters at the Great Falls Smelter of the Anaconda Copper Mining Co.[2]
Milo W. Krejci, The Metaline Plant of the Inland Portland Cement Co., Metaline Falls, Wash.[2]
Earl S. Bardwell, Notes on the Metallography of Refined Copper.[2]
Howland Bancroft, The Tin Situation in Bolivia.

**Thursday, August 21.** Leave for Southern Cross Mine, 8.00 a.m.
Return via Gregson Springs (Swimming Pool).
Banquet, 8.30 p.m.

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[2] Papers published in July *Bulletin*.
[3] Papers published in August *Bulletin*.

**Nominations for Officers.**—The Committee appointed by the Board of Directors to nominate officers for the Institute for the year 1914 desires that every member participate in the work by offering suggestions and proposing names for the various offices, thereby obtaining the best ticket that can be recommended.

The officers to be elected at the annual meeting in February, 1914, are:

One officer, known as Director and President.
Two officers, known as Director and Vice-President.
Five officers, known as Director.

The attention of members is called to Articles V and VII of the Constitution, and to By-Law XIII, which reads as follows:

"The Geographical districts to be considered by the Committee on Nominations shall be as follows, until otherwise ordered by the Board.

District No. 1. New England, New York, and New Jersey, excepting New York City and district, which is provided for in the Constitution. [N. B.—New York City and district is designated District 0.]
District No. 2. Pennsylvania.
District No. 3. Ohio, Indiana, Illinois, Iowa, and Missouri.
District No. 4. Minnesota, Wisconsin, and Michigan.
District No. 5. Montana, North and South Dakota, Wyoming, Nebraska, Kansas, Washington, Oregon, Idaho, and Alaska.
District No. 6. California and Nevada.
District No. 7. Utah, Colorado, Arizona, and New Mexico.
District No. 8. Louisiana and Texas.
District No. 9. Other Southern States and District of Columbia.
District No. 10. Mexico.
District No. 11. Canada."

An excerpt from Article VII of the Constitution reads as follows:

"In making such selections [namely, the 8 Directors to be elected], the Nominating Committee shall, so far as practicable, distribute the representation on the Board geographically, so that seven members shall be residents of the district including New York City and the territory within a radius of fifty miles of the headquarters of the Institute, and one member a resident of each of the geographical districts enumerated in the By-Laws."

The officers of the Institute whose terms expire are as follows:
President, Charles F. Rand (not eligible for re-election).
Past President, Charles Kirchhoff.
Vice-Presidents, Karl Eilers, District 0; Waldemar Lindgren, District 1.
Directors, James Douglas, District 0; James Gayley, District 0; Albert R. Ledoux, District 0; Charles W. Merrill, District 6; and C. Snelling Robinson, District 3.

It is hoped that the members of the Institute will help the Committee to the utmost by making suggestions and proposals as requested above, which should be sent to the Chairman of the Nominating Committee, John Hays Hammond, 71 Broadway, New York, N. Y.

It is the intention of some of the members of the Nominating Committee to attend the meeting at Butte. This will enable visiting members to express their views in person.

JOHN HAYS HAMMOND,
*Chairman, Nominating Committee.*

## READY FOR DISTRIBUTION.

### EMMONS VOLUME ON ORE-DEPOSITS.

*Ore-Deposits—a continuation of the "Posepny" Volume.*

Comprising papers descriptive of ore-deposits and discussions of their origin, edited, with an introduction, by Dr. S. F. Emmons. The volume contains also a Biographical Notice of Dr. Emmons by his associate and friend, Dr. George F. Becker, and a comprehensive Bibliographical Index of the Science of Ore-Deposits, prepared by Prof. John D. Irving, of the Sheffield Scientific School of Yale University. Dr. Emmons had finished his editorial work and written his Introduction before his lamented death in 1910; and the Volume contains his last words upon the subject to which he had given the work of his life, and on which he was justly regarded as the foremost authority.

*Contents.*

Genesis of Certain Ore-Deposits. By S. F. EMMONS.
Structural Relations of Ore-Deposits. By S. F. EMMONS.
Geological Distribution of the Useful Metals in the United States. By S. F. EMMONS. Discussion, by JOHN A. CHURCH, ARTHUR WINSLOW, S. F. EMMONS, and WILLIAM HAMILTON MERRITT.
Torsional Theory of Joints By GEORGE F. BECKER. Discussion, by H. M. HOWE, R. W. RAYMOND, C. R. BOYD, and GEORGE F. BECKER.
Allotropism of Gold. By HENRY LOUIS.
Superficial Alteration of Ore-Deposits. By R. A. F. PENROSE, JR.
Some Mines of Rosita and Silver Cliff, Colorado. By S. F. EMMONS.
Genesis of Certain Auriferous Lodes. By JOHN R. DON. Discussion, by JOSEPH LE CONTE, S. F. EMMONS, G. F. BECKER, ARTHUR WINSLOW, W. P. BLAKE, and J. R. DON.
Influence of Country-Rock on Mineral Veins. By WALTER HARVEY WEED.
Igneous Rocks and Circulating Waters as Factors in Ore-Deposition. By J. F. KEMP.
Consideration of Igneous Rocks and Their Segregation or Differentiation as Related to the Occurrence of Ores. By J. E. SPURR. Discussion, by A. N. WINCHELL.
Chemistry of Ore-Deposition. By WALTER P. JENNEY. Discussion, by JOHN A. CHURCH.
Ore-Deposits near Igneous Contacts. By WALTER HARVEY WEED. Discussion, By W. L. AUSTIN.
Ore-Deposition and Vein-Enrichment by Ascending Hot Waters. By WALTER HARVEY WEED.
Basaltic Zones as Guides to Ore-Deposits in the Cripple Creek District, Colorado. By E. A. STEVENS.
Geological Features of the Gold-Production of North America. By W. LINDGREN. Discussion, by W. G. MILLER and W. L. AUSTIN.
Osmosis as a Factor in Ore-Formation. By HALBERT POWERS GILLETTE.
Ore-Deposits of Sudbury, Ontario. By CHARLES W. DICKSON.
Genesis of the Copper-Deposits of Clifton-Morenci, Arizona. By W. LINDGREN.
Copper-Deposits at San Jose, Tamaulipas, Mexico. By J. F. KEMP.
Magmatic Origin of Vein-Forming Waters in Southeastern Alaska. By A. C. SPENCER.
Genetic Relations of the Western Nevada Ores. By J. E. SPURR.
Are the Quartz Veins of Silver Peak, Nevada, the Result of Magmatic Segregation? By J. B. HASTINGS.
Occurrence of Stibnite at Steamboat Springs, Nevada. By W. LINDGREN.
Summary of Lake Superior Geology with Special Reference to Recent Studies of the Iron-Bearing Series. By C. K. LEITH.
Geological Relations of the Scandinavian Iron-Ores. By H. SJÖGREN.
Formation and Enrichment of Ore-Bearing Veins. (With Supplementary Paper.) By GEORGE J. BANCROFT.
Distribution of the Elements in Igneous Rocks. By H. S. WASHINGTON.
Agency of Manganese in the Superficial Alteration and Secondary Enrichment of Gold-Deposits of the United States. By W. H. EMMONS.
Cognate Papers.
Bibliography of the Science of Ore-Deposits. By J. D. IRVING, H. D. SMITH, and H. G. FERGUSON.

The volume contains 1002 pages. Price, bound in cloth, $5; in half morocco, $6. Both the Emmons and the Posepny Volumes on Ore-Deposits, bound in cloth, $8; in half morocco, $10.

## COMMITTEE ON PRECIOUS AND BASE METALS.

CHARLES W. GOODALE, *Chairman.*

L. D. RICKETTS, *Vice-Chairman.*

DARSIE C. BARD, *Secretary.*

Montana State School of Mines, Butte, Mont.

| | | |
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| Leonard S. Austin, | Samuel S. Fowler, | Willet G. Miller, |
| David W. Brunton, | Thomas J. Grier, | Albert F. Schneider, |
| Theodore B. Comstock, | Hennen Jennings, | George C. Stone, |
| Stanley A. Easton, | Edmund B. Kirby, | Benjamin B. Thayer, |
| | Charles W. Merrill. | |

The experience of the Committee in connection with securing papers for the Butte meeting in August has shown that authors should begin early on papers contemplated for the New York meeting next February. So many papers come in at the last moment that it becomes exceedingly difficult to have them all read, indorsed, and delivered to the printer in time. Authors should remember that those serving on this Committee are doing so without pecuniary compensation, and it is hardly fair to expect them to devote so much of their time at the last moment from their private affairs. So many papers have to be sent back for corrections of one kind or another, that it is desirable that they be sent to the Committee at least three months previous to the date of the meeting.

The Committee desires to be kept informed of contemplated papers on metallurgical subjects particularly. It is hoped that the Institute *Transactions* can be made an up-to-date record of metallurgical advance of all kinds. For the present, at least, the activities of the Committee include the refining and preparation of the non-metallic products such as phosphate rock, precious stones, cement, lime, gypsum, building stones, etc. Papers covering any of these subjects are desired.

General papers covering geological, mining and metallurgical phases of an ore deposit can be sent either to this Committee or to one of the other Committees on Mining Geology or Mining Methods.

C. W. GOODALE, *Chairman.*
DARSIE C. BARD, *Secretary.*

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*Privileges of Local Clubs of Montana.*—The Silver Bow Club of Butte and the Montana Club of Helena have extended to the members of the Institute and their guests the privileges and courtesies of the respective clubs.

## INTERNATIONAL GEOLOGICAL CONGRESS.

(For further particulars apply to the Secretary, 12th International Geological Congress, Victoria Memorial Museum, Ottawa, Canada; cable address, Geocong, Ottawa.)

**The Coal Resources of the World.**—Three quarto volumes of about 400 pages each, 8¼ by 10¾ in., and an atlas of about 70 maps in colors, 13½ by 19½ in., bound in heavy paper cover. Edition limited to 3,000 copies. One thousand copies will be reserved for members of the International Geological Congress, and the remainder of the edition will be distributed in the order in which applications are received. Price, $25 per set. Ready in July, 1913.

This book is an inquiry made upon the initiative of the Executive Committee of the Twelfth International Geological Congress, with the assistance of geological surveys and mining geologists of different countries, published by Morang & Co., Ltd., Toronto, Canada.

*Headquarters During Session at Toronto.*

The headquarters will be at the University of Toronto, where there will be a branch post-office open at all hours and at which registered mail may be received or despatched, money orders sent or cashed, etc. A bank will be established at which money may be exchanged. A typewriting service will be maintained, also a telephone service and messenger service. Railway and steamship agents will also be in attendance.

A restaurant service providing luncheon in the middle of the day will be arranged.

*Special Dates.*

Thursday, August 7.....Opening day of Session at Toronto.
Thursday, August 14...Last day of Session at Toronto. Excursions C. 1 and C. 2 start.
Tuesday, August 26.....Special day at Victoria, British Columbia.

*Railway Privileges.*

To professional members of the Congress and to dependent members of their families a special form of certificate will be given. The possession of this certificate will enable the person to whom it is issued to purchase and use during a limited period of time railway tickets at a much reduced cost, to and from any point in Canada.

Special railway rates are also provided for members attending the Congress from certain points on the Pacific coast of the United States.

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## INTERNATIONAL ENGINEERING CONGRESS.

The International Engineering Congress of 1915 will be held under the auspices of the American Society of Civil Engineers, American Society of Mechanical Engineers, American Institute of Electrical Engineers, Society of Naval Architects and Marine Engineers, and the American Institute of Mining Engineers, in connection with the Panama-Pacific Exposition. Arrangements for this Congress are now proceeding under the General Committee, the Institute representatives on which are given on p. xv. An interesting technical program is promised. The time of the Congress is Sept. 20 to 25, 1915. Members of the Institute are cordially invited to attend and participate in the proceedings of the Congress.

## AFFILIATED STUDENT SOCIETIES.

Any society of undergraduates at a technical school, comprising students in any branch of engineering, metallurgy, chemistry, geology, etc., may be recognized by the Board of Directors in its discretion as an Affiliated Student Society. A circular giving details of the plan of affiliation may be obtained on application to the office of the Secretary of the Institute.

The following societies have been placed by authority of the Board of Directors on the above list:

### AFFILIATED STUDENT SOCIETIES.

The Mining Society of the Sheffield Scientific School, Yale University, New Haven, Conn. *President*, Francis S. Winslow; *Secretary*, Roger W. Newberry.

The University of Illinois Student Branch of the American Institute of Mining Engineers, Champaign, Ill. *President*, M. L. Nebel; *Secretary*, Willis Leriche.

The Engineering Society of the University of Nevada, Reno, Nev. *President*, P. E. Raymond; *Secretary*, R. M. Seaton.

The University of Wisconsin Mining Club, Madison, Wis. *President*, Rudolph J. Stengl; *Secretary*, Mack C. Lake.

The Mining and Geological Society of Lehigh University, South Bethlehem, Pa. *President*, Robert C. Mickel; *Secretary*, Edward B. Snyder.

The School of Mines Society of the University of Minnesota, Minneapolis, Minn. *President*, C. A. Walker; *Secretary*, A. C. Bierman.

The Mining Engineering Society of the Massachusetts Institute of Technology. *President*, Ralph E. Wells, Jr.; *Secretary*, Louis W. Currier.

The Student Auxiliary Society of the American Institute of Mining Engineers of the University of Kansas, Lawrence, Kan. *President*, Walter E. Rohrer; *Secretary*, H. R. Brown.

The Associated Miners of the University of Idaho, Moscow, Idaho. *President*, Hallard W. Foester; *Secretary*, Walter P. Scott.

The State College of Washington Mining and Geological Society, Pullman, Wash. *President*, J. V. Quigley; *Secretary*, S. A. Swanson.

The Tejas Technical Society, School of Mines, University of Texas, Austin, Tex. *President*, G. C. Cartwright; *Secretary*, David S. Alley.

The Ohio State University Student Branch of the American Institute of Mining Engineers, Columbus, Ohio. *President*, Hugh B. Lee; *Secretary*, E. P. Elliott.

The Stanford Geology and Mining Society, Stanford University, Cal. *President*, R. E. Parsons; *Secretary*, E. D. Nolan.

The Senior Mining Society of Columbia University, New York, N. Y. *President*, Roger L. Strobel; *Secretary*, Clark G. Mitchell.

Mining Association of the University of California, Berkeley, Cal. *President*, D. M. Drumheller, Jr.; *Secretary*, J. B. Orynski.

Tufts College Chemical Society, Tufts College, Mass. *President*, P. G. Savage; *Secretary*, W. S. Frost.

University of Washington Mining Society, Seattle, Wash. *President*, Oliver P. Searing; *Secretary*, Albert R. Sherman.

Student Branch of the American Institute of Mining Engineers, Iowa State College, Ames, Iowa. *President*, Lyle A. Butler; *Secretary*, A. J. Crawford.

Missouri Mining Association of the Missouri School of Mines, Rolla, Mo. *President*, L. S. Copelin; *Secretary*, L. J. Boucher.

The Pick and Shovel Club of the Case School of Applied Science, Cleveland, Ohio. *President*, M. T. Whelan; *Secretary*, Lloyd A. Collier.

Colorado School of Mines Scientific Society, Golden, Colo. *President*, Alan Kissock; *Secretary*, George Wilfley.

Mining Engineering Society of the University of Arizona, Tucson, Ariz. *President*, J. Gary Lindley; *Secretary*, H. O. Coles.

Kentucky Mining Society, College of Mines and Metallurgy, University of Kentucky, Lexington, Ky. *President*, Oliver W. Smith; *Secretary*, Charles E. Straub.

Mining Society of Pennsylvania State College, State College, Pa. *President*, Ralph E. Kirk; *Secretary*, Willard M. Lindsey.

## MEMBERSHIP.

Due to the advanced date of publication of this *Bulletin*, the lists of new members, changes of address, etc., for the month of July will be printed in the September number of the *Bulletin*.

### CANDIDATES FOR MEMBERSHIP.

The following persons have been proposed during the period July 1–20, 1913, for election as members of the Institute. Their names are published for the information of members and associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Council, which has the power of final election.

*Members.*

**E. B. Braden**, San Francisco, Cal.
Proposed by Karl Eilers, H. Foster Bain, Thomas T. Read.
Born, May 12, 1864, Indianapolis, Ind. No technical education. 1884–90, Railroad business. 1890–94, Ore buying, sampling, and general mining. 1894–00, Assayer in charge U. S. Assay office, Helena, Mont. 1900–05, Mgr. American Smelting & Refining Co., plants, East Helena, Mont., and Everett, Wash.
Present position: Vice President, Selby Smelting & Lead Co.

**Waldo H. Comins**, St. Francois, Mo.
Proposed by Philip N. Moore, Arthur Thacher, O. M. Bilharz.
Born, 1880, Concord, N. H. 1902, Mass. Institute of Technology.
Present position: Supt. St. Louis Smelting & Refining Co., St. Francois, Mo.

**James O. Elton**, Anaconda, Mont.
Proposed by Richard S. McCaffery, E. P. Mathewson, Frederick Laist.
Born, 1879, Goldendale, Wash. Attended Washington State College. 1899–02, Worked as miner and assayer in Cœur d'Alênes Boundary District and Rossland, B. C., Eastern and Central Idaho. Entered University of Idaho. 1908, was graduated. 1909, Mining Engineer. 1909–11, Testing Department Washoe Smelter, Anaconda. 1911–12, Supt. Success Mine, Wallace, Idaho. 1912, Metallurgical Engineer with Anaconda Smelter Commission.
Present position: Metallurgical Engineer, Anaconda, Mont.

**Eugene N. Engelhardt**, Oakland, Cal.
Proposed by Thomas T. Read, Karl Eilers, Alfred von der Ropp.
Born, 1855, Russia. 1864–74, Military School in Russia. 1882–85, School of Mines, Columbia College. Graduated with degree of Engineer of Mines. 1886, Assayer, Chemist and Asst. Supt. Pueblo Smelting & Refining Co. 1887–88, Supt. of same company. 1889, Asst. Supt. Anaconda Smelting Co. 1890–07, Asst. Supt. Selby Smelting & Lead Co.
Present position: 1907 to date, Supt. of the above mentioned company.

**David J. Evans**, Humboldt, Ariz.
Proposed by Arthur H. Wethey, G. M. Colvocoresses, R. T. Walker.
Born, 1875, Swansea, Wales. 1891, A. A. University of Oxford, England. 1892–96, Technical education three private courses Engineering. 1900–02, Special Engineering Course, University of Wisconsin. 1892–96, Machinist Apprentice, Rock Springs, Wyo. 1896–00, Engineer. 1900–02, Special Engineering Course University of Wisconsin. 1902–09, Mechanical Engineer Course Parrot M. & S. Co., Butte, Mont., A. H. Wethey, Mgr. 1909–10, Asst. to Master Mechanic, United Verde Copper Co., Jerome, Ariz. 1910–11, Master Mechanic, Needles M. & S. Co., Needles, Cal., D. R. Muir, Mgr.
Present position: 1911 to date, Asst. Supt. Consolidated Arizona Smelting Co., Humboldt, Ariz.

**Charles Mason Farnham**, Barre Plains, Mass.
Proposed by Henry S. Drinker, Benj. L. Miller, Howard Eckfeldt.
Born, 1885, Lawrence, Mass. 1905, Diploma in Geology from Teachers' School of Science of Boston. 1908, S. B. in Geology from Harvard University. 1907–10, Cleveland Cliffs Iron Co., Ishpeming, Mich. 1910–11, Mangus Dev. Co. 1911–13, Esperanza Mining Co., Ellho Mining & Ry. Co. 1913, Instructor in Geology, Lehigh University.
Present position: Mining Geologist with Cerro de Pasco Mining Co., Peru.

**Lyman P. Hammond**, Denver, Colo.
Proposed by J. W. Finch, D. W. Brunton, Thos. B. Stearns.
Born, 1880, Bridgeport, Conn. Bridgeport Grade and High School. 1898–99, American Graphophone Co., of Bridgeport, Conn., as clerk in factory accounting department. 1899–00, Liberty Cycle Co., Bridgeport, as time keeper and clerk. 1900–01, Chief Clerk Factory Accounting Dept. of the Crocker, Wheeler Co., Ampere, N. J. 1901–03, Salesman, Crocker, Wheeler Co., N. Y. and Conn. 1903–05, Mgr. Denver Branch, Crocker, Wheeler Co. 1905–10, Sec'y and Sales Mgr. Denver Engineering Co. 1910–12, Mgr. Power Sales, Central Colo. Power Co. and Leadville Light & Power Co.
Present position: Vice President and Gen'l Mgr. Colo. Power Co., successors to Central Colo. Power Co., The Leadville Light & Power Co., and the Salidee Light, Power & Utility Co.

**Robert Mandeville Henderson**, Breckenridge, Colo.
Proposed by F. H. Bostwick, R. J. Grant, D. W. Brunton.
Present position: Gen'l Mgr. The Wellington Mines Co., Breckenridge, Colo.

**Edwin Wesley Hess**, Clearfield, Penna.
Proposed by George L. Miller, George F. Dunkle, L. C. Morganroth.
Born, 1868, Fishing Creek, Pa. 1874–85, Common Schools of Columbia Co., Pa. 1885–87, New Columbia Academy, Luzerne Co., Pa. No college course. 1887–00, Railroad mining and water works engineering, from chainman to charge of works. Louisville & Nashville Railroad Co, Norfolk & Western Railroad Co., Penna. Lines West of Pittsburgh. Choctaw, Oklahoma & Gulf Railroad Co., New York, Wyoming & Western R. R. Co., New York Central & Hudson River Railroad Co. 1900–13, General engineering practice, largely mining work, consulting engineer, designing and directing construction of mining operations, with railroad, municipal, and other engineering work. Office at Clearfield, Pa. Engineer for Clearfield Colliery Co., Good Clay & Coal Co., John M. Chase Estate, Clearfield Creek Coal Co., E. W. Hess & A. L. Bouton, DuBois, Pa., Fairmount Coal Co., Panther Run Coal Co.
Present position: Consulting Engineer.

**G. T. Jackson**, Juneau, Alaska.
Proposed by B. L. Thane, A. K. McDaniel, J. R. Whipple.
Born, 1873, England. 1897–01, Thames School of Mines, New Zealand. 1901–05, Waihi School of Mines, New Zealand. 1903, First Class Mine Manager's Certificate, New Zealand. 1897–99, New Alburnia Gold Mining Co., Thames, New Zealand. 1899–10, May Queen Hanraki, Thames, New Zealand. 1910–05, Waihi Gold Mining Co., Waihi, New Zealand. 1905–08, Nile Valley Gold Mining Co., Egypt. 1908–12, Alaska Perseverance Mining Co., Juneau, Alaska.
Present position: 1912 to date, Alaska Gastineau Mining Co., Juneau, Alaska. Supt.

**William W. Jones**, Albany, N. Y.
Proposed by D. H. Newland, Albert H. Fay, N. H. Darton.
Born, Dec., 1855, Festiniog, Wales, G. B. 1870–74, Teacher's Course. 1876–93, Votty & Bowydd Slate Mines, Mine Accountant, Festiniog, Wales, and charge of advance working measurements. 1893–98, Gen'l Mgr. Diphwys Casson Slate Mines, Festiniog, Wales, Mining, Surveying, and Commercial. 1898–03, Gen'l Mgr. Hower Slate Co., London, Danielsville, Pa. 1904–07, Slate Quarry operator, Granville, N. Y.
Present position: 1911 to date, Inspector of Mines and Quarries, New York State.

**Edward W. Keith**, Denver, Colo.
Proposed by D. W. Brunton, J. W. Finch, Howland Bancroft.
Born, 1869, Grosse Ile, Wayne Co., Mich. 1885, 86, 87, University of Michigan. 1888, Iron Silver Mining Co., Leadville. 1889–94, Practiced Colo. and Utah. 1895, Asst. Mgr. Arkansas Valley Smelting Co., Leadville. 1896, Practiced and operated Cripple Creek. 1897–02, Asst. Mgr. and Mgr. Arkansas Valley Smelting Co., Leadville. 1902–04, Cherokee Laryon Zinc Co., Gas, Kansas.
Present position: 1904 to date, Empire Zinc Co.

**Carl F. Moore**, Salt Lake City, Utah.
Proposed by W. G. Sharp, W. A. Wilson, George W. Riter.
Born, 1872, Williamsport, Pa. 1894, Graduated E. M., Mich. College of Mines. 1894–98, Instructor in Mechanical Engineering at the above college. 1898–99, Mechanical Engineer and Asst. Supt. Arcadian Copper Co. 1899–02, District Supt. of Construction of Wisconsin Bridge and Iron Co., stationed at Houghton, Mich. 1902–03, Engineer, Oliver Iron Mining Co., Duluth, Minn. 1903–05, Engineer and Asst. Supt. Old Dominion Consolidated Copper Mining Co., Globe, Ariz.
Present position: 1905 to date, Chief Engineer and recently consulting engineer, U. S. Smelting, Refining and Mining Co.

**Russell B. Paul**, Denver, Colo.
Proposed by D. W. Brunton, Howland Bancroft, J. W. Finch.
Born, 1879, Russell Gulch, Gilpin Co., Colo. Grad. Colorado School of Mines, 1902. E. M. Degree. 1903, Assayer, Bassick Mine, Quereda, Custer Co., Colo. 1904–05, Mill Supt. with the N. Y. Honduras & Rosario Mining Co., San Juanuita, Honduras, C. A. 1906-07, Mine Supt. Douglas Mining Co., Mina, Nev.
Present position: 1908 to date, Mining Engineer with Empire Zinc Co., Denver, Colo.

**Rudolf W. Rusterholz**, Butte, Mont.
Proposed by Geo. E. Moulthrop, C. W. Goodale, Reno H. Sales.
Born, 1876, Peoria, Ill. 1906, Degree of B. S. in Mechanical Engineering at Purdue University, Lafayette, Ind. Thesis will be submitted for advanced degree at Purdue the coming school year. Member of the American Society of Mechanical Engineers also of the Western Society of Engineers, Chicago. Have assisted in the design of various improvements in rock drills and other mining machinery. 1898–99, Member of 3d U. S. Engineer Corps from date of mustering in the regiment until it was mustered out of the service. 1907, in charge of irrigation work in Louisiana, 1906–13, have been employed by the Ingersoll-Rand Co., from the day I left college until the present day. 1908–13, Mechanical Engineer for Ingersoll-Rand Co.
Present position: Manager, Butte Branch, Ingersoll-Rand Co.

**C. F. W. Rys**, Pittsburg, Pa.
Proposed by Bradley Stoughton, H. M. Howe, Charles F. Rand.
Born, 1877, Essen, Germany. 1883–86, Public schools, Essen. 1886–92, High school, Essen. 1892–94, Krupp Works, Practical course. 1894–96, Royal Technical Schools, Hagen i. W. 1896-97, Melter, Basic and Acid O. H. furnaces, Krupp Works. 1898–02, School of Mines, Freiberg, Saxony, Class of 1902. Dipl. Eisenhütteningenieur. 1903–04, La Belle Iron Works, Engineering Department. 1904, Metallurgical Dept. Homestead Steel Works, Carnegie Steel Co. 1909, Asst. Metallurgical Engineer, Carnegie Steel Co.
Present position: 1911 to date, metallurgical engineer for same company.

**Hugh P. Tiemann**, Pittsburg, Pa.
Proposed by Bradley Stoughton, H. M. Howe, Charles F. Rand.
Born, 1879, New York City. See previous record furnished with application, 1901. Member of A. I. M. E., 1901–05. 1902–05, Asst. Metallurgist, Homestead Steel Works, Carnegie Steel Co. 1906–08, General offices Carnegie Steel Co., Pittsburg, Pa.
Present position: 1908 to date, Metallurgist Carnegie Steel Co.

**Herbert Lawrence Williams**, Denver, Colo.
Proposed by John V. N. Dorr, Allan J. Clark, G. M. Colvocoresses.
Born, 1884, Newton, Mass. 1898–02, Newton High School. 1902–06, Mass. Institute of Technology, S. B. 1906, Algoma Steel Co., Canada. 1907, Bumer Mining Co., Beatty, Nev., Tecopa Cons. Mining Co., Cal. 1908, Homestake Mining Co., S. D. 1909, Mogul Mining Co., S. D. 1910-11, Tigre Mining Co., El Tigre, Sonora, Asst. Consulting Engineer.
Present position: Stearns-Roger Mfg. Co., Sales Engineer.

*Associates.*

**Judd Stewart**, New York, N. Y.
Proposed by A. Eilers, Willard S. Morse, Karl Eilers.
Born, 1867, near Lawrence, Kansas. 1883, after preparing for High School education has been through experience and study, and by association with technical men. No degree other than those awarded by experience and somewhat successful career. Jan., 1883, began work with the Kansas City Smelting and Refining Co., afterwards the Consolidated Kansas City Smelting and Refining Co., taken over by American Smelting and Refining Co. Have been an executive officer for past 20 years and 30 years in Smelting and Refining Business.
Present position: Asst. to the President and General Auditor of American Smelting and Refining Co.

**Burton Y. Van Dyke**, Santiago de Cuba.
Proposed by George W. Pfeiffer, E. H. Emerson, Charles F. Rand.
Born, 1871, Weatherly, Pa. 1877–87 Weatherly, Pa., Public School. 1887–92, Appenticeship, Lehigh Valley R. R. shops, Weatherly, Pa. 1892, Warden Mfg. Co., Phila., Pa. 1893, Central R. R. of N. J., Ashley, Pa. 1893, Lehigh Valley R. R., Buffalo, N. Y. 1894, Northern Pacific R. R., Fargo, N. D. 1895–97, Lehigh Val-

ley R. R., Sayre, Pa. 1897, Bethlehem Steel Co., Bethlehem, Pa. 1898, Florida East Coast Hotel Co., Miami, Fla. 1899, same company at Ormond, Fla. 1900, Lehigh Valley R. R., Weatherly, Pa.

Present position: 1900 to date, Supt. of Machinery, Spanish American Iron Co.

*Junior Members.*

**Durward Bernhard**, Tucson, Ariz.

Proposed by A. L. Waters, Maxwell C. Milton, R. E. Fishburn.

Born, 1889, Barnesville, Ohio. Sept., 1910–June, 1911, Trinity College, Durham, N.C. June–July, 11, Muskingum College, New Concord. Sept. 1911–June, 1913, University of Arizona. June–Aug., 1912, Underground work at Lowell shaft of Copper Queen Mining Co., Bisbee, Ariz.

Instructors' signatures: Charles F. Willis, Dept. Mining and Geology. R. R. Goodrich, Prof. of Metallurgy, University of Arizona.

Present position: Student.

**Herbert George Officer**, New York, N. Y.

Proposed by Robert Peele, Arthur L. Walker, Charles P. Berkey.

Born, 1889, Ludington, Mich. 1907–11, University of S. C. Degrees B. S. and A. M. 1911–13, Columbia University, Engineer of Mines. 1905–07, Bookkeeper and general office man, Eddy Lake Cypress Co., Eddy Lake, S. C. Reference Wm. M. Burgan, 1503 Continental Building, Baltimore, Md.

Present position: Student, Columbia School of Mines.

**William A. Wong**, Honolulu, Hawaii.

Proposed by F. W. Traphagen.

Born, 1891, Honolulu, Hawaii. 1906, Graduated from McKinley High School, Honolulu, Hawaii. 1913, Graduated from the Colorado School of Mines, Golden, Colo.

## STANDING COMMITTEES.

*Executive.*

CHARLES F. RAND, *Chairman.*

JAMES F. KEMP,
ALBERT R. LEDOUX,
JOSEPH W. RICHARDS,
BENJAMIN B. THAYER.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Butte meeting, Aug. 18 to 21, 1913, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1913, when Vol. XLIV. of the *Transactions* will go to press. Any discussion offered thereafter should preferably be in the form of a new paper for publication in Vol. XLVII. (with suitable cross references in both volumes).

# Concentration of Slimes at Anaconda, Mont.

BY RALPH HAYDEN, ANACONDA, MONT.

(Butte Meeting, August, 1913.)

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## I. INTRODUCTION.

IN the wet concentration of ores there is always a certain amount of very fine material with which to contend, and which invariably carries such quantities of value that the treatment of it is worth while, commercially. This fine material usually contains that portion of the ore which hinders the separation of the mineral values: namely, the colloidal material resulting from the decomposition of certain constituents of the ore.

The object of this paper is to show what efforts have been made to solve the slime problem as it occurs in the concentrator of the Washoe

Reduction Works at Anaconda, Mont., where the copper sulphide ores from Butte are treated.

Before entering upon the discussion of the problem, it is necessary to define the material under consideration, in order that the problem may be apparent, and that due consideration may be given to the effort made in trying to solve it.

This fine material is termed slime; but as slime is a widely used and much misused term, further definition is necessary. Local conditions in individual mills determine the constitution of slimes. The material which is called slime in one mill may not be termed such in another mill. Different treatments of the same ore may produce slimes with different characteristics. This latter point is shown by the fact that the concentrator at Great Falls, treating the same general class of ores as is treated at Anaconda, produces a slime containing more copper and a larger proportion of granular material. As a result of this difference in granular content, any settling tank will handle about twice as much Great Falls slime as Anaconda slime, when operated to the same efficiency. Therefore, when conclusions from experiments conducted at Great Falls were applied to Anaconda slime they did not hold good in all cases.

## II. Definition of Anaconda Slime.

Anaconda slime may be defined as that portion of the Butte sulphide copper ores, concentrated by the Anaconda practice, which results from the breaking down of the ore in the mine and the crushing and regrinding of the ore in the mill; and which overflows the settling tanks receiving the overflow of the hydraulic classifiers, plus that which is contained in the head waters of the tables treating the spigot product of these tanks.

The constitution of the slime under consideration is determined largely by the settling capacity of the tanks and by the amount of table head water cut from the table tailings. But as the operating conditions are uniform, there is little variation in the slime produced.

## III. The Source of Slime.

The two flow sheets, Figs. 1 and 2, show the source of slime in Section No. 1 and Sections Nos. 2 to 8, respectively.

Section No. 1 was remodeled during the summer of 1912, according to data obtained from experiments conducted at Great Falls. The classifier installed consists of a single pocket and has center feed and peripheral overflow. It was developed at Great Falls and is known

as the Anaconda single-pocket classifier.[1] Two sets of these are used to remove the slime from the ore before further classification. Six No. 1 classifiers remove the original slime and also what is made in the crushers and in the coarse, fine, and intermediate rolls. Seven No. 3 classifiers remove the slime produced by the middling rolls and the Huntington mills. The overflows of these two sets of classi-

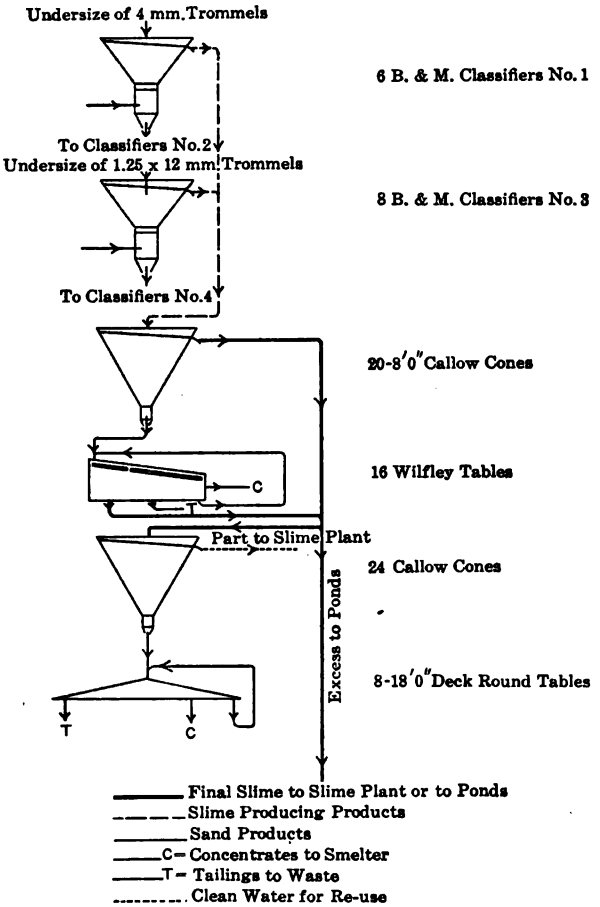



FIG. 1.—SOURCE OF SLIME IN SECTION NO. 1.

fiers are combined and go to 20 Callow tanks, where the coarser sand is removed through the spigot and treated on 16 Wilfley tables. This spigot product carries down some of the colloidal material, which is removed as head water of the tables, and which joins the overflow of the 20 Callow tanks, the combination constituting the slime from Section No. 1.

[1] See Fig. 2 of paper by A. E. Wiggin, The Great Falls System of Concentration, in this *Bulletin*.

The two sets of classifiers, No. 1 and No. 3, use sufficient hydraulic water to just give a clear water spigot, so that no colloidal material is treated on any machine in the section proper.

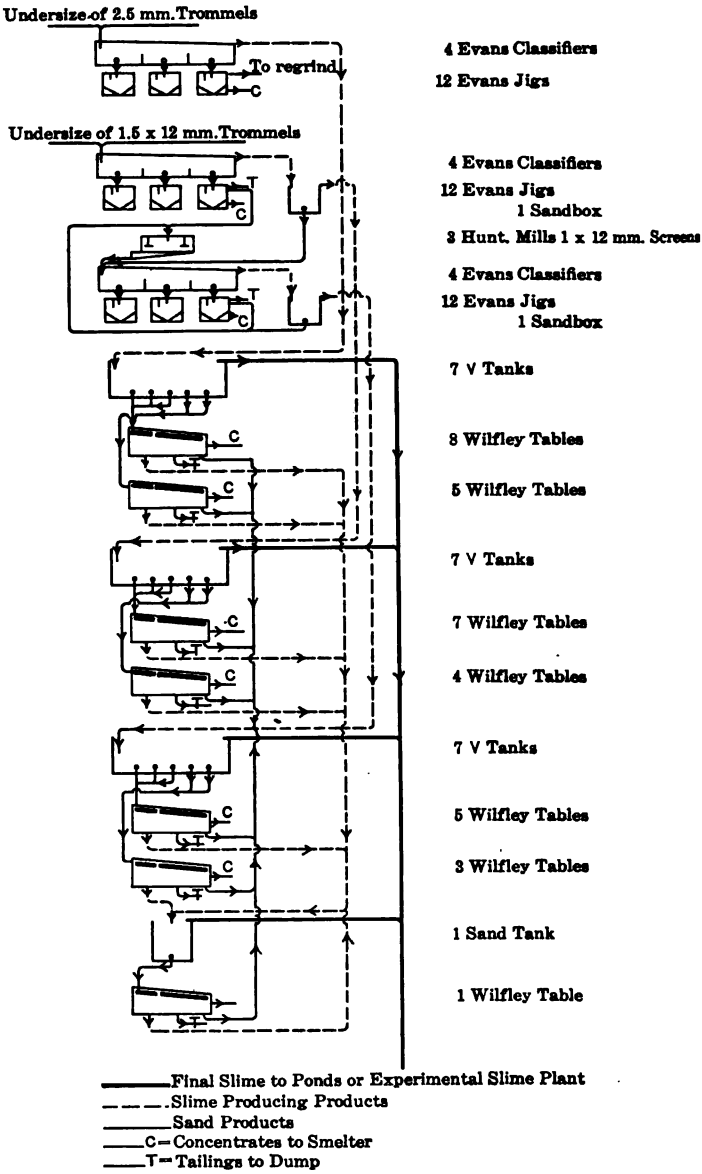



FIG. 2.—SOURCE OF SLIME IN SECTIONS NOS. 2 TO 8.

Sections Nos. 2 to 8 have the original flow sheet. The Evans launder type, three-spigot classifier is used. There are three sets of four each, and the spigot products are treated on Evans jigs. The

first set, or fine classifiers, receives the undersize of the 2.5-mm. trommels and overflows material which contains the original slime and that produced by the crushers and coarse and fine rolls. The second set, or middling classifiers, receives the undersize of the 1.5-mm. trommels and overflows material which contains the slime produced in the middling-rolls. The third set, or Huntington mill classifiers, receives the product of the Huntington mills and overflows material which contains the slime produced in the mills. The last two overflows are desanded in small tanks, the overflows of which, as well as the overflow of the fine classifiers, each go to seven V settling tanks, each 5 by 15 ft., fed at one end and overflowing at the other. The two sets of spigots, coarse and fine, from the tanks are treated on 32 Wilfley tables, the head waters of which are combined and desanded in a tank about 9 ft. square, having center feed and peripheral overflow. The plug of this tank is treated on one Wilfley table, the head water of which joins that of the other tables. The overflow of this square tank plus the overflow of the three sets of V tanks (21 in all) constitutes the slime from Sections Nos. 2 to 8.

Thus the sources of slime in the two flow sheets are similar, and with few exceptions, which will be noted, the slimes are similar.

## IV. Constitution of Slimes.

*Chemical.*—Chemical analyses of slime samples do not vary much. The analyses given in Table I. are of samples covering periods of from several days to a month, and extend over six years.

Table I.—*Chemical Analyses of Anaconda Slimes.*

| | Cu. Per Cent. | $SiO_2$. Per Cent. | FeO. Per Cent. | S. Per Cent. | $Al_2O_3$. Per Cent. | CaO. Per Cent. | Ag. Oz. Per Ton. | Au. Oz. Per Ton. |
|---|---|---|---|---|---|---|---|---|
| June, 1907..... | 1.83 | 58.9 | 3.9 | ..... | 19.8 | 0.4 | 1.6 | 0.004 |
| April, 1908..... | 2.18 | 61.5 | 3.7 | 3.4 | 18.0 | 0.3 | 1.6 | 0.004 |
| April, 1909..... | 2.08 | 60.1 | 3.5 | 3.7 | 21.9 | 0.2 | 1.5 | 0.005 |
| April, 1912..... | 2.04 | 58.8 | 4.5 | 4.4 | 19.4 | ..... | ..... | ..... |
| August, 1912..... | 2.09 | 62.2 | 3.7 | 4.2 | 18.6 | 0.8 | ..... | ..... |
| September, 1912 | 2.14 | 60.8 | 4.1 | 4.4 | 18.9 | 0.6 | 2.0 | 0.005 |

Attention is called to the percentage of $Al_2O_3$, which is from 18 to 22 per cent., while the $Al_2O_3$ in the original ore is only 9 per cent.

*Physical.*—Physical analyses of the slime show that it consists of granular sand, mineral, and middling grains, and colloid, the whole being in mechanical suspension in water. The granular materials are fine undecomposed ore constituents. The colloids are hydrated amorphous substances resulting from the decomposition of some of

the ore constituents, as kaolins, feldspars, etc., and some sulphides. Thus not only is part of the gangue in the colloidal form, but there is some of the copper in the same form, and wet concentration will not recover it. Slime as produced contains approximately 2 per cent. by weight of solids and 98 per cent. by weight of water.

Physical analyses of the slime were made in the following manner:

A sample of slime was procured from the main slime flume, and the colloid separated by repeated dilution and settling out of the sand, followed by a final settling of the colloid, or hydrated amorphous material, by the addition of clear lime water.

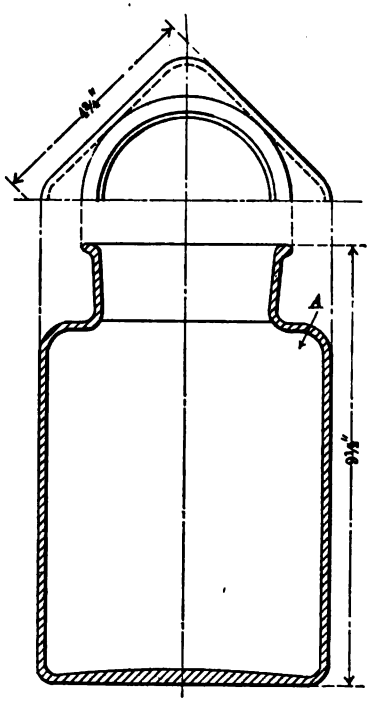



FIG. 3.—GLASS JAR FOR SLIME-SETTLING EXPERIMENTS.

The settling was done in glass candy jars, Fig. 3, which hold about 2.33 liters when filled to the lower end of the neck. These jars are found to be very convenient for such work, as the wide mouth permits rapid pouring, and the shoulder *A* serves as a trap to prevent the unintentional decantation of fine sand which has already settled.

Thirty-five liters of slime water were divided among a number of these jars, and after standing 15 sec. most of the water was poured into other jars and allowed to stand another 15 sec.; this operation being repeated till but very little additional sand settled out. The

residues, containing the coarser sand, were all combined and re-treated in the same way to obtain a sand product nearly free from colloid. The decanted water was then diluted to about twice the original volume, which reduced the size of the individual masses of coagulated, amorphous colloid, and thus lessened their rate of settling, so that, on standing 30 sec., considerable sand settled out, and but very little colloid. This operation was also repeated several times, and the approximately clean sand combined with the coarser sand. The decanted water was again diluted and allowed to stand for still longer periods, this process of stage dilution being continued till the original 35 liters

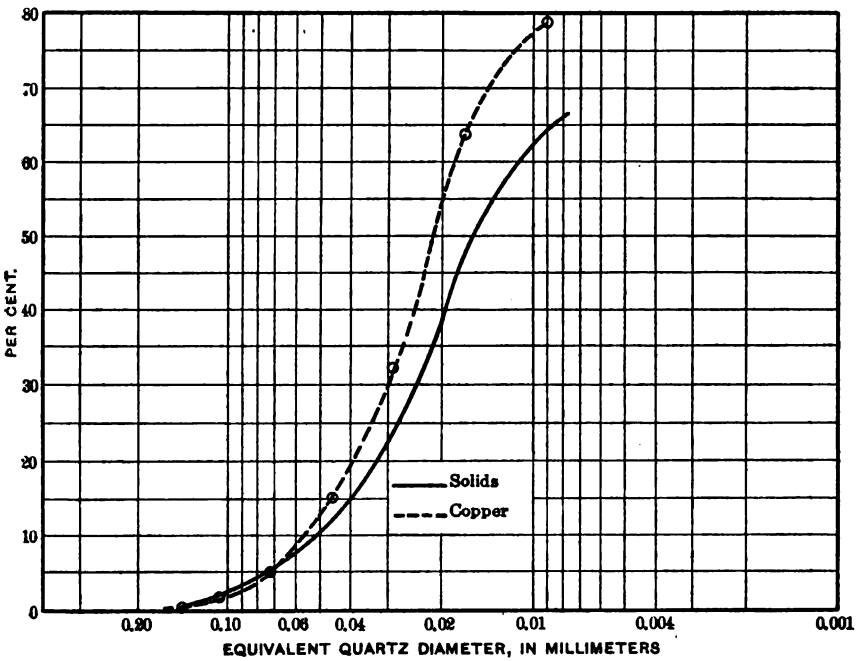



FIG. 4.—SETTLING TESTS OF GENERAL CONCENTRATOR SLIMES.

had become about 800 liters. The very finest crystalline material settled out of the highly diluted colloid in 45 min., but further standing of the decanted colloid for 1.5 hr. failed to settle any more sand. Considerable of this sand-free colloid remained in suspension for more than 24 hr., but was finally precipitated by the addition of a moderate amount of clear lime water.

The approximately clean sand was freed of the last of the colloid, by highly diluting with clear water, and allowing to settle an hour or more, decanting, and repeating the washing till no more colloid

remained, all these wash waters being added to the main bulk of sand-freed colloid.

The clean sand was classified by free settling in the same glass jars, first for 960 sec., pouring off the top 6 in. of water, filling the jar again, and repeating four times. The settled sand was similarly treated for successive periods of 480, 240, 120, 60, 30, and 15 sec. Each product was dried, weighed, and assayed. To determine the range of sizes of the sand products, microscopic measurements were

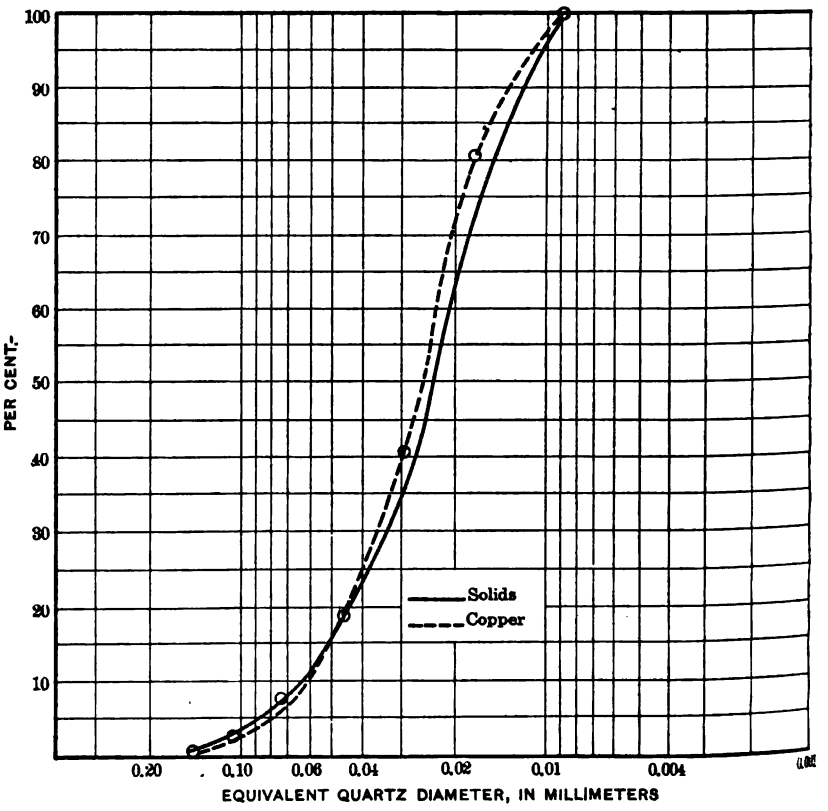



FIG. 5.—SETTLING TEST OF SAND PORTION OF SLIMES.

made on clean quartz obtained under the same settling conditions as those of the sands. From this data, curves, Figs. 4 and 5, were plotted, the former being the settlement curve for the general slime and the latter that for the sand portion only.

During the classification test a small amount of "Secondary colloid" was produced, owing to the chemical effect of water on the large area exposed by the finely divided sands.[2]

[2] Cushman: *Bulletin* Nos. 85 and 92, *Bureau of Chemistry, U. S. Department of Agriculture.*

Table II gives the results of such a physical analysis made on a sample of mill slime; and Table III gives chemical analysis of the colloidal portions.

From Fig. 5, 96 per cent. of the slime is finer than 200 mesh (0.08 mm.).

TABLE II.—*Settling Test on Anaconda Slime.*

| Velocity of Settling Per Second. | Diameter of Quartz of this Settling Velocity. | Solids. | | Copper. | | |
|---|---|---|---|---|---|---|
| | | Direct Per Cent. | Cumulative. Per Cent. | Assay. | Direct Per Cent. | Cumulative Per Cent. |
| Millimeters. | Millimeters. | | | | | |
| 10.16 + | 0.146 + | 0.6 | 0.6 | 1.40 | 0.4 | 0.4 |
| 10.16 — 5.08 | 0.146 — 0.106 | 1.2 | 1.8 | 1.96 | 1.1 | 1.5 |
| 5.08 — 2.54 | 0.106 — 0.074 | 3.1 | 4.9 | 2.54 | 3.5 | 5.0 |
| 2.54 — 1.27 | 0.074 — 0.046 | 7.3 | 12.2 | 3.05 | 10.0 | 15.0 |
| 1.27 — 0.635 | 0.046 — 0.029 | 11.1 | 23.3 | 3.46 | 17.2 | 32.2 |
| 0.635 — 0.318 | 0.029 — 0.017 | 24.0 | 47.3 | 2.94 | 31.4 | 63.6 |
| 0.318 — 0.159 | 0.017 — 0.009 | 17.1 | 64.4 | 1.96 | 15.0 | 78.6 |
| 0.159 — 0.000 | 0.009 — | 0.4 | 64.8 | 0.86 | 0.1 | 78.7 |
| Total sand..... | ..... | 64.8 | ..... | 2.72 | 78.7 | ..... |
| Primary colloid..... | ..... | 33.0 | 97.8 | 1.47 | 20.3 | 99.0 |
| Secondary colloid... | ..... | 2.2 | 100.0 | 1.13 | 1.0 | 100.0 |
| | | 100.0 | ..... | 2.27 | 100.0 | ..... |

TABLE III.—*Chemical Analysis of Colloids.*

| | Free water. | Ignition Loss. | $SiO_2$. | Fe. | S. | $Al_2O_3$. | CaO. | Cu. |
|---|---|---|---|---|---|---|---|---|
| | Per Cent. | Per Cent. | Per Cent. | Per Cent. | Per Cent. | Per Cent. | Per Cent. | Per Ct. |
| Primary colloid..... | 0.46 | 8.08 | 52.5 | 1.7 | 2.3 | 24.8 | 0.9 | 1.47 |
| Secondary colloid.... | 0.43 | 5.18 | 58.1 | 1.3 | 1.7 | 25.4 | 0.9 | 1.13 |

## V. THE SLIME PROBLEM.

The problem of wet concentration of the Anaconda slimes may be divided into two parts, viz.: First, to prepare the slime for use on some concentrating machine by thickening it to the proper density, and at the same time recover water clean enough for re-use in the mill; and second, to recover the mineral values from the thickened pulp in a concentrate which can be smelted.

The quantity of material to be handled calls for an extensive plant and consequently the most efficient and economical installation is sought. The quantity of slime produced is given in Table IV.

TABLE IV.—*Quantity of Slime Produced.*

| Ore Treated Per Section Per 24 Hr. | | Assay. | Slime Produced Per Section Per 24 hr. | | Assay. | Density. | Per Cent. of Original. | |
|---|---|---|---|---|---|---|---|---|
| | | | | | | | Ore. | Copper. |
| | Tons. | Per Cent. Cu. | Gallons. | Tons. | Per Cent. Cu. | Per Cent. | | |
| Section 1..... | 1,470 | 3.25 | 2,275,700 | 193.6 | 1.85 | 1.98 | 12.9 | 7.4 |
| Sections 2 to 8.. | 1,313 | 3 24 | 2,646,000 | 225.0 | 2.03 | 2.01 | 17.2 | 10.8 |

The present output of the mill, based on 10,000 tons of ore treated per 24 hr., is 18,401,000 gal. of pulp containing 1,660 tons of dry slime.

The first part of the problem, then, is to thicken 18,401,000 gal. of pulp at 2 per cent. density to a pulp of 10 per cent. density, necessitating the removal of 15,700,000 gal. of water. The second part is to obtain machines of suitable capacity, efficiency, and cost of maintenance to treat 1,660 tons of dry slime and produce a concentrate which can be smelted.

## VI. EXPERIMENTAL SLIME PLANT.

All of the slime produced previous to July, 1912, was sent to the slime ponds, located below the works, and whatever settled in them was excavated and stored in a pile adjacent to the ponds. A limited amount of the excavated material has been smelted in the blast furnaces. Since the summer of 1911 the Peck centrifugal concentrator has been treating a portion of the slime settled in one of the ponds.

Thus there had been little done to treat all of the slime direct from the mill until July, 1912, when Section No. 1 of the concentrator was remodeled. No decision had been reached as to the best method for treating the slime: but from data obtained from slime experiments conducted at Great Falls, sufficient Callow tanks and round tables were installed in an experimental slime plant to treat one-fourth of the slime produced in one section of the mill. This flow sheet is given in Fig. 1. Since the installation several thickening and concentrating devices have been added and competitive tests made upon them, to determine the machines best adapted to the solution of the slime problem. The various thickeners include the Kuchs-Laist centrifugal separator, the Callow tank, the Garred filter, and the Dorr continuous thickener. The concentrating machines include the round table and four standard makes of concentrators, the latest slime machines on the market. All these machines have received slime from Section No. 1, slime from Sections Nos. 2 to 8, and the total classifier overflow of Section No. 1.

## VII. Thickeners.

The simplest slime thickeners are the open rectangular tanks and the slime ponds, the latter being enlargements of the former. A modification of the open tank is the Callow tank, which has center feed and peripheral overflow. Mechanical thickeners include the Kuchs-Laist centrifugal separator, the Garred filter, and the Dorr continuous thickener. A brief description of the several devices and the results of tests made upon them are given below.

*Slime Ponds.*—There are six of these, each 300 ft. wide and 600 ft. long, and average 15.5 ft. deep. These are operated in cycles, and when possible three are used in series. As soon as a pond becomes filled, it is allowed to drain, and the resulting settlement, containing 35 per cent. of moisture, is excavated by a Lidgerwood bucket excavator. No very complete data of the operation of the ponds are available, but the degree of settlement has been governed by the extent to which excavation has been carried on and empty ponds provided. Whatever has been excavated has been stored for future treatment, by smelting or otherwise.

One of these ponds has been reserved to thicken slime for the Peck machine. It was hoped that the pond could be partly filled with slime and then, by pumping from different depths, a pulp obtained of the proper density for feed: viz., 20 per cent. of solids by weight. Several attempts were made to perfect this scheme, but with no success. The scheme finally adopted is as follows: Two large tanks, 16 by 16 ft. and 8 ft. deep, supplied with agitators, are located under a hopper-bottomed bin, into which is dumped slime dredged from the pond. A portion of this is charged into one of the agitation tanks and mixed with water pumped from near the surface of the pond, until a pulp of about 20 per cent. density is obtained. This is fed to the Peck machine. Meanwhile the other tank is preparing a charge, which is held in readiness until the first tank is emptied. In this way a uniform feed is obtained. This, however, is rather a crude way, particularly where large tonnages are to be handled.

If the ponds are fed with the proper quantity of slime complete settlement is obtained; but difficulty arises in recovering the slime at the proper density.

*Tanks.*—Open tanks include any tanks where no baffles are used, and free settling is used. They may be fed at one end and overflow at the other, as in rectangular tanks, or they may be fed at one or more interior points and overflow peripherally, as in square, round, or rectangular tanks. Baffle tanks have some arrangement of baffles (either vertical or sloping) and may be fed the same as the plain tanks.

A tank 10 ft. 10 in. by 9 ft. 6 in. and 8 ft. deep, fed at the center, having peripheral overflow, and having one spigot from the center of the bottom, gave the following results when fed with slime at 2.5 per cent. density. In this tank the slime forms its own hopper in the bottom of the tank. The spigot density was about 8 per cent.

| Gallons Per Minute. | | Per Cent. of Solids |
|---|---|---|
| Total. | Per Square Foot. | In Spigot. |
| 20 | 0.194 | 100.0 |
| 22 | 0.213 | 95.0 |
| 25 | 0.233 | 90.0 |
| 28 | 0.272 | 85.0 |
| 31 | 0.301 | 80.0 |
| 34 | 0.330 | 75.0 |
| 38 | 0.369 | 70.0 |

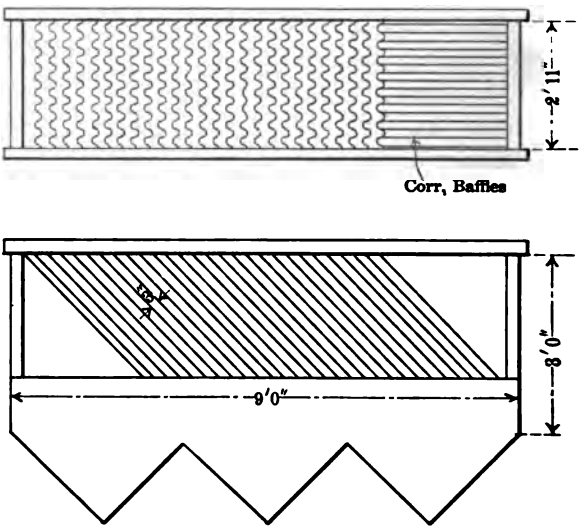



FIG. 6.—EXPERIMENTAL BAFFLE TANK.

*Comparative Efficiency of Open and Baffle Tanks:*—For this experiment, a tank 3 ft. wide, 9 ft. long, and 3 ft. deep, with three hoppers, was used, Fig. 6. Feed entered at one end over an apron, so as to produce as little disturbance as possible, and overflowed at the opposite end, over the entire width. Corrugated iron baffles sloping 45° were placed in different positions, including longitudinal and transverse, and at different depths below the surface. The results of the various tests are given in Table V. and plotted in Fig. 7, the latter also showing a composite curve, *A*, of all the baffle tests. Table VI. summarizes the tests under varying quantities of feed. From this table, if perfect settling is required a baffle tank has no advantage, but if it is desired to slough off a portion of the solids, the baffle tank has a greater capacity.

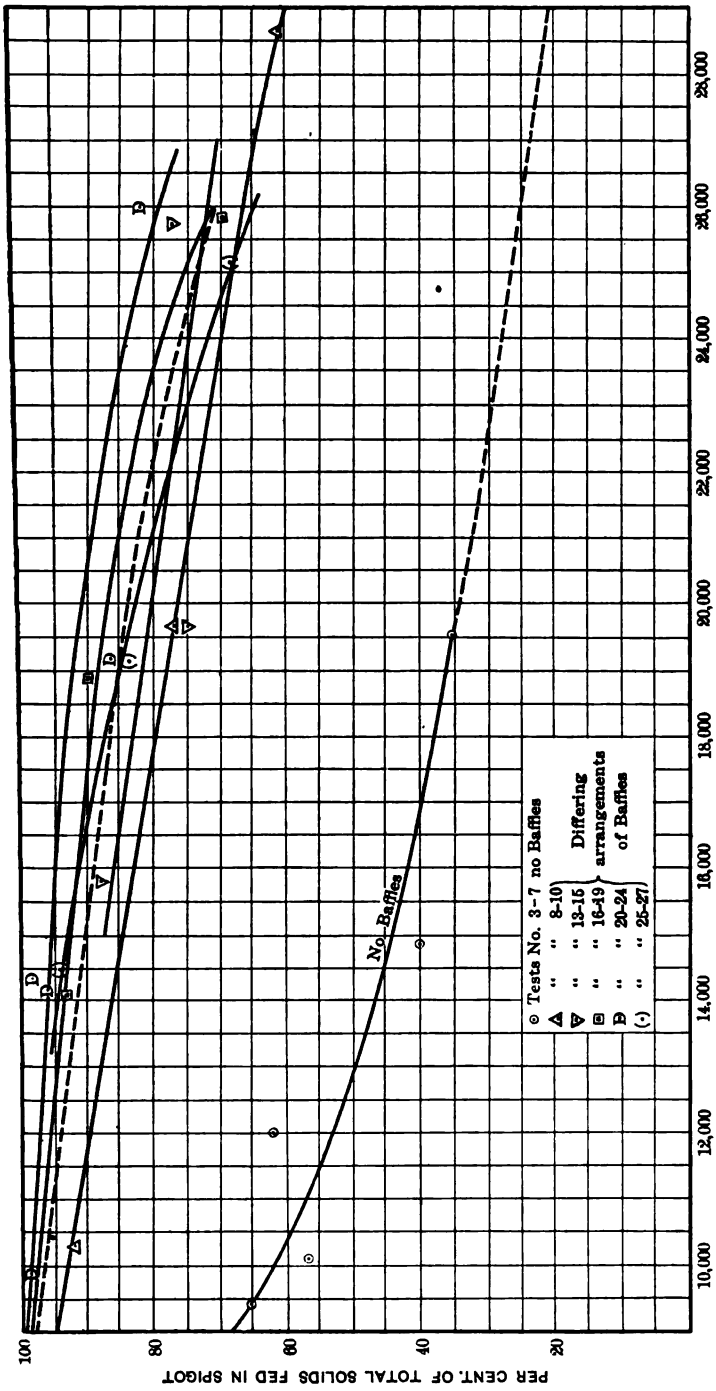



FIG. 7.—CURVE OF BAFFLE TANK DATA.

TABLE V.—*Slime Settling in Open and Baffle Tanks.*

| Test No. | Arrangement of Baffles. | Feed to Tank. | | Feed to Tank Per Foot of Width. | | Spigot Discharge from Tank. | | | |
|---|---|---|---|---|---|---|---|---|---|
| | | Gallons Per 24 Hr. | Pounds Solid Per 24 Hr. | Gallons Per 24 Hr. | Pounds Solid Per 24 Hr. | Per Cent. of Total Feed Vol'me. | Density Per Cent. of Solid. | Per Cent. of Total Solids Fed. | Per Cent. of Total Water Fed. |
| 3 | No baffles | 9,360 | 1,300 | 3,120 | 433 | 9.4 | 10.9 | 65.9 | 68.1 |
| 4 | Ditto | 10,075 | 1,545 | 3,358 | 515 | 9.4 | 10.4 | 57.4 | 59.5 |
| 5 | Ditto | 12,010 | 1,655 | 4,003 | 552 | 7.6 | 12.6 | 63.1 | 66. |
| 6 | Ditto | 14,900 | 2,290 | 4,966 | 763 | 5.4 | 12.2 | 39.7 | 41. |
| 7 | Ditto | 19,500 | 3,240 | 6,500 | 1,080 | 4.6 | 13.8 | 35.3 | 37.5 |
| 8 | Longitudinal baffles 4 in. deep[a] | 10,250 | 1,540 | 3,416 | 513 | 12.8 | 11.9 | 92.3 | 95.1 |
| 9 | Ditto | 19,715 | 2,960 | 6,572 | 987 | 8.0 | 15.7 | 76.9 | 79. |
| 10 | Ditto | 28,630 | 4,300 | 9,543 | 1,433 | 4.6 | 20.5 | 61.1 | 63. |
| 13 | Longitudinal baffles 0.5 in. deep | 15,905 | 2,390 | 5,302 | 797 | 8.3 | 17.0 | 87.6 | 90.7 |
| 14 | Ditto | 19,704 | 2,965 | 6,568 | 988 | 4.7 | 24.3 | 74.9 | 77. |
| 15 | Ditto | 25,885 | 3,885 | 8,628 | 1,295 | 4.5 | 25.4 | 76.6 | 79.4 |
| 16 | Crosswise baffles 4.5 in. deep[a] with tops sloping away from feed end of tank | 9,955 | 1,495 | 3,318 | 498 | 11.2 | 14.2 | 97.4 | 100. |
| 17 | | 14,125 | 2,120 | 4,708 | 706 | 7.9 | 18.7 | 93.6 | 96. |
| 18 | | 18,965 | 2,845 | 6,322 | 948 | 7.4 | 18.8 | 89.4 | 91. |
| 19 | | 25,855 | 3,880 | 8,618 | 1,293 | 4.2 | 25.5 | 70.6 | 73. |
| 20 | Ditto, 0 5 in.[a] | 9,735 | 1,460 | 3,245 | 487 | 9.8 | 16.2 | 98.5 | 98. |
| 21 | Ditto | 14,360 | 2,155 | 4,787 | 718 | 8.7 | 17.7 | 98.3 | 99. |
| 22 | Ditto | 14,255 | 2,140 | 4,752 | 713 | 7.5 | 20.0 | 96.0 | 99. |
| 23 | Ditto | 19,070 | 2,865 | 6,357 | 953 | 5.4 | 24.2 | 86.8 | 90.9 |
| 24 | Ditto | 25,965 | 3,900 | 8,655 | 1,300 | 4.5 | 26.9 | 82.3 | 86.9 |
| 25 | Crosswise baffles 0.5 in. deep[a] with tops sloping toward feed end of tank | 14,355 | 2,155 | 4,785 | 718 | 4.8 | 28.6 | 93.4 | 94. |
| 26 | | 19,040 | 2,860 | 6,347 | 953 | 6.0 | 21.7 | 84.8 | 87. |
| 27 | | 25,010 | 3,755 | 8,337 | 1,252 | 3.6 | 27.7 | 68.3 | 72. |
| 28 | Ditto, but 5 in. deep at feed end and 4 in. at overflow end of tank | 19,065 | 2,865 | 6,358 | 955 | 5.1 | 22.1 | 76.7 | 80. |
| 29 | | 25,475 | 3,825 | 8,492 | 1,275 | 4.1 | 23.4 | 62.9 | 66. |

[a] The depth here, means the depth of the top edges of the baffles below the overflow of the tank.

TABLE VI.—*Summary of Open and Baffle Tank Data.*

| Gallons of Feed Per Foot of Width Per 24 hr. | Per Cent. Settled. | | Rates of Settling Efficiency. |
|---|---|---|---|
| | Baffle Tank. | Open Tank. | |
| | Per Cent. | Per Cent. | |
| 1,883[a] | 100.0 | 100.0 | 1.00 |
| 2,000[a] | 99.9 | 94.8 | 1.05 |
| 3,000 | 97.5 | 67.8 | 1.44 |
| 4,000 | 94.0 | 53.5 | 1.76 |
| 5,000 | 90.3 | 44.8 | 2.02 |
| 6,000 | 86.8 | 38.0 | 2.28 |
| 7,000 | 82.0 | 32.6 | 2.52 |
| 8,000 | 75.9 | 28.0 | 2.71 |
| 9,000 | 67.8 | 23.8 | 2.85 |

[a] These figures estimated by extending the curves.

*Callow Tanks.*—The Callow tanks in the slime plants are 8 ft. in diameter, center feed, peripheral overflow, and discharge through goose necks from the bottom of the cone. Twenty of these receive the classifier overflow. They are each handling 118,700 gal. in 24 hr. (82.4 gal. per minute) of pulp at 2 per cent. density and discharging 40 to 45 per cent. of the solids in a pulp of 8 per cent. density. Twenty-four tanks are used to thicken whatever feed is to be treated

on the round tables or competing machines. At the Great Falls plant tanks corresponding to these are handling 25 gal. per minute and overflowing clear water, but owing to the difference in constitution of the slimes 9 to 11 gal. per minute is the maximum capacity when settling Anaconda slime to a clear water overflow. The operation of these tanks under different conditions of feed and efficiency is shown in Table VII.

TABLE VII.—*Operation of Callow Tanks Under Different Feed Conditions.*

| | Gallons Per Tank Per Minute. | Pounds of Solids Per Tank Per Minute. | Pulp. Density. | Pulp. Assay. Per Cent. Cu. | Per Cent. of Total. Solids. | Per Cent. of Total. Pulp. |
|---|---|---|---|---|---|---|
| *A.* Feed: Total classifier overflow from Section No. 1. | | | | | | |
| Feed..... | 11.1 | 1.93 | 2.1 | 2.22 | 100.0 | 100.0 |
| Spigot..... | 1.9 | 1.85 | 11.0 | 2.26 | 94.5 | 95.5 |
| Overflow.... | 9.2 | 0.10 | 0.1 | 1.56 | 5.5 | 4.5 |
| *B.* Feed: Slime from Section No. 1. | | | | | | |
| Feed..... | 12.1 | 1.73 | 2.0 | 1.83 | 100.0 | 100.0 |
| Spigot..... | 1.9 | 1.64 | 10.0 | 1.85 | 94.5 | 95.4 |
| Overflow... | 10.2 | 0.09 | 0.1 | 1.56 | 5.5 | 4.6 |
| *C.* Feed: Same as *B.* | | | | | | |
| Feed..... | 11.6 | 1.63 | 2.0 | 1.84 | 100.0 | 100.0 |
| Spigot..... | 1.9 | 1.48 | 9.0 | 1.88 | 91.1 | 93.0 |
| Overflow... | 9.7 | 0.15 | 0.2 | 1.43 | 8.9 | 7.0 |
| *D.* Feed: Slime from Sections No. 2 to 8. | | | | | | |
| Feed..... | 13.9 | 2.37 | 2.0 | 2.03 | 100.0 | 100.0 |
| Spigot..... | 1.6 | 2.11 | 14.5 | 2.07 | 89.2 | 91.2 |
| Overflow... | 12.3 | 0.26 | 0.3 | 1.66 | 10.8 | 8.8 |
| *E.* Feed: Same as *D.* | | | | | | |
| Feed..... | 10.9 | 1.91 | 2.0 | 1.93 | 100.0 | 100.0 |
| Spigot..... | 1.7 | 1.79 | 11.7 | 1.98 | 94.0 | 95.6 |
| Overflow... | 9.2 | 0.12 | 0.1 | 1.45 | 6.0 | 4.4 |

*Kuchs-Laist Centrifugal Separator.*—This machine is designed to remove the colloidal material and excess water from the slimes and produce a spigot discharge containing all the granular material and which is suitable for feed to some slime-concentrating machine. As shown in Fig. 8, it consists of a set of radial chambers mounted on a horizontal hollow shaft, about which it revolves. At the outer periphery of each chamber is a spigot discharge plug to remove the heavier material which settles in the chambers due to centrifugal force. Feed enters at one end of the hollow shaft, through a stuffing-box connection to the feed pipe, and passes to the chambers, which are provided with baffles. These baffles serve to check the

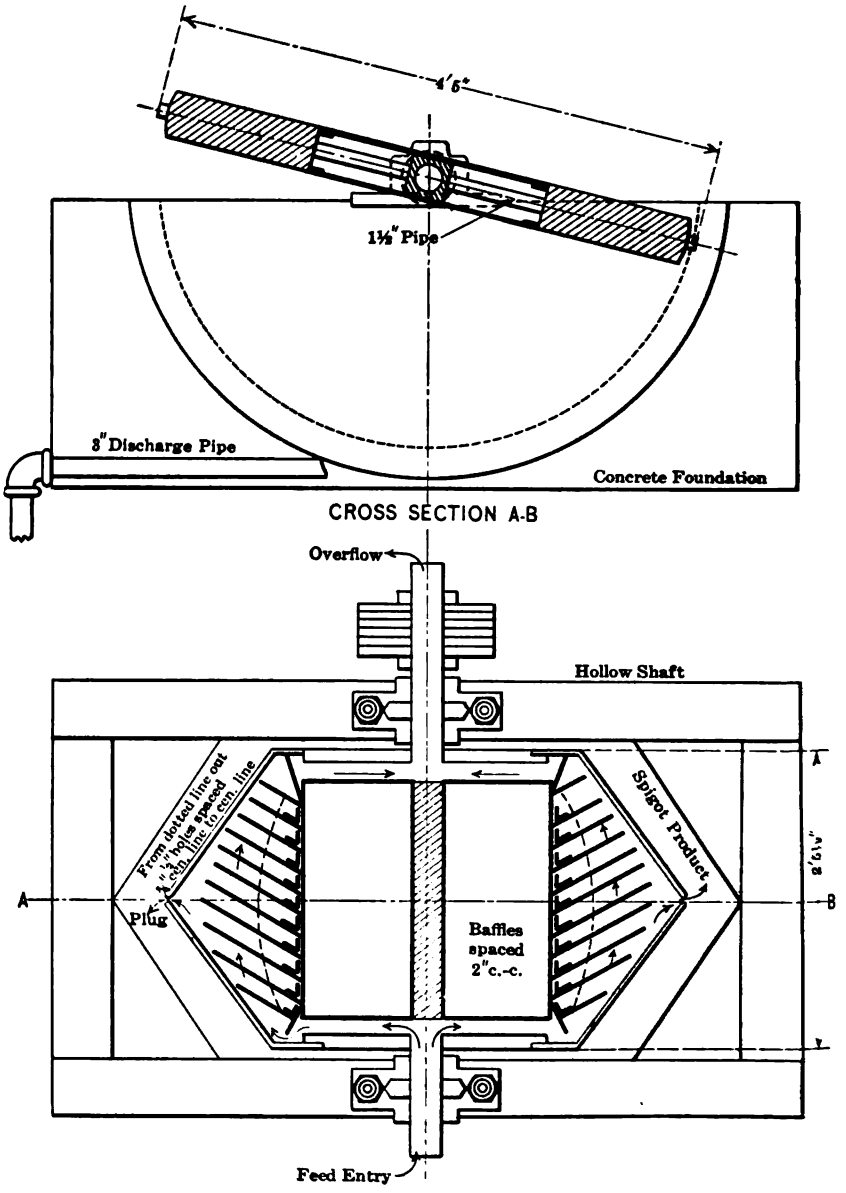



FIG. 8.—KUCHS-LAIST CENTRIFUGAL SEPARATOR.

flow of material and direct the heavier material to the point of discharge. The heavier material is discharged through the plugs, and the lighter is forced back to the hollow shaft and is discharged at the opposite end from which it enters. This particular machine has two chambers, but the circumference could be completed with chambers so that the machine would have about 12 times its present capacity. The speed has been varied from 640 to 1,200 rev. per min., and, roughly, the capacity varies directly with the speed, if the same density of spigot product be maintained.

Considerable difficulty was experienced in obtaining material of which to make discharge plugs which would wear a reasonable length of time. Plugs of cast iron, steel, topaz, agate, and rubber were used. The holes in these plugs varied from $\frac{1}{16}$ to $\frac{1}{8}$ in. when new and gave a spigot density of from 15 to 30 per cent. After running from 3 to 5 hr. these holes were enlarged to such an extent that the density dropped to from 8 to 15 per cent., and in the case of rubber plugs the holes became choked with pieces of rubber torn from the plug. Finally a set of plugs of sintered alumina (called "alundum") having a hole $\frac{1}{30}$ in. in diameter was tried. These plugs were in use for 85 hr. and the density dropped from 30.7 to 16.2. The high density at the start is due to the small opening.

Between the speed limits of 640 and 1,200 rev. per min. the speed at which the machine is run does not materially affect the life of the plug.

Table VIII. gives the data of two tests run at 1,200 and 640 rev. per min., respectively.

TABLE VIII.—*Operation of Kuchs-Laist Centrifugal Separator on Slime from Sections Nos. 2 to 8.*

| | Gallons. | | Solids. | | Copper. | | Density. |
|---|---|---|---|---|---|---|---|
| | Per Minute. | Per Cent. of Total. | Pounds Per Minute. | Per Cent. of Total. | Assay. | Per Cent. of Total. | |
| *A.* Speed of machine, 1,200 rev. per min. | | | | | | | |
| Spigot | 8.57 | 9.3 | 10.2 | 61.7 | 2.90 | 76.1 | 13.1 |
| Overflow | 83.76 | 90.7 | 6.3 | 38.3 | 1.47 | 23.9 | 0.9 |
| Total | 92.33 | 100.0 | 16.5 | 100.0 | 2.3[illegible] | 100.0 | 2.1 |
| *B.* Speed of machine, 640 rev. per min. | | | | | | | |
| Spigot | 5.43 | 9.8 | 5.0 | 55.1 | 2.36 | 67.9 | 10.3 |
| Overflow | 50.82 | 90.2 | 4.1 | 44.9 | 1.37 | 32.1 | 0.7 |
| Total | 56.25 | 100.0 | 9.1 | 100.0 | 1.92 | 100.0 | 2.0 |

To show the extent to which the colloid is separated, a colloid analysis, as previously described, was made on the product of a test similar to *B*, Table VIII., with the results shown in Table IX.

TABLE IX.—*Colloid Analysis of Products of Kuchs-Laist Separator.*

| | Spigot. | | | | | Overflow. | | | | | Total Products | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Solids. | | Copper. | | | Solids. | | Copper. | | | Solids | | Copper | |
| | Per Cent. of Total Spigot. | Per Cent. of Feed. | Assay. | Per Cent. of Total Spigot. | Per Cent. of Feed. | Per Cent. of Total Overflow. | Per Cent. of Feed. | Assay | Per Cent. of Total Overflow. | Per Cent of Feed. | Per Cent. of Total. | Assay | [illegible] | [illegible] |
| Sand.... | 85.0 | 56.1 | 2.96 | 91.6 | 76.8 | 7.2 | 2.5 | 1.96 | 14.1 | 2.3 | 58.6 | 2.92 | [illegible] | [illegible] |
| Colloid | 15.0 | 9.9 | 1.54 | 8.4 | 7.1 | 92.8 | 31.5 | 0.95 | 85.9 | 13.8 | 41.4 | 1.09 | [illegible] | [illegible] |
| Total ... | 100.0 | 66.0 | 2.75 | 100.0 | 83.9 | 100.0 | 34.0 | 1.02 | 100.0 | 16.1 | 100.0 | 2.16 | [illegible] | [illegible] |

*Garred Filter.*—An experimental Garred filter has recently been installed to prepare feed for the round tables, slime from Sections Nos. 2 to 8 being used for feed. The filter consists of 100 leaves, each 6 by 10 ft., made of 10-oz. canvas, and connected to a centrifugal pump. The 100 leaves are divided into five sections of 20 leaves each, and connected with water at 15 lb. pressure in such a way that the suction on the several sections can be cut off and the water turned into the leaves. After the filter has been in operation for a while, the suction is cut off from one section of leaves and the clean water turned on. This forces the cake, which is formed on the outside of the leaf, from the leaf, and it falls to the bottom of the tank containing the leaves. The several sections are similarly treated, with the result that a thickened pulp is discharged through plugs provided in the bottom of the tank.

The filter has the advantage of positive action, with 100 per cent. efficiency in settling, provided the leaves are kept in good condition. This filter treating slime at the rate of 400 gal. per minute produced a pulp of 11.9 per cent. density. The filter was run at this capacity, as it produced sufficient pulp to feed the eight round table decks used to treat the pulp. No definite data as to maximum capacity has been obtained, but in all probability the 100-leaf filter will handle 1,000 gal. of slime per minute as an average during the life of the filter leaves.

*Dorr Continuous Thickener.*—One Dorr thickener was installed near the slime plant, and fed with slime from Sections Nos. 2 to 8. As this was just recently installed, only a few tests have been made upon it, but the results are very favorable.

The tank is cylindrical, is 27.5 ft. in diameter and 9.7 ft. deep, and has center feed and peripheral overflow. The discharge is through one spigot at the center of the bottom. Near the bottom of the tank is a set of four rakes attached to a central vertical shaft which makes about 3.3 rev. per hour, and rakes the settled slime to the point of discharge. The rakes are set at 12° with the horizontal, so that a flat cone of the slime is formed. In later tests the rake arms were made horizontal. This machine has the advantage of, first, a large settling area, without the disadvantage of the conical tank with subsequent excessive loss of mill head; and, second, only one spigot of reasonable size, instead of numerous small spigots as in the case of small settling units. The capacity of the 27.5 ft. Dorr thickener per square foot of settling area is practically the same as that of the Callow tanks when operated to the same efficiency.

Experiments were made to determine the capacity and efficiency of different depths of tank. For this purpose the overflow launder was lowered to make the tank 3 ft., and then 2 ft. deep. The rakes were made horizontal. The data obtained are given in Table X., and indicate that when operating to about 98 per cent. efficiency a 3-ft. tank has 85 per cent. of the capacity of a 9.7-ft. tank; while a 2-ft. tank has about 85 per cent. of the capacity of a 3-ft. tank.

TABLE X.—*Treatment of Slime from Sections Nos. 2 to 8 in Dorr Continuous Thickener.*

| Depth of Tank | Feed. | | | Spigot Discharge. | | | | | Overflow. | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Rate Per 24 Hr. | | Density. PerCent. | Rate Per 24 Hr. | | Density. PerCent. | Per Cent. of Total. | | Rate Per 24 Hr. | | Per Cent. of Total. | |
| | Pulp. | Solids. | | Pulp. | Solids. | | Pulp. | Solids. | Pulp. | Solids. | Pulp. | Solids. |
| Feet. | Gal. | Lb. | Solids. | Gal. | Lb. | Solids. | | | Gal. | Lb. | | |
| 9.7 | 195,110 | 34,380 | 1.90 | 39,860 | 34,205 | 9.6 | 20.0 | 99.5 | 155,250 | 175 | 80.0 | 0.5 |
| 3.0 | 163,350 | 31,250 | 2.26 | 24,210 | 30,640 | 13.2 | 14.9 | 98.0 | 139,140 | 610 | 85.1 | 2.0 |
| 2.0 | 162,940 | 29,320 | 2.13 | 22,800 | 26,380 | 12.7 | 14.0 | 90.0 | 140,140 | 2,940 | 86.0 | 10.0 |

## VIII. CONCENTRATORS.

Two types of concentrating machines have been tested very thoroughly, both on short and long time tests. These are the round table and the Peck centrifugal concentrator. In addition to these, competitive tests were made on four standard machines, each furnished by a different manufacturer, and which are the latest types of slime machines on the market. As these machines were furnished by the makers and there were differences in the recoveries made by the several machines, and not wishing to publish data detrimental to the less efficient ones, when treating this slime, the results of these tests will be discussed only in a general way.

*Round Table.*—Probably one of the oldest types of concentrating machines, the round table, or "buddle," has been revived of late years, and with the improvements made in the construction, is now a serviceable and efficient machine. Those installed in the experimental slime plant and on which these tests were made have a steel frame, with concentrating surfaces of cement. They are 18 ft. in diameter, and are built in stands of four decks each, the decks being spaced about 5 ft. apart. The whole stand is driven from the top and makes 12 rev. per hour.

In practically all the work, each of the eight decks has received the same kind of feed, and any middling product resulting from them has been returned to the feed of the decks. Feed has been prepared principally by Callow tanks, but the Garred filter and the Kuchs-Laist separator have been used to some extent. The first two thickeners in combination with the round tables give about the same recovery, but with the latter the recovery is slightly higher, as the separator removes a great deal of the colloid, and gives a cleaner sand product to the round table.

Table XI. gives the results of treating the products of the several thickeners on the round tables, and the effect of removing the colloids is evidenced by the part *C*, even when figured back to the original slime.

TABLE XI.—*Operation of Round Tables.*

*A.* Preceded by Callow tanks.

| | Solids. Pounds per 24 Hr. per Deck. | Assay. | | | Copper. Pounds per 24 Hr. | Per Cent. of Total | |
|---|---|---|---|---|---|---|---|
| | | Cu. | Insol. | FeO. | | Solids. | Copper |
| | | Per Cent. | Per Cent. | Per Cent. | | | |
| Feed | 9,325 | 2.18 | ..... | ..... | 204 | 100.0 | 100.0 |
| Concentrate | 1,425 | 7.31 | 55.1 | 13.8 | 104 | 15.3 | 51.2 |
| Tailing | 7,250 | 1.21 | ..... | ..... | 88 | 77.8 | 43.1 |
| Tank overflow | 650 | 1.81 | ..... | ..... | 12 | 6.9 | 5.7 |

*B.* Preceded by Garred filter.

| | Rate per Deck per 24 Hr. | | Density. Solids. | Assay. | | Copper. Pounds per 24 Hr. | Per Cent. of Total. | |
|---|---|---|---|---|---|---|---|---|
| | Gal. | Lb. | | Cu. | Insol. | | Solids. | Copper |
| | | | Per Cent. | Per Cent. | Per Cent. | | | |
| Feed | 30,625 | 11,199 | 11.9 | 1.98 | ..... | 222 | 100.0 | 100.0 |
| Concentrate | 13,803 | 1,631 | ..... | 7.02 | 58.8 | 114 | 14.6 | 51.6 |
| Tailing | 16,822 | 9,568 | ..... | 1.13 | ..... | 108 | 85.4 | 48.4 |

2. Preceded by Kuchs-Laist separator.

| Test No. | Separator Spigot Round Table Feed. Rate per 24 Hr. Gal. | Lb. | Density Per Cent. Solids. | Assay. Per Cent. Cu. | Per Cent. of Original. Solids. | Copper. | Round Table Concentrates. Per Cent. of Original. Solids. | Copper. | Per Cent. of Cu in Feed to Table. | Assay. Cu. | Insol. | FeO. |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | | | | | | Per Cent. | Per Cent, | Per Cent. |
| 1 | 8,240 | 12,260 | 16.0 | 2.26 | 67.0 | 78.2 | 16.0 | 55.9 | 71.5 | 6.75 | 59.6 | 13.0 |
| 2 | 8,380 | 11,320 | 14.5 | 2.46 | 63.6 | 78.5 | 16.7 | 58.5 | 74.5 | 6.95 | 57.7 | 13.7 |
| 3 | 9,050 | 11,300 | 13.0 | 2.53 | 65.9 | 78.6 | 14.8 | 53.6 | 68.2 | 7.69 | 52.5 | 14.9 |
| 4 | 9,800 | 13,070 | 14.0 | 2.81 | 61.0 | 76.9 | 13.3 | 56.0 | 72.8 | 9.68 | 47.1 | 16.8 |

One round table deck used in this test.

*Peck Centrifugal Concentrator.*—The Peck machines, of which there are two, have been treating the slime which has settled in one of the slime ponds, which under the present operating conditions are sloughing off an appreciable amount of colloid and fine sand. No direct comparison of the work of this machine with that of the round table can be made except where the feed for the two machines is prepared in a similar manner.

The Peck concentrator is an intermittent centrifugal machine, and consists primarily of two cones, one within the other, both revolving in the same direction about a vertical shaft. The feed is introduced through an annular feed box, attached to the shaft, into the annular space between the two cones at the lowest point, and rises through it, where the concentration takes place. Centrifugal force throws the solids to the inner wall of the outer cone, and the water in the feed, rising in the annular space, carries the lighter material along with it and is discharged through a set of plugs distributed around the upper edge of the outer cone. The result is, a layer of concentrate adheres to the wall of the cone and the tailing has been discharged through the plugs. When this layer is sufficiently thick the feed is shut off; the outer cone is slowed down, thus reducing the centrifugal force on the concentrate; and wash water is turned in. At the same time launder connections with the discharge plugs are changed, so that a middling product is discharged, and finally the concentrate itself is discharged.

The effect of centrifugal force is to increase the apparent weight of the slime constituents and thus allow of greater capacity, but there is, of course, no effect on the ratio of the specific gravities of these constituents.

The changes of the cycles, which are about 3.5 min., are accomplished by an ingenious automatic device, operated hydraulically.

The older of the two machines treats 120 tons of dry slime per day. The speeds of its cones are 473 rev. per min. for the outer cone and 271 for the inner cone. The corresponding figures for the newer machine are 150 tons, and 505 and 285 rev. per min.

*Competitive Tests on Standard Concentrating Machines.*—These four machines were fed with slime from Sections Nos. 2 to 8, slime from Section No. 1, and the total classifier overflow from Section No. 1. and the results were very consistent. Tests on any one feed were run simultaneously on all the machines, the feed being distributed by a rotary feeder to be described. The results of these tests compared with the work of the round table under operating conditions show that the best machine does slightly better work than the round table; that the next best machine does just about the same work; but that the other two are not as efficient. Assuming that the machines were doing their best work when being tested, under the supervision of representatives of the manufacturers, the round table would be just about as efficient as the best of them if they were under operating conditions.

*Twenty-Deck Round Table.*—As the round table type of concentrator has proved so satisfactory as regards floor space occupied, maintenance, operation, tonnage, and efficiency, a machine of this type with 20 concentrating surfaces superimposed one on the other has been constructed, Fig. 9. This may be called a multiple-deck concentrator, the 20-deck machine being a special case. This 20-deck round table occupies a circle of floor space 20 ft. in diameter and is 30 ft. high including the feed box. It will therefore occupy about the same space as the four-deck stands previously installed, but will have five times the capacity.

The decks are about 1 ft. apart, allowing room for inspection of each deck. The bottom deck acts as a support for the upper ones, and is itself supported near the periphery by wheels running on a circular track. The power is applied by a motor, through gears, to a circular rack attached to the lower deck. The surfaces of the decks are concrete made of two parts tailing sand which is under 1.5 mm. and one part cement, and are supported by sheet steel. The concrete is reinforced with expanded metal lathing fastened to the sheet metal. The rest of the machine, other than the concentrating surfaces, is of steel. Suitable steel launders are provided on each deck for the various products of the decks. Each deck also has its own feed and wash water box.

In the center of the structure, and passing through each deck, is a circular opening, designed as a shaft to carry the feed and water

pipes, and sufficiently large to permit the table operator to pass up and down and inspect the decks and feed boxes.

No formal tests have been made as yet, but preliminary sampling of the products indicate that the 20 decks are handling 140 tons (dry weight) of slime in 24 hours containing 2 per cent. copper, and producing a concentrate containing 7.25 per cent. copper, and a tailing containing 1.10 per cent. copper. The recovery is about 53 per cent.

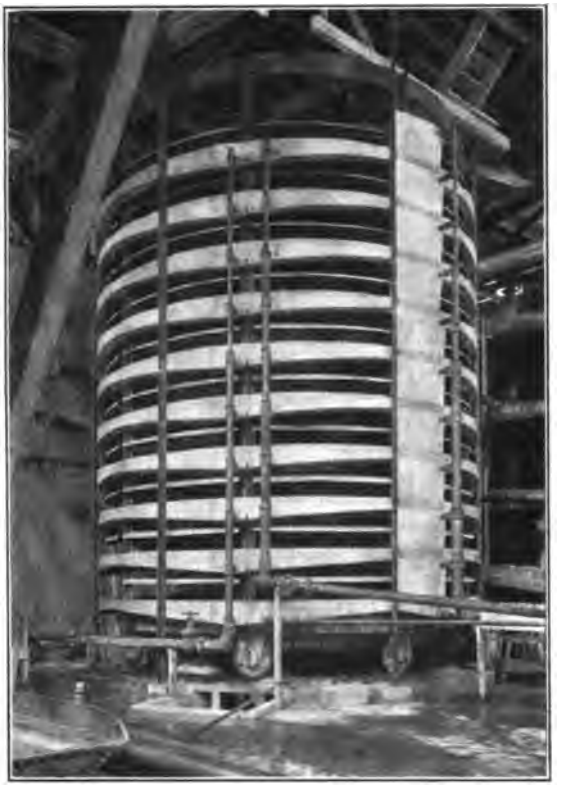
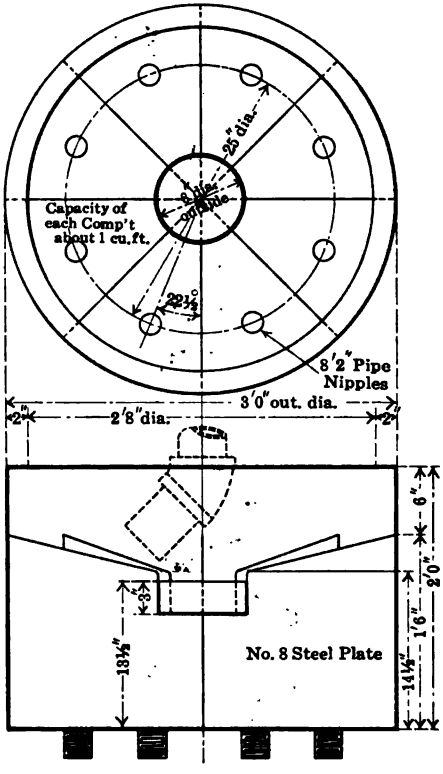


FIG. 9.—TWENTY-DECK ROUND TABLE.

*Slime Feed Distributor.*—In connection with test work in the slime plant, there has been devised a feeding device, Fig. 10, to give equal distribution of feed to any set of machines to which it is desired to send the same feed. The same device is used as a distributor for the 20-deck round table. It consists of two cylindrical tanks, one within the other, and the annular space between divided radially into as many equal parts as there are machines to be fed. Each compartment is connected by a pipe to a machine. Above the tanks

revolves a pipe with a 45° elbow on it, which distributes the feed to the several compartments. The top edge of the inner cylinder is slightly lower than the radial partitions, so that if one machine has to be closed down the feed which went to its compartment would overflow into the inner cylinder, which in turn would overflow at the other compartments, and the feed would still be equally divided. This device has been checked by time samples for tonnage, volume, and assay, and found to give very accurate distribution of the feed.




CROSS-SECTION ON CENTER LINE

FIG. 10.—FEED DISTRIBUTOR FOR SLIMES.

## IX. CONCLUSION.

The data given in the preceding part of this paper are representative of the work which has been done on the problem. In many cases the figures given are averages of several tests or are tests of several days' duration. Moreover, operating tests have been cited where possible, as these are the conditions under which machines have to work in practice. There are therefore sufficient data to indicate the mode of procedure followed, and the results obtained in the attempt to solve the slime problem.

It is evident that large settling areas are required to thicken the slime where free settling is to be used. If filters or centrifugal machines are resorted to, then space can be economized, but increased maintenance costs and operation costs, together with complex machines, enter into the problem.

The Dorr tank has nearly the same capacity per square foot of settling area as the smaller units; namely, 0.20 to 0.25 gal. per square foot per minute; but it has the advantage of large units, say 50-ft. tanks, which can be built 3 ft. deep and three in a stand; and consequently requires few discharge plugs, and these of good size. The filter is a very compact and efficient machine, but has the disadvantage of operating in cycles as well as having high maintenance costs.

The Kuchs-Laist separator is very compact and is continuous in operation, but requires power, and also a satisfactory discharge plug to give constant density. Its product is freer of colloids than that of the other thickeners, which allows concentrators to make a greater net recovery from it. If a uniform spigot density could be maintained this machine would be very satisfactory.

The round table has proved very efficient for handling slime, it being simple in construction and operation, and a number of decks may be placed one above the other, thus economizing on floor space, where the mill height is available.

The Peck machine has a good principle, namely, the utilization of the effect of centrifugal force, which allows it to handle a good tonnage in a small space. It makes a good recovery on the slime settled in the ponds.

The best net recovery of copper in the original slime is from 50.0 to 52.0 per cent. except where the Kuchs-Laist machine is used. The slime contains about 20 per cent. of colloidal copper, which is not recoverable by wet concentration; and there are middling grains to be considered as well. Thus the net recovery of recoverable copper is probably 65 per cent., which is very good work for wet concentration on this class of material.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.
[SUBJECT TO REVISION.]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Butte meeting, Aug. 18 to 21, 1913, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1913, when Vol. XLVI. of the *Transactions* will go to press. Any discussion offered thereafter should preferably be in the form of a new paper for publication in Vol. XLVII. (with suitable cross references in both volumes).

# Application of Hindered Settling to Hydraulic Classifiers.

BY EARL S. BARDWELL, GREAT FALLS, MONT.

(Butte Meeting, August, 1913.)

In his paper entitled, Development of Hindered-Settling Apparatus, Dr. Richards has related the history of the development of the hindered-settling classifier and given illustrations of the several types of this apparatus which he has designed.[1] An earlier paper,[2] Close Sizing Before Jigging, shows the nature of the hindered-settling phenomenon and the results which it produces. In the first-mentioned paper Dr. Richards describes the method of getting hindered settling and shows the relation that the constriction bears to the classifier. The question of proper ratios between the area of the sorting column and the area of the constriction opening Dr. Richards leaves open, aside from certain statements as to the ratios which have been found to give good results in practice.

While working with a hindered-settling classifier of the type now in use at the Boston & Montana Reduction Plant of the Anaconda Copper Mining Co. a number of interesting facts were brought out which have thrown considerable light on the question of proper ratio between the area of the sorting column and the area of the constriction and which will be made the subject of this paper.

*Free and Hindered Settling.*—If a mineral particle be dropped into a vessel containing still water, the particle will sink in accordance with the laws of free settling in water, which are as follows:

1. Of two grains having the same specific gravity but differing in size, the larger will settle in still water at the greater velocity.

2. Of two grains having the same size but differing in specific gravity, the heavier grain will settle at the greater velocity.

From these laws it becomes evident that the velocity of settling depends primarily on the specific gravity and the size of the mineral grain. The fracture of the mineral may exert a very considerable

[1] *Trans.*, xli., 421 to 439 (1910).
[2] *Trans.*, xxiv., 409 to 486 (1894).

influence on the settling velocity. Rittinger[3] has expressed the settling velocity of mineral grains by the formula, $V = K \sqrt{D(G-1)}$, where V = velocity in millimeters per second, K is a constant dependent in part on the shape of the mineral grain as determined by its characteristic fracture, D is the diameter of the grain in millimeters, and G its specific gravity. Richards[4] has shown that this general formula holds true only within a specified range of sizes; in the case of quartz down to 0.2 mm.

If the mineral grain be subjected to the action of a rising current of water, as in a hydraulic classifier, we find that the settling velocity is exactly equal to the velocity of the rising current of water which will keep the grain poised, neither causing it to rise nor allowing it to sink.

If, instead of a single mineral grain, we drop a considerable number of grains into the vessel containing still water, we find that some of the grains are prevented from settling at their normal velocity by reason of the movement of neighboring grains. The grains are hindered in their efforts to settle, and this I shall term the hindered-settling effect. The same effect is observed to an even greater extent when the grains are subjected to the sorting action of a rising current of water, as in a hydraulic classifier. The action is similar to that which we might expect were the grains settling through a medium of higher specific gravity than that of water. The constriction which is placed in the sorting column of the hindered-settling classifier causes to be maintained in the sorting column a bed of grains, each grain of which is poised freely in the water. Dr. Richards has likened these grains to the grains of sand sometimes observed in similar motion in a boiling spring. This quicksand column acts as a filter, through which the other grains must make their way in order to be able to pass out of the apparatus through the spigot. The smaller grains of lighter material are unable to penetrate this bed to any depth and are consequently turned back, thus giving a spigot product with the high diametral ratios between average light mineral and average heavy mineral particles which are characteristic of hindered-settling classifiers.

Thus hindered settling may take place in a free-settling classifier when the classifier is overfed. The hindered-settling classifier differs from the free-settling classifier in that it is designed to make the best possible use of this hindered-settling effect and at the same time to furnish means of controlling and maintaining these conditions.

[3] *Lehrbuch der Aufbereitungskunde*, p. 191 (1867).

[4] *Trans.*, xxxviii., 231 to 235 (1907).

*The Constriction.*—The constriction is the essential part of the hindered-settling classifier. Without some form of constriction at the base of the sorting column or teeter chamber no classifier can properly be classed as a hindered-settling classifier.

The constriction may consist of a plate or diaphragm with a central circular opening of less diameter than that of the sorting column or with a series of smaller circular openings, or of a plate in which a series of short pipes or tubes have been inserted. The latter form of constriction has been found to give the best results when the classifier is designed to overflow quartz grains with a diameter of 0.75 mm. or larger. The former is sufficient in the case of deslimers.

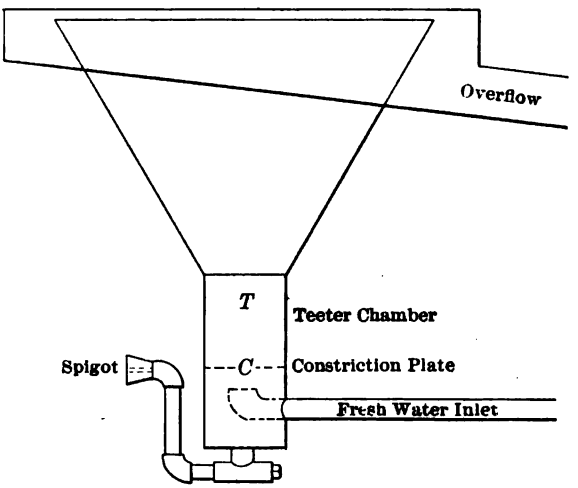



FIG. 1.—SINGLE-POCKET CLASSIFIER.

The effect produced by the constriction is as follows: It is evident from Fig. 1, which represents the type of classifier in use at the concentrator of the Boston & Montana Reduction Works of the Anaconda Copper Mining Co., that with a given volume of water rising through the constriction, *C*, the velocity through this constricted area will be greater than the velocity of the rising water in the teeter chamber, *T*; the velocities varying inversely as the respective cross-sectional areas.

Such being the case, grains of sand, the free-settling velocities of which are greater than the rising velocity in the teeter chamber, will sink and find their way into the teeter chamber, *T*. Such of these grains as have a free-settling velocity greater than the rising velocity through the constricted area will pass through the opening in the

constriction plate and out the spigot. Grains which are able to sink into the teeter chamber but are not able to sink through the swifter rising current in the constricted area accumulate in the teeter chamber and remain there poised freely in the water, moving among themselves and forming a quicksand bed which is in its effect somewhat analogous to a liquid having a higher specific gravity than water.

When grains have thus accumulated in the teeter chamber until a point is reached such that, in any cross-section taken through the sorting column or teeter chamber, the area occupied by sand is to the area occupied by water as the constricted area is to the area of the sorting column, then a condition of equilibrium is reached, which the classifier, if properly designed, will maintain. The classifier may now be said to have become bedded. The grains which now enter the classifier, and which are able, by reason of their higher settling velocity, to penetrate the teetering mass of grains in the teeter chamber, tend to upset this state of equilibrium and will either pass through the teeter chamber and out the spigot, or, if they have not a settling velocity sufficiently great, will, by increasing the density of the teetering column, cause a sufficient amount of material to be discharged through the spigot to restore the condition of equilibrium above mentioned. The bed or teetering mass of grains in an open-spigot classifier may be considered as composed of grains of a definite size; *i. e.*, they are the grains which will sink into the teeter chamber and which are not able under free-settling conditions to pass through the opening in the constriction plate.

Observation of classifiers under operating conditions has shown that for each size grain there is a certain density that can be maintained in the teeter chamber, and that the ratio existing between the area of the sorting column and the area of the constriction opening depends primarily upon this permissible density. Dr. Richards points out that when the ratio is too large full teeter will take place in the sorting column, but no sand will be discharged through the spigot, while if the ratio is too small there will be no bed; in other words, the classifier will act by free settling.

With a closed-spigot classifier, such as Dr. Richards used in his experiments, this statement holds true. In the case of an open-spigot classifier it appears that if the ratio between the area of the sorting column and the area of the constriction opening be too large the result is a decrease in the capacity of the classifier and tendency to banking. There is a ratio which gives the maximum capacity obtainable from the classifier, and this ratio is the one arrived at by design-

ing the classifier to maintain a teetering column of maximum permissible density.

The permissible density which can be maintained with any given size of grains is that density which exists when all of the grains in the sorting column or teeter chamber are poised in the water, free to move up or down among themselves. This condition Dr. Richards has very appropriately called "full teeter." The density at full teeter varies, being greater when the bed of grains in the classifier is composed of large grains than when it is composed of finer particles. Experiments which will be described later seem to indicate that this permissible density varies inversely as the square root of the total surface exposed in a given weight of grains. This law apparently holds within the range of sizes that are usually handled by classifiers. There is an upper limit to the density that it is theoretically possible to maintain, and when this has been reached the theoretical ratio of the area of the sorting column to the area of the constriction opening becomes unity. Beyond this point we are unable to maintain a bed of teetering grains and the classifier acts by free settling. As this point would be reached in a classifier with which we wished to overflow 2.35-mm. material, it is of no practical consequence as far as the problem of hydraulic classification is concerned.

*Permissible Density.*—In order to determine what the full-teeter densities were the following experiment was tried. A quantity of tailings material was screen sized and the separate sizes treated in a sorting tube with closed spigot. This sorting tube was similar to the one used by Dr. Richards in his experiments. The weight of the grains of each size contained in a given volume at full teeter was determined. The densities which were obtained in this way formed a series varying from 4,618 g. to the gallon in the case of grains having an average diameter of 2.19 mm. to 866 g. to the gallon in the case of grains 0.07 mm. in size. A considerable number of relations were tried, but it was finally found that the relation that seemed to hold most nearly true was that the densities varied inversely as the square root of the total surface exposed in 1 g. of grains of each of the given sizes. The assumption was made that the particles were spheres with diameters equal to the average screen size. On this assumption the volume of one grain, the weight in grams of one grain, the number of grains in 1 g., the surface of one grain in square millimeters, and the total surface exposed in 1 g. of grains of each size were computed. Table I. shows the experimental results side by side with the computed results.

TABLE I.—*Permissible Densities of Quartz Grains in Quicksand Column.*

| Diameter Mm. | Volume 1 Grain. Cu. mm. | Weight 1 Grain Grams. | Number Grains in 1 Gram. | Surface 1 Grain. Sq. mm. | Total Surface 1 Gram. | Sq. Root Total Surface. | Density at Full Teeter. Grams per Gallon. | Permissible Density. | Cu. mm. Sand in 1 Gallon. | Cu. mm. Water in 1 Gallon. | Ratio. |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 2.19 | 5.5 | 0.0146 | 69 | 15.05 | 1,040 | 32.2 | 4,618 | 4,830 | 1,830,000 | 1,955,000 | 1.0[illegible] |
| 1.84 | 3.26 | 0.00863 | 116 | 10.62 | 1,235 | 35.2 | 4,489 | 4,420 | 1,670,000 | 2,115,000 | 1.2[illegible] |
| 1.54 | 1.91 | 0.00506 | 197 | 7.45 | 1,466 | 38.4 | 4,239 | 4,050 | 1,520,000 | 2,265,000 | 1.4[illegible] |
| 1.20 | 0.903 | 0.00239 | 419 | 4.53 | 1,900 | 43.5 | 3,989 | 3,580 | 1,350,000 | 2,435,000 | 1.[illegible] |
| 0.92 | 0.408 | 0.00108 | 926 | 2.65 | 2,460 | 49.6 | 3,747 | 3,140 | 1,190,000 | 2,595,000 | 2.1[illegible] |
| 0.841 | 0.311 | 0.000825 | 1,212 | 2.22 | 2,691 | 51.9 | 3,000 | 3,000 | 1,131,000 | 2,654,000 | 2.34 |
| 0.77 | 0.239 | 0.000635 | 1,580 | 1.86 | 2,940 | 54.2 | 3,353 | 2,870 | 1,085,000 | 2,700,000 | 2.4[illegible] |
| 0.60 | 0.113 | 0.000300 | 3,330 | 1.13 | 3,770 | 61.5 | 2,763 | 2,530 | 953,000 | 2,832,000 | 2.97 |
| 0.50 | 0.0655 | 0.000173 | 5,780 | 0.785 | 4,530 | 67.4 | 2,311 | 2,310 | 875,000 | 2,910,000 | 3.3[illegible] |
| 0.42 | 0.0388 | 0.000103 | 9,700 | 0.555 | 5,380 | 73.3 | 2,573 | 2,120 | 800,000 | 2,965,000 | 3.7[illegible] |
| 0.35 | 0.0224 | 0.0000595 | 16,800 | 0.385 | 6,470 | 80.4 | 1,937 | 1,940 | 729,000 | 3,056,000 | 4.2[illegible] |
| 0.26 | 0.0092 | 0.0000244 | 41,000 | 0.212 | 8,700 | 93.5 | 1,684 | 1,670 | 630,000 | 3,155,000 | 5.0[illegible] |
| 0.166 | 0.0024 | 0.00000635 | 15,7000 | 0.0866 | 13,600 | 117 | 1,333 | 1,330 | 504,000 | 3,281,000 | 6.5[illegible] |
| 0.070 | 0.00018 | 0.000000476 | 2,100,000 | 0.0154 | 32,300 | 180 | 866 | 870 | 327,000 | 3,458,000 | 10.5[illegible] |

In Table I. the line marked "Density at Full Teeter" gives the densities that were determined by experiment and are in fact the average of several results which agreed closely. The density in the case of grains 0.841 mm. in diameter being practically what we have found in practice, was assumed to be correct, and the values given in the line marked "Permissible Density" were computed on the assumption that the densities varied in the manner stated in the preceding paragraph. As will be seen, this relation seems to hold approximately true.

The ratio of area of sorting column to area of constriction opening may be computed from the formula $R = \frac{K}{\sqrt{D}} - 1$, where K is a constant approximately equal to 3.

We have so far considered the bed as composed of quartz grains of the size which would not ordinarily be able to penetrate the current rising through the constriction opening and which would not overflow under free-settling conditions. As a matter of fact, there must be some middlings grains and some heavy mineral grains in the classifier bed. This would not affect the results to any considerable extent. Wherever we have had occasion to use constrictions with these ratios in classifiers the classifiers have carried a bed and have given satisfactory spigot products. The curve, Fig. 2, covers the range of sizes with which we are ordinarily concerned in designing classifiers. The size or diameter in millimeters of the maximum quartz grain to be overflowed is plotted as abscissa and the corresponding ratio between the area of the sorting column and the

area of the constriction opening is plotted as ordinate. Thus in the case of a four-spigot classifier in which we wish to make the first spigot deliver 2 to 1.25 mm. quartz; the second spigot, 1.25 to 0.75 mm.; the third spigot, 0.75 to 0.35 mm., and the fourth spigot 0.35 to 0.10 mm. quartz, the ratio should be 1.8, 2.6, 4.2, and 8.9, respectively. To work properly, each pocket would have to be designed with reference to the tonnage of each of these sizes which the classifier might be expected to handle.

Having established the proper ratios for area of sorting column to area of constriction, it may be of interest to see how the other computations are made.

*Size of Constriction Opening.*—In designing a hindered-settling classifier the size of the constriction opening is the factor that determines the capacity of the classifier. It is, therefore, the part of the classifier which must first receive attention.

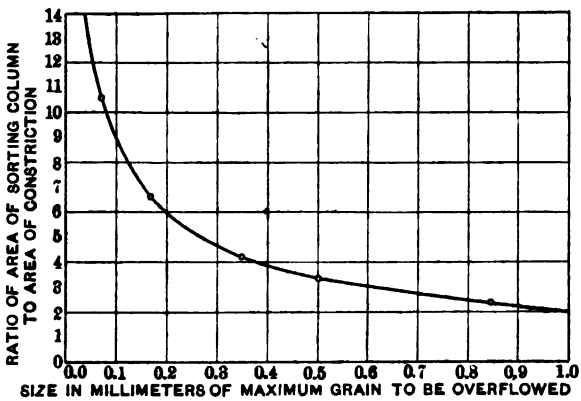



FIG. 2.—CURVE SHOWING PROPER RATIO OF CONSTRICTION TO BE USED IN SINGLE-POCKET CLASSIFIER IN ORDER TO OVERFLOW QUARTZ OF DIAMETER SHOWN BY ABSCISSA.

The minimum size of constriction opening depends upon the amount of spigot product that the classifier is to be called upon to deliver. If we know the average diameter of quartz grain which is to be in the spigot and the maximum tonnage of spigot product to be delivered, we would proceed as follows: From a table of settling velocities[5] find the average settling velocity of the average quartz grain above referred to. Now, from the tonnage calculate the volume that must pass the constriction in a given time; say, in one second. Divide the volume found in this way by the free-settling velocity of the average quartz grain, the former being in cubic inches and the latter in inches per second, and we have the area that would be re-

[5] *Trans.*, xxxviii., 229 (1907).

quired for the opening in order that it should just pass this volume of material. We have found from experience that this area must be multiplied by about 3, for reasons that will be stated later. This latter statement refers only to classifiers which have a rising current of water in the sorting column. The area found in this manner must now be multiplied by the proper ratio, as shown on the curve, Fig. 2, in order to obtain the area of sorting column that will be necessary.

*Design of Classifier Top.*—The top of the classifier should be so designed that the rising current at the top due to the combined effect of fresh water and feed water is equal or nearly equal to the rising current through the constriction or to the free-settling velocity of the largest quartz grain which the classifier is designed to overflow. If this is done, then the finer particles which we wish to keep out of the spigot, and which are not able to penetrate the bed of teetering grains in the sorting column, overflow at once. When the classifier is designed with a conical top or pocket and when the feed water quantity varies within wide limits it is practically impossible to so design a classifier as to enable it to do uniformly good work. When, however, the classifier is to treat the spigot of a deslimer and the volume of feed is kept constant the top can be so designed as to do very accurate sizing. In practice it has been found advantageous to feed a classifier of this type through a central feed cone so designed that the annular space at the bottom will give a rising current equal to the rising current through the constriction, with the annular space at the top approximately 25 per cent. greater than that at the bottom. In this way the full effect of the feed water may be utilized for the work of sorting.

*Method of Designing a Classifier.*—It may be of interest to outline the method that is used in designing a classifier of the general type illustrated in Fig. 1 so as to satisfy a given set of conditions. We will suppose a case in which we want a single-pocket classifier capable of desliming 500 tons per 24 hours, the material being the undersize of 5-mm. trommels. We will further assume that we have 8 ft. of head room, that we wish the deslimed product to have a density of 1,200 g. to the gallon, and that the feed to the classifier contains 10 per cent. of material finer than 200 mesh which we wish to overflow as slime. The feed water amounts to 300,000 gal. per 24 hours.

*Size of Spigot.*—In order that we may obtain a spigot discharge with a density of 1,200 g. to the gallon, the spigot discharge must amount to 340,200 gal. per 24 hours, $\frac{450 \times 2{,}000 \times 453.6}{1{,}200} = 340{,}200$. This

is equivalent to 3.938 gal. per second, or 910 cu. in. per second. With a 2.25-in. spigot, which has an opening equal to 3.97 sq. in., we would require a velocity of 910 ÷ 3.97, or 229 in. per second in order to obtain the above-mentioned spigot discharge. From the formula $V^2 = 2gh$, we can compute the head room theoretically required for the classifier. This comes out equal to about 68 in., or 5 ft. 8 in. This would be satisfactory, as we have allowed ourselves 8 ft. of head room. To allow for friction we would probably so construct the classifier as to have the overflow about 6 ft. higher than the spigot opening. This may be further modified when we come to design the classifier top.

*Area of Constriction Opening.*—If we are to make 450 tons per 24 hours of spigot product then we shall have:

$$\frac{450 \times 2{,}000}{86{,}400} = 10.42 \text{ lb. per second, or}$$

$$\frac{10.42 \times 1{,}728}{64.5 \times 3.1} = 90.05 \text{ cu. in. per second.}$$

This calculation is based on the assumption that the specific gravity of the material is 3.1.

We will assume that we have made a sizing test on the feed after first screening out the through-200-mesh material, and that we have obtained results as follows:

| Screen Size. | Per Cent. Total. | Cumulative Per Cent. |
|---|---|---|
| On 5 mm. | 0.6 | 0.6 |
| 2.38 | 23.2 | 23.8 |
| 1.41 | 23.5 | 47.3 |
| 0.841 | 19.2 | 66.4 |
| 0.500 | 13.3 | 79.8 |
| 0.350 | 4.2 | 84.0 |
| 0.166 | 10.6 | 94.6 |
| 0.078 | 5.4 | 100.0 |
| Total | 100.0 | |

The average screen size figures out 1.7 mm., and this grain has a settling velocity of 136 mm. per second, or, say, 5.35 in. per second. In order that 90.05 cu. in. of sand may pass the constriction with an average velocity of 5.35 in. per second the constriction opening must have an area of at least 16.8 sq. in. To allow for variations in feed we will make the constriction opening 5 in. in diameter, giving us an area of 19.6 sq. in.

*Size of Sorting Column.*—If we were making the calculations for a

classifier which was to be operated with a rising current and intended to overflow 200-mesh material we would multiply the constriction area which we have just found by 10.56, the ratio obtained from the curve, Fig. 2. This would necessitate a sorting column 16 in. in diameter. For a deslimer we have found that this does not seem to be necessary and that a 12-in. sorting column will answer the purpose. In the case of a deslimer the sorting column should be made at least 4 in. in length. The coarser the material which we wish to overflow the longer should the sorting column be. A sorting column 8 in. long is, however, sufficient in most cases.

*Area of Classifier Top.*—A grain of quartz 0.078 mm. in diameter settles at the rate of 4 mm. per second. We will therefore make the rising velocity at the top of the classifier 3.5 mm. per second, in order to be sure that we overflow the least possible amount of on-200-mesh material. In a deslimer of this sort we need use only enough fresh water to supply the spigot, and all the water which will overflow is the water which comes to the deslimer with the feed. This we have stated amounts to 300,000 gal. per 24 hours, or $\frac{300,000}{86,400} = 3,472$ gal. per second, or 802 cu. in. per second.

If our rising velocity is to be 3.5 mm. per second, or 0.138 in. per second, we must have a top with an area of 5,812 sq. in. or a top with diameter of 86 in. If we make our cone 48 in. in vertical height, the classifier will have to be somewhat more than 6 ft. in vertical height, in order to allow for the sorting column, pressure chamber, and spigot discharge. About 7 ft. of head room would be required. As we have 8 ft. of head room, this will work out nicely. The classifier designed in this way will deliver its overflow with a density of 150 g. to the gallon.

As was stated previously, a classifier in which there is to be a rising current would be designed in a similar manner; the only difference being that when the size of the constriction opening has been computed the area found should be multiplied by about three. The reason for this is that with a given rising velocity through the constriction, and a definite volume of sand sinking through the constriction at a known average velocity, if the area of the constriction opening is made double the sand area the rising velocity is doubled and the classifier will bank or choke, since the rising velocity becomes greater than the average rate of settling of the grains in the spigot discharge. We will suppose a case in which the average settling velocity of the grains going into the spigot discharge is 170 mm. per second and where we have a rising velocity of 100 mm. per second.

In this case, if the sand area were doubled to get the constriction area the effective rising velocity would become 200 mm. per second. If the sand area be multiplied by three then the constriction area when sand is passing is reduced one-third and the rising current becomes 150 mm. per second, or may have this value momentarily. Now, the average settling velocity is 170 mm. per second, and experience has shown that under these conditions the classifier will operate satisfactorily and will handle its rated tonnage. It might in some cases be advisable to make the constriction opening a little larger in order to be on the safe side, but in order that we may make the most economical use of water the classifier should be made only sufficiently large for the work that it is called upon to do.

If the classifier is made too large for the amount of feed which it is to handle there will not be enough material coming into the classifier to maintain a bed and the classifier will tend to act by free settling. Classifiers designed in the manner outlined have been found to operate satisfactorily and to be extremely efficient as regards water consumption.

In the mills we have two variables which affect the operation of classifiers; namely, variations in tonnage and variations in density of feed. For this reason a classifier, to do its best work, should be especially designed to meet the conditions under which it is to operate. This is, however, impracticable from the point of view of the manufacturer of milling machinery, who is obliged to standardize the parts and confine himself to a few standard sizes. Of the two variables, the one which is the more serious as regards operation of the classifier is the variation in volume of feed. This can be made negligible in classifiers of this type by desliming the pulp previous to classification, the classifier receiving its feed directly as the spigot product of a deslimer. In this way a few standard sizes may be designed so as to satisfy practically any condition which may arise in practice.

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# Preparation of Ore Containing Zinc for the Recovery of Other Metals Such as Silver, Gold, Copper, and Lead by the Elimination and Subsequent Recovery of the Zinc as a Chemically Pure Zinc Product.

BY S. E. BRETHERTON, SAN FRANCISCO, CAL.

(Butte Meeting, August, 1913.)

THIS title introduces the subject I wish to describe to my fellow members, very few of whom, I hope, have ever had as much trouble with the smelting of ore containing much zinc, either in the lead blast furnace or the matting furnace, as circumstances have forced upon me: First, for several years in Leadville, Colo., after the sulphide ores containing zinc had been developed with greater depth and were sold to the interests with which I was connected for smelting in the blast furnace for the recovery of the lead, silver, and gold; then, in New Mexico and Arizona, where the principal source of ore supply for our custom smelter was from miners who sent most of their product in the form of concentrates, which we briquetted and smelted in the blast furnace, producing a matte containing copper, silver, and gold, for which we were paid; together with some zinc and lead, which was penalized if in excess of a maximum base set by the refinery to which we sold our copper matte; last, but not least, my experience in Shasta county, Cal., where for the sake of recovering $13 per ton value in copper, silver, and gold, I have smelted ore containing as high as 40 per cent. zinc (the last 18,951 tons smelted averaged $15\frac{3}{10}$ per cent. zinc) which we not only lost, but which it cost us money to lose.

After such an experience, and knowing that, instead of being worth only $13 per ton, such 15 per cent. zinc ore should be worth about $30, including zinc at the present market value, the question of recovering the zinc in a profitable manner from such ore became very interesting to me. During several years' time I sent samples, of from 5 to 1,200 lb. each, to be tested by the different best-known zinc-recovery plants, and experimented with several processes proposed by others for the treatment of such ore, with but little encouragement, until I

tried leaching with ammonia and carbon dioxide. I referred to this process and to my objections to other processes in the *Mining and Scientific Press* of April 12, 1913. The discouraging feature of this process was that, after using it at the Hoboken works, near Antwerp, for several years, it was reported as being abandoned, although it was claimed that the loss of ammonia was suprisingly small. Since we have worked this process on a somewhat larger scale than laboratory tests, I am not surprised at any trouble they may have encountered when leaching zinciferous oxides left after cupellation; in fact, to get the proper roast is the most delicate part of the operation.

My first attempt to leach with ammonia and carbon dioxide ($NH_3$ and $CO_2$) was made upon flue dust containing 30 per cent. zinc, just as it came from the dust chamber without any preparation, with the result that an extraction of 72 to 78 per cent. of the zinc was obtained. This I considered good for the reason that some of the zinc in the flue dust is there carried over mechanically as fine particles of insoluble ZnS. I then had some ore roasted in a custom assay office, and attempted to leach it, but with only 56 per cent. zinc extraction. The trouble in this test was that the ore had been roasted at too high a temperature, so that some of it was partly sintered or fused. Knowing the cause of this poor extraction, I was not discouraged, but discontinued my experiments because my time was too fully occupied with other matters. However, I suggested to some persons who had a large quantity of flue dust rich in zinc that they try out the ammonia process, which they did with very good results as to the zinc extraction, but the zinc oxide product was comparatively worthless, due to the arsenic content which had been originally in the siliceous ore used in the blast furnaces for flux; and due to concentrating the arsenic from these ores into the flue dust and a further concentration when it was extracted with the zinc in the form of oxides. This and more recent experiences have proved to us that, while the ammonia process is suitable—and I feel safe in saying the most suitable process—for the treatment of the zinc ore of Shasta county, Cal., where I have not found more than traces of arsenic in such ore, the process is not suitable for fused products or ore containing much arsenic or zinc silicates, the latter being insoluble.

The following results were reported by our Metallurgical Chemist, F. L. Wilson: First, by what we call the bottle test, that is, taking 100 g. (about 3½ oz.) of the finely ground, roasted ore from the Afterthought mines of Shasta county, Cal., and agitating in an ordinary 6-lb. acid bottle on rollers for 20 hr. for leaching and then for 4 hr. for washing. The results of these tests are shown in Table I.

TABLE I.—*Preliminary Bottle Tests.*

| Roast. | Contents. Per Cent. | | Extractions, Per Cent. | | | |
|---|---|---|---|---|---|---|
| | | | Solution. | | Residue. | |
| | Zn. | Cu. | Zn. | Cu. | Zn. | Cu. |
| Sulphatizing. Lime added 20 minutes before finish, . . . . | 20.80 | 2.12 | 83.00 | 49.10 | 84.60 | 43.90 |
| Sulphatizing. Lime added at beginning, . . . . . | 20.04 | 2.71 | 95.00 | 60.33 | 83.40 | 56.10 |
| Sulphatizing. Gypsum added at beginning, . . . . . | 19.04 | 2.35 | 85.68 | 62.77 | 83.14 | 51.35 |
| Sulphatizing. 2 hr. roast, . . . | 21.38 | 3.00 | 88.30 | 58.66 | 83.55 | 62.40 |
| Sulphatizing. 3 hr. roast, . . . | 35.98 | 0.93 | 100.0 | 83.2 | 87.52 | 85.37 |
| Sulphatizing. 3 hr. roast, . . . | 36.09 | 3.27 | 95.32 | 70.27 | 91.44 | 67.89 |
| Sulphatizing. 3 hr. roast, . . . | 33.40 | 2.31 | 92.60 | 83.4 | 92.10 | 80.90 |
| Sulphatizing. 3 hr. roast, . . . | 36.86 | 1.28 | 93.60 | 52.0 | 94.40 | 88.30 |
| Sulphatizing. 3 hr. roast, . . . | 19.70 | 2.67 | 93.80 | 60.9 | 89.13 | 71.16 |
| Sulphatizing. 3 hr. roast, . . . | 21.0 | 2.58 | 86.38 | 53.13 | 81.33 | 30.00 |

Second, by a continuation of the tests with 25- to 50-lb. lots, concluding with tests made under pressure in 8 hours' time for leaching, precipitating copper and washing, instead of 24 hr. The data are given in Table II.

TABLE II.—*Tests on* 25- *to* 50-*lb. Lots.*

| Roast. | Contents. Per Cent. | | Time Hr. | Extraction-Residue Assay. | | |
|---|---|---|---|---|---|---|
| | Zn. | Cu. | | Zn. | Cu. | |
| Sulphatizing. 3-hr. roast, | 21.3 | 2.99 | 24 | 82.16 | 50.83 | ..... |
| Sulphatizing. 3-hr. roast, | 21.3 | 2.99 | 24 | 82.4 | 53.54 | ..... |
| Sulphatizing. 3-hr. roast, | 21.7 | 2.77 | 24 | 83.73 | 57.76 | ..... |
| Sulphatizing. 3-hr. roast, | 21.7 | 2.77 | 24 | 81.84 | 50.0 | ..... |
| Sulphatizing. 3-hr. roast, | 21.5 | 2.40 | 8 | 81.8 | ..... | Pressure. |
| Sulphatizing. 3-hr. roast, | 22.12 | ..... | 8 | 80.0 | ..... | Pressure. Improper roast. |
| Sulphatizing. 3-hr. roast, | 21.5 | 2.43 | 8 | 79.4 | 53.7 | Pressure. Improper roast. |
| Sulphatizing. 3-hr. roast, | 21.8 | ..... | 8 | 87.46 | ..... | Pressure. Correct roast. |

In Table I., over the two columns showing zinc and copper extraction is marked solution, then residue; this is simply to show that it was by determining the zinc in the residue that the results were finally obtained; assaying the solutions was a quicker method and showed a higher percentage of extraction, but the results were not so reliable. The copper extraction was not as large, but in our case, where we intend to smelt the residue for the remaining copper with the silver and gold, the less copper extracted the better. Where the copper extraction also is sought, more time is necessary for leaching.

In order that the members of this Institute will more readily understand this process, with its advantages, I will briefly describe it: The

ore is first crushed to less than walnut size and ground, either wet or dry, in a Hardinge ball mill, to about 30 mesh; roasted in a Wedge type of roasting furnace at comparatively low temperature to about 7 per cent. sulphur, the object being to oxidize the zinc and form as much zinc sulphate as possible. (This crushing and roasting would have to be done for any method of treatment; in fact the high-grade concentrates have to be roasted for retorting purposes until all the sulphur possible is eliminated.) After roasting, the ore is ground in an ordinary pebble mill to 200 mesh—or perhaps I should say slimed. The slimed ore is now agitated in a solution containing preferably about 9 per cent. ammonia and 9 per cent. carbon dioxide, as stated by Schnabel in his hand-book of metallurgy. After sufficient agitation has been given the pulp for proper extraction, it is allowed to settle and the solution drawn off by decanting or filtering, or both; the pulp is properly washed to avoid mechanical loss of ammonia. The residue is smelted by the ordinary reverberatory type of matting furnace; in some cases cyanidation was adopted successfully where copper does not interfere, since the sliming and sweetening will certainly have been accomplished and only 50 to 60 per cent. of the weight of the original ore remains to be treated, depending on the original zinc and sulphur contents.

It is not necessary to go further into the treatment of the residue in this article, except to state that in washing we found it necessary to wash, first, with a solution containing ammonia and carbon dioxide to avoid precipitating some of the zinc as a basic carbonate before it was washed out; then wash out the ammonia with water steam, boiled solution, or air. The first wash solution, rich in ammonia and carbon dioxide, containing some zinc, can go direct to the agitation tanks for leaching purposes; the second wash water can be brought up to strength with ammonia and carbon dioxide, which has been boiled over from the still; the other wash waters can be used over again for washing, and the last wash waters can go to the pebble mill. Enough fresh wash water will have to be added to each fresh charge of ore to make up for the mechanical loss carried off in the residue, and no more.

I will now refer back to the solution containing zinc and some copper, as it comes from the agitation tanks. To precipitate the copper from this we pump it through a long upright tank containing sheet zinc, either old scrap purchased from the junk shop, or made by ourselves locally. The copper is allowed to settle, drawn off, filtered, washed, and thrown into the matting furnace with the residue. After the solution is free of its copper, it is forced into the still—a battery

of conical-bottomed tanks—where the free ammonia and carbon dioxide are distilled off into absorbers and a basic zinc carbonate allowed to settle, which is drawn off and filtered. Very little washing is necessary for this zinc carbonate for the reason that any free or combined ammonia and carbon dioxide is recovered from the Wedge muffle furnace when the zinc is calcined to zinc oxide. We find this zinc oxide almost chemically pure and ready to market.

In order to make our own sheet zinc for precipitating copper and avoid the high price of such pure spelter, we intend to convert part of our zinc into metal direct from the basic zinc carbonate, thus avoiding the expense of calcining. The carbon dioxide as liberated is conveyed back to the absorbing tanks for future use, and any ammonia that may be present is taken up with the $H_2SO_4$, to be recovered later by the addition of lime. The great difficulties of precipitating zinc, especially pure zinc, from an acid solution, which allows other impurities to go down with the zinc, we have overcome in the following manner: Dissolve the zinc carbonate or zinc oxide into solution of proper density, then maintain this standard solution by feeding automatically the granular precipitate of basic zinc carbonate fast enough to supply the zinc as precipitated and neutralize the $SO_3$ (sulphur trioxide) as liberated. The fact that our basic zinc carbonate contains no impurities to be precipitated enables us to make a very pure spelter.

We find that the ore from the Afterthought properties, Shasta county, California, roasts and leaches very well. Also from another ore, containing 2.1 per cent. copper, 6.5 per cent. lead, and 10 per cent. zinc, which was sent us from Sinaloa, Mexico, we were able to get an extraction of 73 to 80 per cent. of the zinc, most of the lead remaining in the residue.

I believe the expense of installing and operating a plant for this process will be less than with the ordinary methods of treating even such zinc ores as those suitable for concentration before roasting and smelting in the retort furnace. The cost of the crushing and roasting plant would be the same in each case. We have the advantage in not requiring such a "sweet" roast. Then it is a question of tanks, pumps, filter presses, and steam boiling-out apparatus in our case, as against the small capacity of clay retorts, special clay, and expensive fuel required with the retort furnaces. In our case we have practically all the silver, gold, and copper from the original ore in the residue and only about 50 per cent. of the original ore to be smelted, whereas, with the retort furnaces, there is such a loss of the precious metals by volatilization and the cinder from the retorts is so badly

mixed with ash from the coal used, broken retorts, etc., that it is often too low in value to pay the smelting charge for the recovery of these metals.

In addition, we are also able to treat ores that so far have not proved suitable to other processes, and we have a higher recovery of the zinc from the original ore, viz.: 90 to 95 per cent. from a 35 per cent. zinc ore; 85 to 90 per cent. from a 25 per cent. ore; 80 to 85 per cent. from a 20 per cent. ore, and from 73 to 80 per cent. from a 10.4 per cent. zinc sulphide ore containing some lead and copper. Iron and wooden covered tanks answer the desired purposes, but it is necessary to avoid the use of copper, brass, or zinc in the construction of this plant.

Since we first started to experiment with this process for the recovery of zinc and preparation of the ore for further treatment, we have found that, instead of requiring nearly 24 hr. to leach by agitation, filter, and wash, we are now able to obtain equally as good or better extraction in less than 8 hr. by agitating under pressure, and we use less solution. In this connection it may be well to state that several patents have been applied for in the United States and foreign countries. The cost of operation can be estimated as being the same as preparing the ore for cyaniding, plus the cost of roasting in our case. The cost of leaching is about the same as cyaniding, except that less time is required. The steam for boiling out the ammonia would be extra, but this can be furnished by the waste heat of the reverberatory matting furnace, as they are now generating steam for power purposes at the large smelting plants in Montana and elsewhere. The loss of chemicals is about the same as for cyaniding. In our case it is ammonia, which we find to be about 1 lb. less per ton of ore treated, and lime, 20 to 50 lb. of which are used. The loss of carbon dioxide is not considered, as it is obtained when we make the burnt lime; in fact, what little burnt lime we use is afterwards utilized in the matting furnaces as flux. The pure acid we would use for making metallic zinc, should we decide to do so, instead of marketing the zinc oxide, is used over again continuously, less some loss as sulphate of lime when recovering the fixed ammonia. The cost of smelting the residue in the matting furnace is comparatively cheap, on account of the elimination of the zinc and sulphur; in fact, we have to add a little raw sulphide ore to furnish sufficient sulphur for matting purposes, and we then smelt siliceous gold ore to flux the iron which has been concentrated by the removal of the zinc. By having this roasted residue and fine ore, we are able to adopt the

reverberatory type of matting furnace, using only the cheaper fuel oil for matting purposes and avoiding the use of either coke or coal.

It seems strange to me why those persons who used ammonia and carbon dioxide for the removal of zinc ozide from fused products in the metallurgy of silver, where impurities conflicting with the process must have been encountered, did not adopt this process for the extraction of zinc from ores for the recovery of that metal and for preparing the ore for the recovery of other metals. I mention fused products for the reason that we attempted to leach the zinc from slags containing a high percentage of zinc with no satisfactory results, and also found that ore roasted at an extremely high temperature did not leach well; in fact, we have had to depart in detail from many of the ideas suggested to us by Schnabel, Lunge, and others. The process may have been tried on ore not suitable, or if suitable, the experiments may not have been given proper attention.

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# The Determination of Arsenic and Antimony in Converter and Electrolytic Copper.

BY E. E. BROWNTON, BUTTE, MONT.

(Butte Meeting, August, 1913.)

THIS paper will be confined to the treatment of methods as applied in the laboratory of the Boston and Montana Reduction Department of the Anaconda Copper Mining Co. at Great Falls, Mont.

As the electrolytic plant of this company operates at a current density of approximately 35 to 40 amperes per square foot when running up to capacity, the arsenic and antimony content of the anodes delivered to the plant is of considerable importance as bearing on the purity of the refined cathodes turned out.

Owing to the rather high relative percentage of antimony, it has always been the custom to determine antimony as well as arsenic in all forms of copper. Thus, anodes have been analyzed in which the antimony was about equal to the arsenic although the average shows about 2 parts of arsenic to 1 of antimony. In the refined copper the relative proportions of the two impurities vary considerably, but probably in the average are about equal. In this latter case, the percentage of the two is so small that ordinarily no attempt is made to separate them, the report of analysis showing percentage of As and Sb combined.

Although rapid and exceedingly accurate determinations of arsenic may be made by means of various distillation schemes, the determination of antimony, either by itself or together with arsenic, has always been more or less cumbersome and has required a considerable amount of time. By the methods, details of which follow, the determination of arsenic and antimony, either separated as in the case of anodes or together in case of refined copper, is accomplished in 24 hr. from the time the sample is received by the chemist.

As to the accuracy of the results obtained, it can be quite positively stated that in this respect these methods are very satisfactory, much more so than any heretofore employed in our work, and when the rapidity of determination is considered, the results are very accurate.

*Arsenic and Antimony in Converter Copper.*

Treat 10 g. of the sample in a No. 5 beaker with 30 cc. of sulphuric acid, 20 cc. of nitric acid, and about 50 cc. of water. When solution is complete, heat to boiling to drive off all red fumes, add water to fill beaker about two-thirds and electrolyze about 2.5 hr. at 4 amperes.

The copper is deposited on a large platinum gauze cylinder cathode having the following dimensions: height, 3 in.; diameter, 3 in. The gauze is made of wire 0.0085 in. in diameter and has 36 meshes to the inch. To strengthen the cathode, supporting rings of platinum wire 0.005 in. in diameter are provided at top and bottom. The precipitation is continued until the solution contains about 0.25 g. of copper. It then has a faint blue color. A little experience teaches the proper point at which to remove from the battery. The cathode is rinsed with a little water from the jet when removed from the solution.

The acids are now neutralized with ammonium hydroxide and the solution made acid with hydrochloric acid with 2 or 3 cc. of the acid in excess. Pass a rapid stream of hydrogen sulphide about 30 min. and allow the precipitated sulphides of copper, arsenic, antimony, etc., to settle about 30 min. Filter through a 12.5-cm. paper (C. S. & S. No. 597) in a 3-in. smooth funnel. Discard filtrate. Wash precipitate once on paper with water and then wash sulphides into No. 3 beaker with very smallest possible amount of water. Instead of No. 3 Griffin shaped beaker it is much better to use a No. 5 tall, usual form beaker, as the subsequent evaporation may be carried out much faster without loss by spattering. The beaker containing the sulphides is placed under the funnel and the small amount of sulphides still adhering to the filter is dissolved with 25 or 30 cc. of hot aqua regia poured carefully around the edges of the paper. It is usually well to pour about 10 cc. of this aqua regia into the original No. 5 beaker and transfer from there to the paper, as sometimes a film of antimony forms on the bottom of this beaker In order to prevent too great action of the aqua regia on the filter paper it may be diluted about one-quarter with water. The whole must, however, be very hot to attack sulphides on paper. Wash No. 5 beaker and paper with a small amount of water. At this point it is of great importance to keep the solution as low as possible, consistent with good work, as any unnecessary solution means extra time to evaporate.

Cover the beaker containing the sulphides and aqua regia, boil about 25 or 30 min. to thoroughly break up the sulphides, remove

the cover after washing with a small amount of water from a fine jet and rinse down the sides of the beaker with the fine jet. This should add very little to the bulk of the solution in the beaker. Continue evaporation to complete dryness at a temperature just high enough not to cause loss by spattering. The residue should smell "sweet."

Add about 4 or 5 g. of stick potassium hydroxide and 30 cc. of water and boil vigorously for 10 or 15 min. All arsenic and antimony will go into solution. Add 25 cc. of strong sodium sulphide solution (1 lb. of fresh mono-crystals to 2 liters of water) and again boil vigorously 5 or 10 min. Allow to cool, decant the clear liquid through an 11-cm. paper (C. S. & S. No. 597) into a No. 2 beaker, again add 25 cc. of the strong sodium sulphide solution to the black sulphides left in the beaker, stir well and transfer to filter paper, washing well with a jet of dilute sodium sulphide (1 lb. to 8 liters of water) from wash bottle. If the operator be sufficiently careful to keep the sulphides well wet on the paper, the above washing may be more conveniently accomplished with a jet of hot water. However, if the copper sulphide be allowed to dry, washing with water is liable to carry through some copper. A convenient bulk for the filtrate is from 150 to 175 cc. Add 5 cc. of good hydrogen peroxide and heat until the strong yellow color of the solution fades. Unless too much organic matter has been taken up by the action of too concentrated aqua regia on the filter paper, the solution should become nearly colorless. Cool and electrolyze over night at from 0.10 to 0.15 ampere. Antimony alone is precipitated.

In the morning remove the cathode, carefully washing the adhering solution which contains arsenic into a clean No. 4 beaker with a jet of water, wash in water and two changes of alcohol, dry carefully over an alcohol flame, and weigh for antimony. When the current is interrupted, the antimony coated cathode should be removed, washed, and dried with as much dispatch as possible, as the antimony is quite soluble in the sodium sulphide solution. The cathode employed for this deposition is an ordinary split, foil cylinder, one that has been roughened by long use.

Transfer the solution from the No. 2 beaker to the No. 4 beaker containing washings from the cathode. Make distinctly acid, using dilute sulphuric acid (1 part of $H_2SO_4$ to 4 parts of water). Pass a strong stream of hydrogen sulphide about 10 min. Allow to settle 20 or 25 min., and filter through 11-cm. paper. The arsenic is retained on the paper, finely divided sulphur, only, passing through. It is best to decant the partly clear liquid, only, through the paper. Transfer the heavy arsenic sulphide and sulphur with any accompanying solution directly to a No. 2 beaker. This is done with a jet of

water. Now decant any excess water from the No. 2 beaker through paper and place this No. 2 beaker, which contains the arsenic sulphide, under the funnel. Dissolve the small amount of arsenic sulphide on the paper through into the beaker with dilute ammonium hydroxide (1 part concentrated $NH_4OH$ to 4 parts of water. It is best to wash down the sides of the No. 4 beaker in which the sulphides were precipitated with 10 or 15 cc. of this dilute ammonium hydroxide in order to recover any arsenic that may have adhered to the beaker. Then pour this washing from the No. 4 beaker around the edges of the filter paper, thus carrying any arsenic on the paper down into the bulk of the arsenic in the No. 2 beaker. Wash the No. 4 beaker and the paper with a small amount of water from the jet.

Make the contents of the No. 2 beaker acid with sulphuric acid and add 7 or 8 cc. in excess. Evaporate to sulphuric fumes at a temperature just high enough not to cause loss by spattering and then place on very high heat from 1 to 1.5 hr. Cool, wash the rim and the sides of the beaker down thoroughly with water, add water to half fill the beaker, neutralize the acid with ammonium hydroxide, make slightly but distinctly acid with hydrochloric acid, and filter through an 11-cm. paper into a No. 4 beaker, washing thoroughly with hot water. Add water to about half fill the beaker, neutralize the acid with sodium bicarbonate and add 3 or 4 g. excess. Cool to the temperature of the room, add starch solution, and titrate with standard iodine solution.

Titrating solutions:

Starch Solution.—Make a thick emulsion of starch in 30 cc. of water by hard boiling and dilute to 200 cc. Cool, and use as desired. This solution should be made fresh each day.

Iodine Solution.—Dissolve 10.21 g. of iodine in 8 or 10 cc. of water with 17 g. of potassium iodide and dilute to 2 liters. Each cubic centimeter of iodine solution is equal to about 0.0015 g. of arsenic.

To standardize the iodine solution, dissolve 50 mg. of arsenic trioxide in about 30 cc. of water and 1 g. of potassium hydroxide, make acid with hydrochloric acid, alkaline with sodium bicarbonate, add 3 or 4 g. excess of the sodium bicarbonate, and titrate with the iodine solution after adding starch indicator.

In employing the above method for determining arsenic in converter copper, the arsenic should, in order to work most satisfactorily,

be not lower than 0.04 or 0.05 per cent. When the arsenic is present in lower amounts it is well to use a charge of 20 g., in which case the original solution should be made in a correspondingly larger amount of the sulphuric nitric acid mixture, that is, from 90 to 100 cc. This mixture is made up in the proportion of 200 cc. of nitric to 300 cc. of sulphuric acid.

*Arsenic and Antimony in Refined Copper.*

The lack of a short, satisfactory method for this determination has long been a source of much lost time and of delay to the management of this company. By the following scheme, results which are exceedingly reliable may be turned out in a very short time as compared to methods heretofore employed.

Dissolve 25 g. of the sample and 50 mg. of pure iron in a No. 6 beaker with from 85 to 90 cc. of nitric acid and about 75 cc. of water. When the more violent action ceases, heat gently until the copper is completely dissolved and then boil until all red fumes are expelled. Dilute to about 600 or 700 cc. with very hot water and make ammoniacal. The ammonium hydroxide should be added, with constant stirring, until all copper hydroxide is redissolved. The beaker should be well filled at this point, if necessary by the addition of hot water. Stir thoroughly to collect the ferric hydroxide. Boiling is not allowable, as the copper in ammoniacal solution is too liable to be oxidized. Allow to settle 10 or 15 min., this depending on the time at the disposal of the operator.

Filter through a 12.5-cm. paper, preferably No. 86 Dreverhoff, as this is a very fast paper, admirably adapted for filtering off ferric hydroxide. In this filtration there is a tendency for copper hydroxide to form as the solution becomes cold or is washed with too cold water. A little concentrated ammonia will redissolve this hydroxide at any time it may appear. In the event of any of the ferric hydroxide adhering to the sides of the beaker, it may be conveniently washed down with a jet of dilute hydrochloric acid, 1 part of acid to 1 of water, or weaker. This acid washing is made ammoniacal and poured on paper with the main precipitate. The main part of the blue color should now be washed out. If necessary, a little hot ammoniacal solution, 1 part of ammonium hydroxide to 1 part of water, poured around the edge of the paper, will greatly hasten this operation.

This ferric hydroxide precipitate, which contains all of the arsenic as ferric arsenate and all the antimony, probably as an ammonium antimoniate occluded by the voluminous iron precipitate, is dissolved

through paper with 3 cc. of nitric acid and 5 cc. of sulphuric acid which has been diluted to nearly 50 cc. with water and heated to boiling. Wash well with hot water. It is well to reserve a little of the hot acid solution to pour around the edges of the paper after washing with water. If any yellow color should show in the paper wash again with hot water.

All of the copper that has been held by the ferric hydroxide may now be removed by electrolytic deposition on a small platinum gauze cathode having the following dimensions: height, 2.25 in.; diameter, 1.5 in. This cathode, as employed in our laboratory, is constructed of 52 mesh gauze and has supporting rings of heavy wire at top and bottom. The gauze is made of wire 0.0040 in. in diameter.

The solution, containing the iron, arsenic, antimony, and a small amount of copper, is heated quite or almost to boiling and a current of 0.5 ampere is passed for 2 hr. After about 1.5 hr., the sides of the beaker are washed down with a jet of water. This is to redissolve and remove any copper that may have spattered up from the main solution. At the end of 2 hr., the beaker is removed as quickly as possible, in order to avoid any re-solution of copper. It is unnecessary to wash the anode and cathode, as the small amount of solution adhering thereto will have absolutely no effect on the result, while attempting to save this solution may ruin the determination by copper being dissolved from the cathode.

Now heat the solution, from which all the copper has been removed, to boiling to be sure of complete oxidation, render ammoniacal, boil to aid in settling of precipitate, and when well settled filter through a firm 11-cm. paper. Wash once or twice with hot water to remove nitrates. In washing iron precipitate both in this and in the original filtration, it is well to disturb the precipitate on paper as little as possible.

Dissolve the precipitate through paper with a jet of dilute hydrochloric acid, washing alternately with hot water and dilute acid until no yellow color shows in the paper when it is washed with acid. The dilute acid should be made up of about 1 part of concentrated acid to 4 parts of water. Add ammonia carefully and finally drop by drop until the solution is but very slightly acid. For best results in this neutralization and subsequent reduction the solution should be near boiling temperature.

The reduction may be effected by ammonium bisulphite or, as suggested in Lunge's *Technical Methods of Analysis*, Vol. II., Part I., sodium hypophosphite may be used. In either case no more than is necessary to effect reduction of the iron should be employed. When

ammonium bisulphite is used long boiling is necessary to remove excess sulphur dioxide, hence the necessity of extreme caution to avoid any excess of the bisulphite. However, if the hypophosphite be used no boiling is required and the solution may be transferred at once to the hydrogen sulphide supply. In either case, cool and pass hydrogen sulphide 15 or 20 min. Weigh sulphides the following morning. The sulphides are weighed on an asbestos pad in a platinum Gooch crucible. The pad is prepared of moderate thickness of the very best quality of asbestos. It is washed with alcohol, dried at 105° C., and after cooling in a dessicator for 15 min. the crucible containing the pad is carefully weighed. The sulphides of arsenic and antimony are filtered on to this pad and are very thoroughly washed with water to remove all ferrous and ammonium salts. Wash well with alcohol to remove all water and then with carbon disulphide. After washing with carbon disulphide, using about 20 cc., follow with a thorough washing with alcohol. Dry 1 hr. at 105° C., cool 15 min. in dessicator, and weigh again.

The weight of the sulphides multiplied by 2.4 will give the result directly in percentage of arsenic and antimony, providing 25 g. was taken for assay. This factor has been arrived at after a great deal of experimenting.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.
[SUBJECT TO REVISION.]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Butte meeting, Aug. 18 to 21, 1913, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1913, when Vol. XLVI. of the *Transactions* will go to press. Any discussion offered thereafter should preferably be in the form of a new paper for publication in Vol. XLVII. (with suitable cross references in both volumes).

# Arsenic Trioxide from Flue Dust.

BY JAMES O. ELTON, ANACONDA, MONT.

(Butte Meeting, August, 1913.)

THIS paper covers, besides laboratory work, a study of actual operation at the Washoe Smelter over a considerable period of time, together with the results of a visit to the Midvale plant of the United States Smelting & Refining Co., near Salt Lake. For the latter privilege, as well as permission to publish the resulting data herein given, I am indebted to George W. Heintz, General Manager of the Midvale plant, and here take pleasure in expressing my appreciation for the many courtesies extended to me by Mr. Heintz and members of the technical staff at the smelter.

Almost all of the world's supply of arsenic is recovered as a by-product in the smelting of other metals. It is seen on the market in various forms, the most common being arsenic trioxide, the "white arsenic" of commerce, which is sold as a heavy white powder, or in white, glassy, translucent masses. The trioxide is used in the manufacture of pigment, glass, and insecticides. About 50 per cent. of the domestic consumption is supplied by imports.

*Statistics of Arsenic Trioxide in United States.*[1]

| Year. | Production. Tons. | Consumption. Tons. | Price Per Pound. Cents. |
|---|---|---|---|
| 1907, | 1,010 | 5,471 | 5 |
| 1908, | 1,302 | 6,098 | 3¾ |
| 1909, | 1,007 | 4,600 | 2⅞ |
| 1910, | 1,526 | 5,428 | 2½ |
| 1911, | 3,081 | 6,539 | 2 |
| 1912,[2] | 2,926 | | Closed at 4⅞ |

Arsenic occurs in combination with sulphur and the metals in a large number of minerals. It is present in small amounts in many ores. In the roasting and smelting furnaces many of its compounds break down, liberating arsenic trioxide, which is volatilized and

[1] *Mineral Industry*, vol. xx. (1911.)
[2] *Engineering and Mining Journal*, vol. xcv., p. 96 (Jan. 11, 1913).

carried by the gases into the flue system, where a part is condensed and settles with the flue dust.

Flue dust is the name applied by metallurgists to the solid material caught in smelter flue systems. It includes both dust and fume. Dust consists of fine particles of the charge, or products of the charge, blown or carried by the gas current into the flue system. All smelter charges contain certain substances which volatilize wholly or in part when subjected to high heat in the reducing or oxidizing zones of the furnaces. Upon cooling, these vapors condense, forming minute particles known as fume. Dust can be settled without lowering the temperature, while fume must be condensed before settlement is possible.

The Anaconda metallurgists recognized the fact that in copper smelting the copper values are nearly all contained in the dust, and designed their flue system to catch the maximum amount of dust. Large dust chambers near the furnaces settle the coarse dust. The finer dust, along with the fume, is carried by the gases through departmental flues to the main flue system. As the temperature decreases fume condenses, and a part settles with the dust. The amount settled is dependent upon the decrease in temperature and velocity. The main flue is 2,230 ft. long and discharges into a brick stack 300 ft. high and 30 ft. in inside diameter. The first 1,230 ft. of the main flue is 60 ft. wide and 30 ft. deep, with a center line of hoppers spaced 5 ft. apart, for the removal of dust. The remainder of the flue is 120 ft. wide and is constructed as two parallel 60-ft. flues with a common center wall. There is a line of hoppers under each section. The dust from the first 195 hoppers is sent to the reverberatories and is smelted direct; the last 115 hoppers supply the feed for the arsenic plant.

The general practice has been to save the metal values and allow the fume to escape, unless it contained other valuable constituents, or damage to animal and vegetable life compelled its removal.

The separation of fume from gas may be accomplished by filtering through bags, settling in large chambers hung with wires, or precipitating by electrostatic apparatus. When the fume is settled wholly or in part, it must be disposed of, or the accumulation becomes a burden on the system, if not stopping it entirely. At Anaconda, the flue dust from the last 115 hoppers of the main flue has been treated to recover $As_2O_3$ since 1903. The production of arsenic has been small, and costs have been high. In January, 1913, an investigation was begun with a view to improving recovery and increasing the tonnage treated.

*Arsenic Recovery.*

Arsenic trioxide is recovered from fume or flue dust by roasting or distilling off the volatile $As_2O_3$ and collecting the fumes in condensing chambers. The settled product, called "crude arsenic," is redistilled to further purify it. This refined product often contains 99.97 per cent. of $As_2O_3$. It is pulverized and barreled for market.

*Available Arsenic.*

A study of the arsenic plant records showed very erratic recoveries, varying between 30 and 70 per cent. Experiments were undertaken to locate the trouble and to determine the proper conditions for maximum "burnout."

A coal-fired muffle assay furnace was selected for the roasting experiments. The temperature was measured with a Bristol pyrometer and thermo-couple. Ten-gram samples of the dust contained in 6-in. roasting dishes were roasted at different temperatures, care being taken to maintain as nearly as possible the same heat throughout the roasting.

Two representative samples were selected, one a composite made up from a number of "hopper dusts." The other was a sample taken from Hopper 195. (The arsenic plant feed comes from hoppers 195 to 315.)

When the desired temperature was obtained in the furnace, five samples of each dust were placed in the muffle in such a way that the corresponding number of each sample would be subjected to the same furnace variation. Every 5 min. the temperature was read and recorded. At the end of 10, 20, 30, 40 and 60 min. a sample of each dust was removed from the furnace, cooled, and weighed. Later these residues were analyzed for $As_2O_3$. The amount of $As_2O_3$ volatilized was calculated, and curves were constructed showing the percentage decrease in weight due to $As_2O_3$ and weight burnout ($A^1$ to $A^6$, Figs. 1 and 2). The above procedure was followed with roasts made at temperatures ranging from 800° F. to 1,600° F. Using the above results, a composite curve was also constructed, showing the relative "burnout" of total weight and $As_2O_3$.

To show the effect of a gradually increasing temperature, the samples were placed in the furnace when cooled to 600° F. and the temperature gradually raised to 1,600° F. At the regular intervals, samples were removed as before, cooled, weighed, and assayed. (Roast $A^7$, Fig. 2.) Sulphur and zinc "burnout" curves were constructed, showing the elimination of these elements.

The legend on the plots gives the results obtained.

At the higher temperatures, there was a difference of nearly 200° in different parts of the muffle. This explains the seeming variation in Roasts $A^4$, $A^5$ and $A^6$. In all roasts except $A^1$ and $A^7$, the "burnout" of the volatile $As_2O_3$ was complete at the end of 10 min.

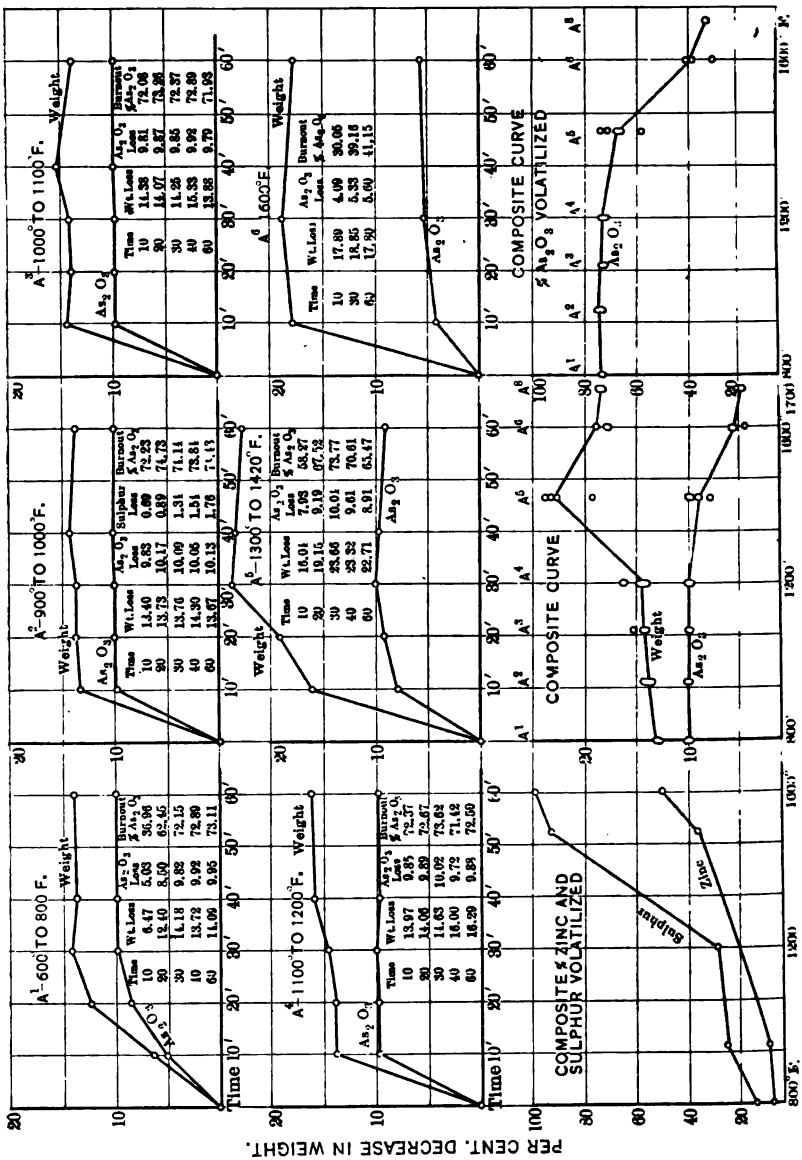


FIG. 1.—MUFFLE ROAST OF FLUE DUST. SHOWING PER CENT. OF TOTAL WEIGHT VOLATILIZED, PER CENT. OF DECREASE IN WEIGHT DUE TO $As_2O_3$ BURNED OUT, WITH TIME AND TEMPERATURE VARIATION, ON COMMERCIAL SAMPLE.

It is readily seen that all roasts below 1,200° F. give practically the same percentage "burnout" of $As_2O_3$. Above 1,200° a lower "burnout" results, owing to combination of the arsenic with the

metallic oxides, forming arsenates. The amount of other substances burned out increases from 1,200° F. to 1,500° F., above which it decreases slightly, due to the burning in of the arsenic instead of out (composite curve). The metallic sulphates decompose, $SO_2$ and $SO_3$

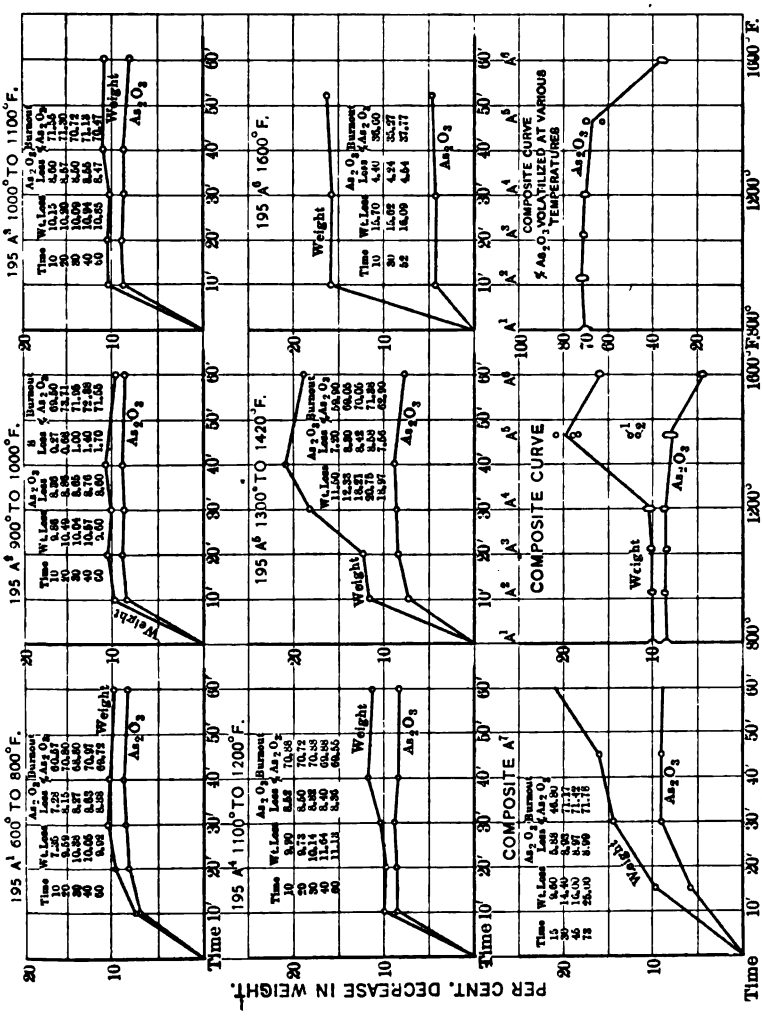


FIG. 2.—MUFFLE ROAST ON FLUE DUST. SHOWING PER CENT. OF TOTAL WEIGHT VOLATILIZED, PER CENT. OF DECREASE IN WEIGHT DUE TO $As_2O_3$ BURNED OUT, WITH TIME AND TEMPERATURE VARIATION, ON SAMPLE 195.

are liberated,[3] and the oxides are partly volatilized as the temperature increases.

[3] Decomposition of Metallic Sulphates, Hofman and Wanjukow, *Trans.*, xliii., 543 (1912).

*Extract from Hofman and Wanjukow's Table.*

| Metallic Sulphates. | Temperature of Beginning of Decomposition. Degrees. | | Temperature of Energetic Decomposition. Degrees | |
|---|---|---|---|---|
| | C. | F. | C. | F. |
| $FeSO_4$, | 167 | 337 | 480 | 896 |
| $Fe_2O_3\ 2\ SO_3$, | 492 | 917 | 560 | 1,040 |
| $PbSO_4$, | 637 | 1,179 | 705 | 1,301 |
| $CuSO_4$, | 653 | 1,207 | 670 | 1,238 |
| $ZnSO_4$, | 702 | 1,296 | 720 | 1,328 |
| $2\ CuOSO_3$, | 702 | 1,296 | 736 | 1,357 |
| $3\ ZnO\ 2\ SO_3$, | 755 | 1,391 | 767 | 1,413 |

Fulton's *Principles of Metallurgy*, p. 223, gives an interesting paragraph on the roasting of arsenic and antimony ores. As it applies more or less to the roasting of flue dust in arsenic furnaces, it is here quoted in full:

"Arsenic or antimony may be present in ore to be roasted, often in the form of sulphides, or as arsenides or antimonides of other metals and sometimes in very complex mineral form. The sulphides of arsenic $As_2S_2$ (realgar) and $As_2S_3$ (orpiment) are readily volatile, and $Sb_2S_3$ (stibnite) is also volatile, but not to the same extent as the arsenic sulphides. Some of the arsenic and antimony is eliminated in the early stages of the roasting by the formation of these sulphides, aided by the distillation of sulphur from the pyrites present. The lower oxides of these elements form $As_2O_3$ which is readily volatile at 218° C., and $Sb_2O_3$ which is volatile only at a low red heat. Considerable of the arsenic and antimony may be eliminated in this way. Further oxidation, however, changes these oxides into the higher oxides, $As_2O_5$ and $Sb_2O_5$, which when present alone are again readily dissociated at a full red heat, but in the presence of certain base metal oxides, such as iron and copper, are converted into very stable arseniates and antimoniates of these metals, which will persist as such in the roasted material. In the process of roasting, the ore particles, as already mentioned, are alternately on the surface of the ore bed subject to an oxidizing atmosphere, and submerged among partially roasted particles from which sulphur is distilling. This sulphur vapor and $SO_2$ gas will reduce arseniates and antimoniates and again cause the volatilization of sulphides of arsenic and antimony, and their lower oxides. The arsenic and antimony are therefore most readily eliminated by alternate oxidation and reduction. A roasting furnace may be specially operated in this way, by methods of firing, when much arsenic or antimony is present."

The last part of the paragraph suggests the possible recovery of

additional $As_2O_3$ by roasting with pyrites. Several roasts were made with varying amounts of pyrites, to see if a better recovery could be obtained. No better recovery resulted than when roasted without pyrites. Then the residues from the high-temperature roasts were mixed with 20 per cent. of pyrites and 100-g. samples of the mixture were roasted at various temperatures up to 1,600° F. The samples were frequently rabbled, the object being to reproduce furnace conditions as nearly as possible. There was no decrease in weight, due to the roasting of the pyrites; but the total amount of arsenic contained in the sample remained undiminished. Powdered coke was substituted for pyrites, with the same results. The conclusion reached was that arsenic in flue dust, when once "burned in" is not again broken up at temperatures below 1,600° F.; also that about 30 per cent. of the arsenic is present as compounds, stable at temperatures below 1,600° F. It must be borne in mind that the amount of available arsenic in flue dust varies. The above applies to Anaconda dust. (Certain samples of Midvale dust gave 90 per cent. of available $As_2O_3$). The best temperature indicated for furnace operation is 950° F. At this temperature the $As_2O_5$ and $As_2S_5$ break down and $As_2O_3$ is volatilized. Only a small amount of $SO_3$ is liberated, and the metallic oxides have a very low vapor pressure at this temperature.

*Gas Saturation.*

While at the University of California, H. V. Welch determined the vapor pressure of arsenic trioxide up to 400° F. (His results are to be published shortly.) Using the same character of apparatus, the vapor pressure of $As_2O_3$ from flue dust was determined at 833° F., the boiling point of sulphur. We are not absolutely certain about the exact location of this last point, due to the use of flue dust instead of pure $As_2O_3$. However, the range of error attained is most probably within the range of accuracy of the curve plotted. Using these two points, a logarithmetic vapor-pressure curve, Fig. 3, was constructed.

The vapor-pressure curve is plotted as a gas-saturation curve, and shows the amount of arsenic trioxide a cubic foot of gas will contain when heated in the presence of sufficient $As_2O_3$ to saturate it at atmospheric pressure. It will always be impossible to get complete saturation in actual practice in the furnace. However, when the hot gas leaves the arsenic furnace, only partly saturated, it will not give up its load of arsenic until the temperature decreases to the saturation point; upon further cooling, the $As_2O_3$ is condensed, the amount

remaining in the vapor state being that indicated by the curve for that temperature. When cooled to 250° F. only a trace remains in the vapor state. This may be taken as the temperature necessary for complete condensation.

This suggests that the nearer the gas approaches to saturation upon leaving the furnace, the smaller will be the amount of gas to be heated and later cooled to condense the $As_2O_3$. In other words, the gas volume is proportional to the fuel bill, after allowing for radiation, heat carried out in the residue, and latent heat of vaporization of the $As_2O_3$.

*Anaconda Arsenic Plant.*

The first refining is done in wood-fired, revolving-hearth reverberatories of the Brunton type. The arsenic plant feed is the dust from the last 100 hoppers under the 120-ft. flue, which in reality is a large dust chamber, in which the velocity is decreased to 11 ft. per sec.

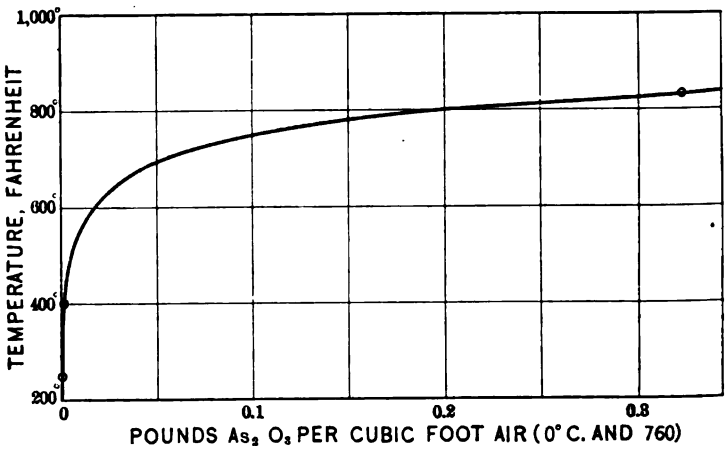


FIG. 3.—ARSENIC TRIOXIDE SATURATION CURVE.

The Brunton furnace is circular, 14 ft. 6 in. in diameter, with two outside fire-boxes. The hearth revolves upon a central shaft. The dust is fed through a dust-sealed hopper on to the center of the hearth, and is slowly moved to the outer edge by three rows of stationary rabbles fastened in the roof of the furnace. There is 3 ft. of clear space between the hearth and the roof. The "residue" is dropped into receiving hoppers and furnishes a regular part of the copper reverberatory feed.

The hot flames from the fire boxes pass directly over the charge, heating it and carrying out the $As_2O_3$ vapor through brick flues to the condensing chamber. The condensing chamber is 8 ft. high, 16 ft.

wide, and 240 ft. long. It is connected with the main flue system to provide draft and to receive any flue loss. The main condensing chamber is divided into small rooms, or kitchens, by cross walls at intervals of 7 ft. These kitchens are connected by openings through the cross walls, so arranged as to give the gas a zigzag course, causing it to come in contact with the cold walls and ceiling, on which the $As_2O_3$ crystals form. There is an opening from the outside into each kitchen for the removal of the crude arsenic. These openings are closed with wooden doors while first refining is going on. At intervals of three or four months the Bruntons are closed down and the kitchens opened; the crude arsenic is shoveled into wheelbarrows and taken to the refining furnace.

*Refining.*

The "refining" furnace is a small coke-fired reverberatory; hearth area, 10 by 20 ft. It has three hoppers, from which the "crude" is dropped into the furnace in 1-ton charges. There is one fire box, in which coke is burned. The hot gas from the burning coke passes over the charge, volatilizing the $As_2O_3$ and carrying it to the refining kitchens. These are similar in all respects to the first-refining kitchens.

When through refining, the kitchens are opened and the product taken in wheelbarrows to the pulverizer, where it is ground and barreled for market. The grinding is done in a closed room with glass windows, through which the operation can be watched. The dust enters the barrels through a leather spout. Many precautions are taken to prevent dust and protect the laborer from arsenic poisoning.

*Midvale Arsenic Plant.*

The Midvale arsenic plant was originally modeled after the Anaconda arsenic plant. The furnaces are of the same type. The kitchens are of the same pattern. Iron doors are used instead of wooden ones, to give additional radiation. A double row of first-refining kitchens allows continuous operation. The feed for the arsenic plant is the dust from the bays of the blast furnace bag house. It averages about 30 per cent. of arsenic and often goes as high as 50 per cent. The dust when fed to the Bruntons had a tendency to fuse and ball up on the hearth and rabbles. This formerly caused a lot of trouble. By controlling the temperature and feeding pyrite with the dust, the tendency to fuse was overcome, and a larger capacity and better recovery is claimed. The pyrite furnishes a large source of heat at the point where it is most needed. It keeps the charge open and prevents flowing and sintering on the hearth. The sulphur vapor and

$SO_2$ may reduce a portion of the arsenic present as arsenate; although this is not indicated by our laboratory experiments.

The first kitchen of each line is equipped with a Bristol recording pyrometer. The temperature is never allowed to go above 650° F. This means about 950° in the furnace (the ideal temperature for maximum recovery). A bag system prevents flue loss. Draft is supplied by an electrically driven 70-in. Sturtevant blower. The draft is equivalent to $\frac{5}{16}$ in. of water, measured at the fan. The pressure on the bags is equivalent to 2 in. of water, also measured at the fan.

The second-refining furnace, like the Anaconda furnace, originally had three charging hoppers. The third hopper from the fire box was removed and the opening closed. This shortened the time necessary to volatilize a charge of "crude." (The third hopper charged crude into the cold end of the furnace.) Powdered lime, about 1 in. thick, is spread on the hearth before starting refining. This prevents the crude from sticking to the bottoms and facilitates the removal of sinter. The line of kitchens was too short to effect the necessary cooling, so cold air was let in through openings covered with burlap. The burlap prevents outside dust from being drawn into the kitchens. The admission of cold air caused the formation of fume, which would not settle in the kitchens. Bags were installed to prevent loss. The bags are shaken every 2 or 3 hr. When the kitchens are filled with refined arsenic the furnace is shut down. The product is shoveled into buggies and transported by overhead tram to the pulverizing room. Pulverizing and barreling are done the same as at Anaconda.

Special one-piece outer garments are provided and kept clean by the company, A small bonus is paid for work in the arsenic plant.

Experiments were undertaken to determine the per cent. of arsenic saturation of the gas and to furnish the data for a heat balance.

*Midvale Heat Balance on First Refining.*

| | Pounds. | Temperature C. | | Latent Heat. | Sp. Heat. | Pound-Calories. | Pound-Calories Absorbed. | Pound-Calories Evolved. | Per Cent. Heat Absorbed. | Per Cent. Heat Evolved. |
|---|---|---|---|---|---|---|---|---|---|---|
| | | In. | Out. | | | | | | | |
| Dust | 2,000 | | | | | | | | | |
| Pyrite | 338.4 | | | | | | | | | |
| Total $As_2O_3$ | 392.9 | | | | | | | | | |
| Coke | 195.6 | | | | | 6,889 | | 1,347,488 | | 5.[illegible] |
| Sulphur | 90.2 | | | | | 2.64 | | 195,193 | | 12.[illegible] |
| Residue | 1,914.2 | 20 | 434 | | 0.22 | | 178,557 | | 11.0 | |
| $As_2O_3$ | 84.9 | | | 70 | | | 5,943 | | 0.4 | |
| Gas | 33,546.2 | 20 | 149 | | 0.2375 | | 1,027,772 | | 66.6 | |
| Radiation *a* | | | | | | | 330,409 | | 22.0 | |

*a* By difference.

This calculation is based on the amounts of coke and sulphur burned per ton of dust treated as shown by the 1912 arsenic plant balance. The volume of gas was measured at the fan, where the temperature was 149° C. The $As_2O_3$ indicated in the table is the amount that would be in the vapor state in the gas volume at 149° C. and would still retain its latent heat of vaporization. The weight of residue is the residue from treating 2,000 lb. of dust and 338.4 lb. of pyrites. The sulphur represents that burned from the pyrites, and is assumed to be burned to $SO_2$. As the "sulphur burned" is about one-half of the total sulphur present, it is assumed to represent the loosely bonded atom of the $FeS_2$ and no allowance is made for the heat of decomposition.

The measured temperature in the furnace was 510° C. Since the volume of gas cooled from 510° to 148° C. and the radiation was only 22 per cent., it is obvious that there was a large amount of cold air mixed with gas to help cool it. 195.6 lb. of coke burned to $CO_2$ will only heat 10,996 lb. of air to 510°, allowing 5 per cent. for furnace radiation. (The heat from the burning sulphur counterbalances the heat carried out in the residue.) The 10,996 lb. of gas was then used to carry out the 392.9 lb. of arsenic trioxide. The gas at 510° C. was less than 1 per cent. saturated (see gas-saturation curve).

The above calculation points to a furnace giving the best possible contact between the hot gas and the flue dust, to give better saturation.

## *Conclusion.*

The experiments, calculations and observations suggest certain points, which if embodied in arsenic plant construction and operation should greatly improve recovery and reduce costs.

The temperature in first refining should be about 950°, and under no consideration should be allowed to exceed 1,200° F.

The gas must be cooled to 250° F. before the $As_2O_3$ is completely condensed.

## *Furnaces.*

Nearly all furnaces will give a good recovery if properly operated; but to reduce costs as well, the furnace should have as many of the following points as possible: A large capacity; easy heat and draft control; good transference of heat to the charge; large hearth area, with means for constantly shifting surface of the dust; intimate contact of the hot gas with the dust, preferably a constant falling of the hot dust through a slow-moving current of the hot gas; continuous operation, with sealed feed and residue-discharge openings; gradually increasing temperature of the dust towards the residue-discharge end; a decreas-

ing gas temperature to the saturation point at the gas-discharge end; simple operation; and low cost of upkeep.

The length of time the dust should remain in the furnace depends upon the thickness of the bed of dust and the rabbling it receives. If given an opportunity to volatilize, the $As_2O_3$ will be eliminated in from 30 to 45 min.

Rotating cylinder furnaces of the Oxland or White Howell type seem to fulfill most of the above points. The tendency to dust could be overcome by employing the scheme suggested by Rothwell;[4] i. e., to construct a dust chamber at the end of the furnace, so that the dust would slide back into the furnace.

Any of the multiple-hearth furnaces could be remodeled to make good first-refining furnaces. Cast-iron hearths could be used to give better transference of heat. At 950° F. the rabble arms and castings will stand up without cooling.

### *Cooling or Precipitation Chambers.*

There are two general methods, either of which has advantages for the separation of $As_2O_3$ from the furnace gas.

1. *Cooling Followed by Bags.*[5]—The gas from the furnace is mixed in a cooling chamber with cold air, to lower its temperature to 250° F. It is then filtered through bags to separate the resulting fume. The product in this case is a fine dust, and can be drawn from hoppers. No expensive flue system is required; the filtered gas can be discharged in the building.

Constant attention is required to look after the motor, shake the bags and regulate the cold-air supply. Coke must be used in the furnaces, as soot and moisture from burning wood or coal spoil the bags and contaminate the arsenic.

2. *Precipitation by Decreasing the Saturation Point of the Gas by Radiation.*—When using the precipitation method, the gas volume should be as small as possible. The hot gas is conducted into kitchens of the Anaconda type; the walls should be of good conducting material, and gas-tight, to prevent infiltering air. The number of kitchens must be sufficient to give proper cooling in summer time, to prevent excessive loss. Where this method is used, there is always a small flue loss due to the formation of fume. The gas, upon coming in contact with the cold surface, is chilled, the saturation point of the gas is decreased, and the $As_2O_3$ precipitates in crystalline form on the cold surfaces.

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[4] *Mineral Industry*, vol. i., p. 234 (1892).

[5] Midvale practice.

An extra set of kitchens is necessary to permit continuous operation, because the chambers must be entered to remove the deposit from the walls.

*Second Refining.*

The reverberatory refining furnace as operated at Midvale gives excellent results. Better transference of heat could be effected by making the hearth of cast iron and having the hot gas enter and travel the length of the furnace underneath the hearth, returning over the top of the charge to carry out the $As_2O_3$ vapor.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.
[SUBJECT TO REVISION.]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Butte meeting, Aug. 18 to 21, 1913, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1913, when Vol. XLVI. of the *Transactions* will go to press. Any discussion offered thereafter should preferably be in the form of a new paper for publication in Vol. XLVII. (with suitable cross references in both volumes).

# Timbering in the Butte Mines.

BY B. H. DUNSHEE, BUTTE, MONT.

(Butte Meeting, August, 1913.)

THIS paper is not intended to be a technical discussion of square-set framing as used in mines, but merely a short description of the different kinds of framing that have been used in the Butte mines, and what has been decided upon, after years of practical experience, as being the best, taking into consideration the physical condition of the veins and country rock, and also the supply of timber available for mining purposes.

Butte is, essentially, a square-set district. Virginia City, Nev., was the first mining camp to use this method of timbering, and to Philip Deidesheimer, who went to the Ophir mine in 1860, is given the credit of solving the problem of square-set framing, where large bodies of ore were stoped and the timbers were supposed to hold the ground from caving without any filling. That he did his work well is shown by the fact that the main features of his framing are in use to-day.

In the Butte mines the conditions are such that this method of timbering is generally used. We do not, of course, depend upon the timbers any more than is necessary. The worked-out stopes are filled with waste rock as close behind the miners as possible and not interfere with mining operations. The waste rock is obtained from development work on the different levels or from the surface. In the earlier days of mining in Butte there were numerous independent companies and naturally there were different ideas as to the best way to frame the square sets, although the general features were the same.

Timber was abundant for mining purposes and the common practice was to use sawed lumber, generally 10 by 10 in. or 12 by 12 in. square. Some of the mines used for girts 6 by 10 in. or 8 by 12 in. timber, depending upon whether they used 10-in. square or 12-in. square timber for posts and caps. This avoided the necessity of framing the ends of the girts, and also required less timber for the

same work. It proved very satisfactory, as the cap, which is the stronger of the horizontal members of the square set, is placed perpendicular to the strike of the ore body and resists the pressure from the hanging wall of the vein; while the main function of the girt is to resist the side swinging of the caps as the weight from the walls comes on them.

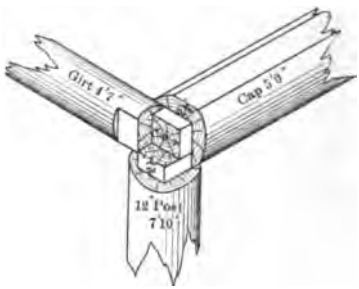
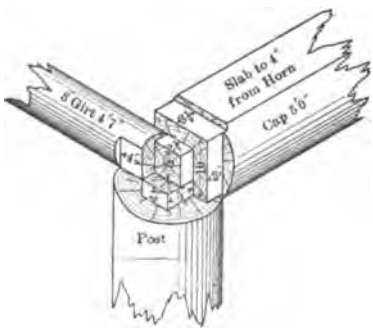



Fig. 1.—Stope Set.



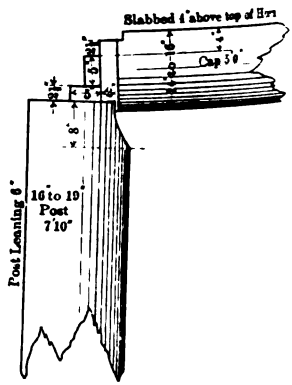




Fig. 2.—Sill Set.

FIGS. 1 AND 2.—TIMBER SETS AT GAGNON MINE.

The Gagnon mine was an exception in that round timbers were used for square sets long before it became a practice in the other mines. About 1886, while the management of the mine was under C. W. Goodale, a machine was installed to frame round timbers. With slight alterations, the same framing is in use to-day at this mine, as the Gagnon still has an independent framing mill. Fig. 1 gives a perspective of the kind of framing used at this mine at the present time. The post has a horn 5 in. square and 2½ in. high on each end. The cap is similarly framed, but in addition has a shoulder taken off the bottom, 5 in. from the center, to allow the cap to fit snugly on top of the post, and on top, also 5 in. from the center,

a slab is taken off the full length to allow for a level floor in the stopes. The girt is generally less than 10 in. in diameter and is framed on two sides only. The posts are 7 ft. 10 in. in length, making the sets 8 ft. 3 in. from center to center. The caps are 5 ft. in length and butt end to end. The girts are 4 ft. 7 in. long. This makes the sets 5 ft. from center to center, either cap-way or girt-way. On the main working levels of the mine practically the same framing is used, with some slight changes which can readily be seen in Fig. 2. The timbers are especially selected and much larger, and the posts,

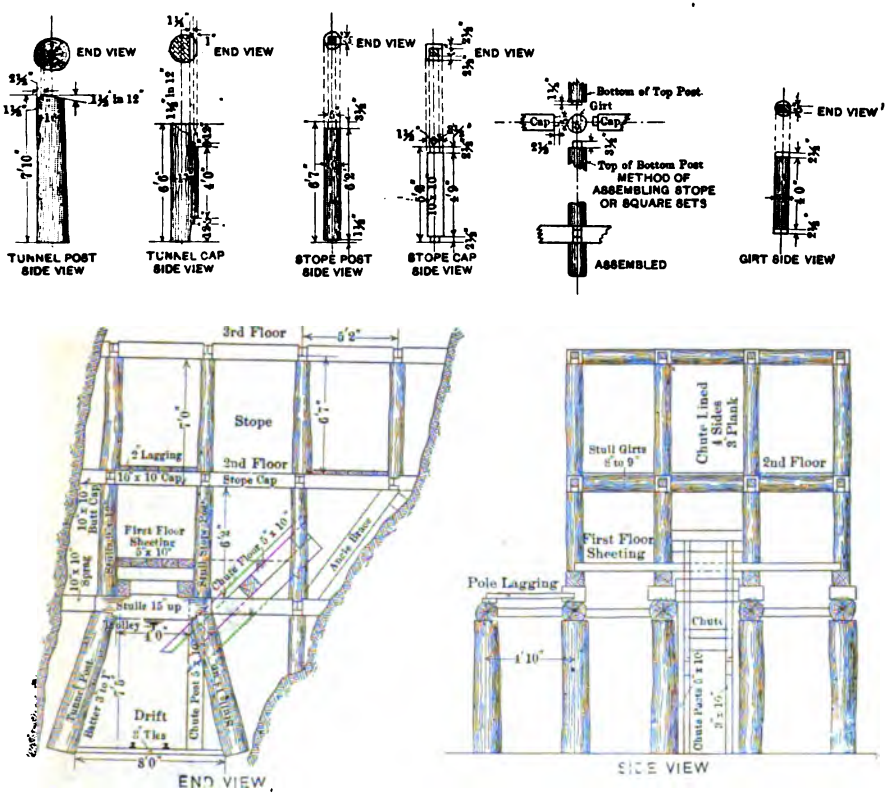



FIG. 3.—TIMBERING METHODS AT STEWARD MINE.

which are flat bottomed, are 1 ft. farther apart on the floor of the sill than they are at the cap. This gives additional strength to resist the side pressure which tends to push the posts into the drift, thus adding materially to the life of the drift before repairs are necessary.

The Steward mine also has an independent framing mill and continues the same method of framing that has been in use for a number of years. (See Fig. 3.) The posts have a horn 5 in. square by 3½ in. at

the top, while the horn at the bottom of the post is 5 in. square by 1½ in. It was supposed to be easier and quicker to stand a post in the stopes with a short horn than with a long one. It necessitates the horn of the cap being out of center, which does not weaken the cap, but, generally speaking, the more symmetrical the timbers, the easier they are to put in place in the stopes. The posts and girts are round timbers, while the caps are 10 by 10 in. square. The posts are 6 ft. 7 in. in length, making the sets 7 ft. from center to center. The caps are 5 ft. 2 in. and butt end to end. The girts are 4 ft. 5 in., making the sets 4 ft. 10 in. center to center girt-way. The timber for the support of the ground in the main working levels is especially selected from the best stulls, varying in size from 14 to 23 in. in diameter. The framing is simple and inexpensive. The posts are framed on the top end only, being sized back 2½ in. and down 1½ in. to form a shoulder for the cap. Back of this shoulder both the cap and post are cut on a batter of 1½ in. to the foot, giving the post a pitch of 3 in. to 1 ft. toward the foot or hanging wall of the vein. This makes the distance between the posts on the floor of the drift 8 ft. While it might appear that the batter given the posts is too great, experience has shown such not to be the case in this mine.

The end view, Fig. 3, also shows the method of putting in the sheeting above which the stope is filled with waste rock as soon as possible after the ore has been removed. This sheeting is placed at a sufficient height above the cap of the drift set to allow for repairs when necessary. The general practice is to use small stulls for sheeting, or larger ones sawed in half. There is also shown the position of short, flat-bottomed posts and angle braces; also the chute which is used for ore or waste, and is carried up with the stopes.

Figs. 4 to 8 show the different framing in use at several of the mines before the advent of round timber.

Fig. 4 shows the framing at the High Ore mine. The post has a long and rather small horn, which is very liable to break with heavy side pressure, and with heavy pressure from the top the horn tends to crush.

With the Syndicate group (see Fig. 5) we have, perhaps, the other extreme. The horn of the post is only 1 in. in height, giving a small shoulder for both the cap and girt; besides the framing of the girt is unnecessarily complicated, which would, of course, increase the cost of timbering.

Fig. 6 shows the framing at the Anaconda group. This is, I think, the best method of framing sawed timber for square sets. It is simple, cheap, and retains the full strength of the timbers in whatever direc-

ion the pressure may come. The horns on the posts are 6 in. square by 2 in. They are strong and give a good shoulder for the cap and girt. The caps butt end to end with a horn 6 in. square by 3 in., while the girts are framed 6 by 10 in., with a 2-in. shoulder to fit. The girts are unnecessarily large compared with the other timbers in the set, and the framing, as shown in Figs. 7 and 8, which is the same as Fig. 6 except that the girts are 4 in. less one way and require no framing at all—is just as good and less expensive.

Fig. 7 shows the framing for 10-in. square timber. The girt is 6 by 10 in., and 4 ft. 6 in. long. The sets are 5 ft. from center to center, either cap-way or girt-way, and are 7 ft. 6 in. from center to center in height.

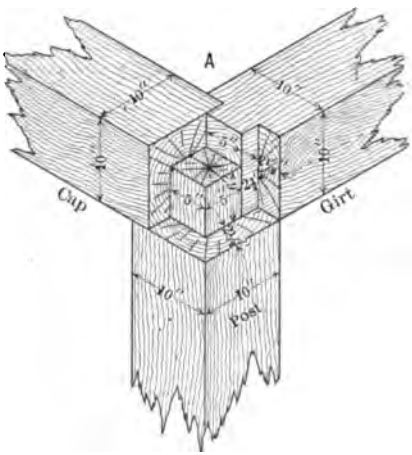



Fig. 4.

Fig. 8 shows framing for 12-in. square timber, where the girts are 8 by 12 in. This size of timber was not used except in stopes where the ground was unusually heavy.

Timber suitable for the saw mill and available for the Butte mines became rapidly less, owing to the great and ever increasing demand from the mines. There were extensive forests of pine and fir trees growing in the mountains surrounding Butte which were large enough for mine timbering if round timbers could be used for square sets; besides, the expense of timbering, which is a large item in mining costs, would be materially decreased.

The first machine that was installed in the Rocker framing mill, which is located at Rocker, about 3 miles from Butte, where most of the timber for the Butte mines is framed, was made to cut a strictly

mitered set. The post had a flat top 8 in. square, with no horn, and a miter cut to the outside of the timber. The cap had a 5-in. square horn, 4 in. on the top and bottom and 2½ in. on the sides to the beginning of the miter. The girt had a horn which was 1½ in. on top and bottom, while on the sides the miter extended to the end of the girt.

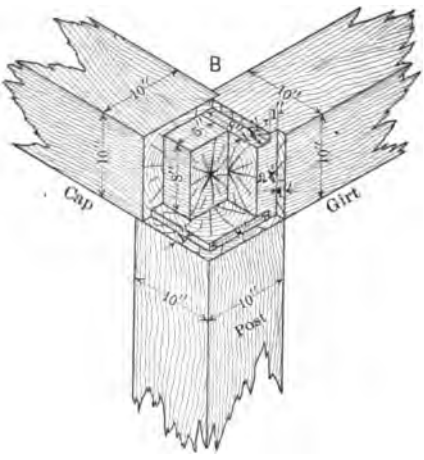



Fig. 5.

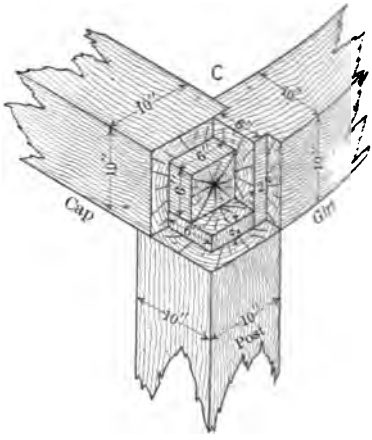



Fig. 6.

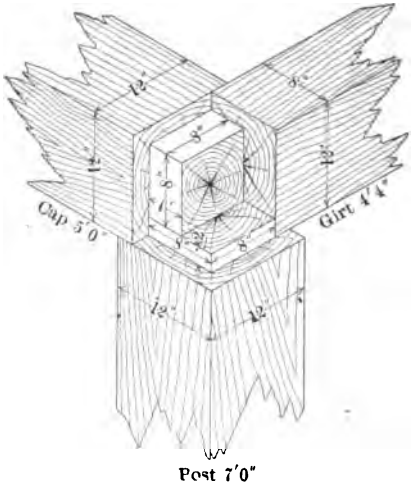



Fig. 8.

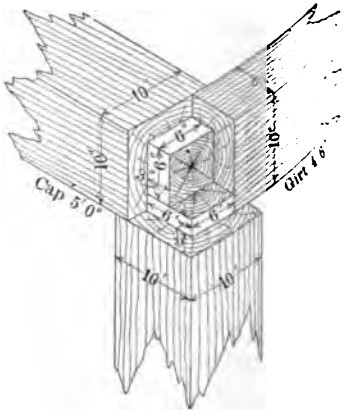



Fig. 7.

FIGS. 4 TO 8.—FRAMING METHODS BEFORE ADVENT OF ROUND TIMBERS.

This style of framing was used for some time in many of the mines, but it was not easy to set the timbers in place and it was found difficult to block the timbers so that the sets would resist the pressure of the ground satisfactorily. Also the concussion caused by blasting gave trouble.

An improvement was made in what was known as the combination square and bevel framing, shown in Fig. 10. The posts have a short horn 8 in. square by 1½ in., then the miter cut to the outside of the timber. The cap framing is somewhat complicated—the horn for a length of 1½ in. is 8 by 5 in., then for 2½ in. it is 5 in. square. At the base of the horn is a 1½-in. shoulder from which the miter begins.

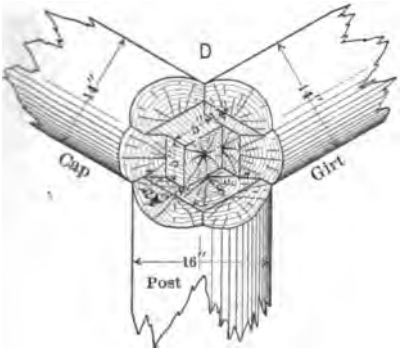



Fig. 9.

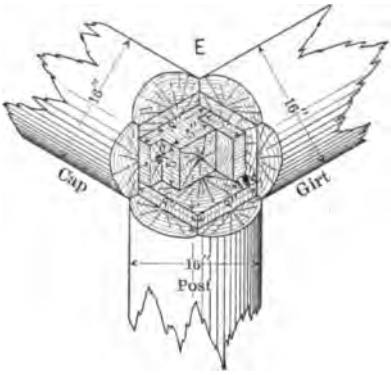



Fig 10.

FIGS. 9 AND 10.—COMBINATIONS: SQUARE AND BEVEL, ROUND TIMBERS.

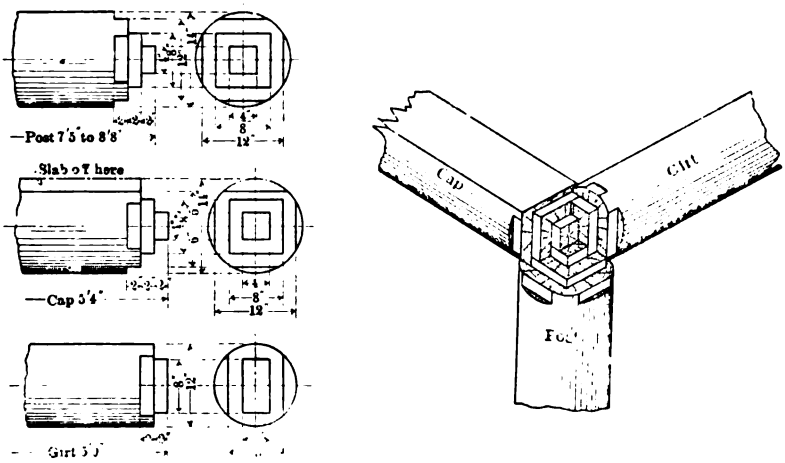



FIG. 11.—STEP-DOWN TIMBER FRAMING.

The girt has a horn 5 in. square by 1½ in., a shoulder of 1½ in., then it is mitered the same as the other members of the set. The posts are 7 ft. 5 in., making the sets 7 ft. 10 in. center to center. The caps are 5 ft. 10 in. and butt end to end. The girts are 5 ft. 5 in. These lengths make the sets 5 ft. 10 in. either cap way or girt-way. These dimensions were the same in the mitered framing.

This framing, although an improvement, was not altogether satisfactory and a further improvement was made by eliminating the miter entirely, and having nothing but square shoulders, locally known as step-down timber framing. (See Fig. 11.) The post starts with a horn 4 in. square, then a step down of 2 in., and another horn 8 in. square, another step down, and, if the post is large enough, another horn 12 in. square. The cap is framed exactly like the post, and in addition one side is slabbed off 5 in. from the center of the stick to allow for a level floor for the stopes. The girt starts with a horn 4 by 8 in., then a step down and a horn 8 by 12 in., but the girt is the smallest member of the set and seldom reaches 10 in. in diameter. The size of the square set was changed as well as the style of the framing. The posts are 7 ft. 5 in., making the sets 7 ft. 9 in. from center to center. The caps are 5 ft. 4 in. and butt end to end. The girts are 5 ft. These lengths make the sets 5 ft. 4 in. from center to center either cap-way or girt-way. This change was made for two reasons: First, 5 ft. 4 in. square is large enough for most of our stopes, as the ground is frequently heavy; second, the common length for pole lagging, stulls and 2-in. plank, which is sometimes used for lagging as well as for flooring in the stopes, is 16 ft. and three pieces 5 ft. 4 in. long can be cut from a 16-ft. stick without waste.

This is, I think, the best framing for round timbers of various sizes. We obtain, as nearly as possible, the full strength of each member of the set regardless of the size of the stull. If the stulls were all of the same diameter we could have a more simple framing, somewhat similar to that shown in Figs. 6 and 7 for square timber. For the main working levels the timbers are especially selected, generally from 12 to 18 in. in diameter for posts and caps, the girts being smaller. The same framing is used, except the posts are framed on one end only. It is the intention ultimately to use this method of framing in all of the Anaconda Copper Mining Co.'s mines.

The framer used, shown in Figs. 12 and 13, is a new type made by Greenlee Bros. & Co., Rockford, Ill., from specifications and drawings prepared by the Anaconda Copper Mining Co. The first machine has been in operation only a few months, while the second one has just been installed. The framing is done by saws with insert teeth mounted to form a cutter head on a horizontal arbor. The round timbers are stepped down by a series of 2-in. shoulders. The cutter heads are made up to form a series of faces 2 in. wide with saws stepping down 4 in. in diameter each time. Each of these faces of the cutter head is made up of three saws equipped with insert teeth 0.75 in. wide, staggered 0.5 in. ahead of one another, and with the teeth over-

lapped so that they cut a face on the timbers just 2 in. wide. There are two arbors carrying the cutter heads at each end of the machine, one above the other. As the stull is fed through the machine it is completely framed on the top and bottom sides. The stick is turned

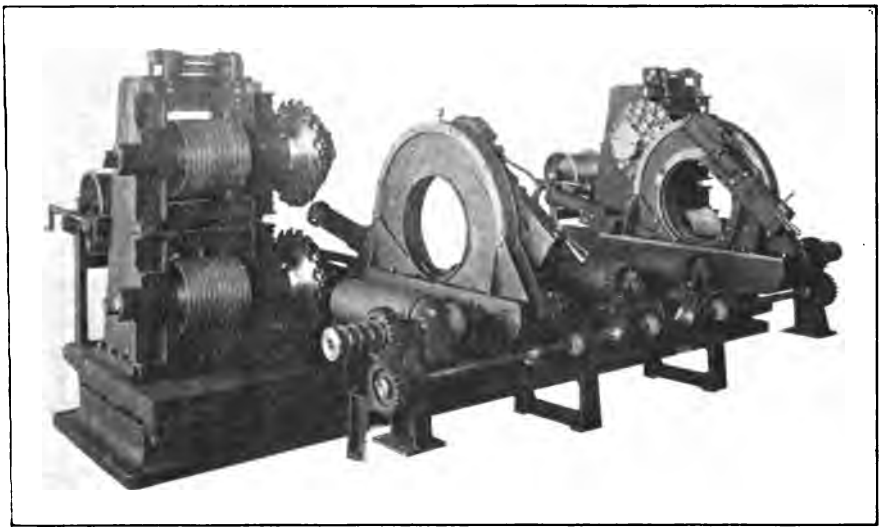


Fig. 12.

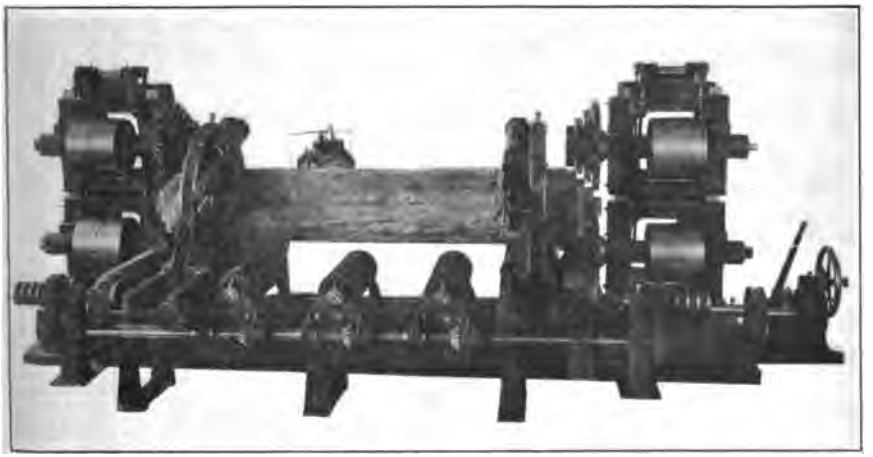


Fig. 13.

FIGS. 12 AND 13.—MACHINE FOR FRAMING MINE TIMBERS.

90° and again fed through the machine to complete the framing. Over the saws are hoods connected with suction fans which take the saw-dust and shavings to the fire boxes of the boilers.

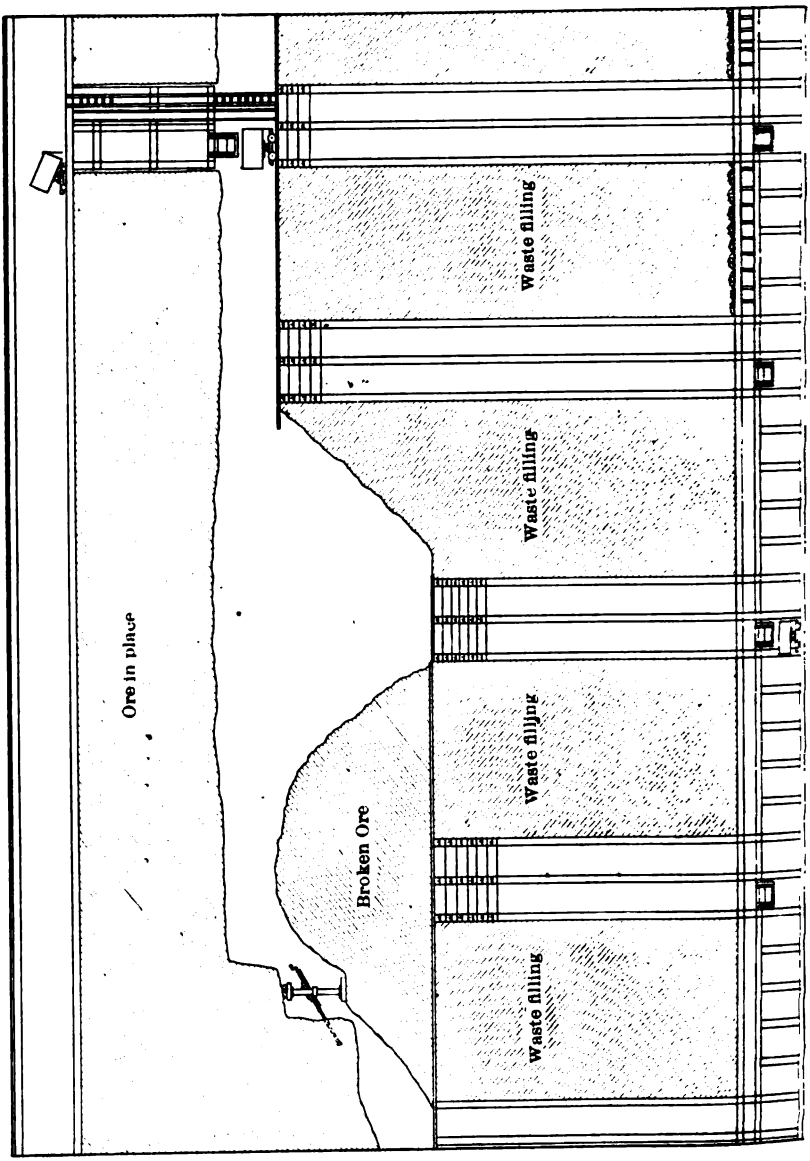



FIG. 14.—BACK FILLING SYSTEM OF MINING

It might be interesting to describe a system of mining known locally as the "back-filling system," although it does not come under the head of timbering, but rather of mining without timbers. It is made use of whenever the physical conditions of the vein and wall rock are such that it is safe to do so. Both the ore and country rock must be hard and free from faulting planes. Fig. 14 gives a general view of this system of mining.

After a sill has been driven on the vein, a raise is put through to the level above. The ore on the first floor is then mined, and chutes started about every 25 ft. along the drift. Between the chutes, sheeting, composed of small stulls cut in half and resting on timbers a few feet above the drift sets, is placed from the foot to the hanging wall of the vein.

Starting from the raise the miners begin to break the ore. After the ore has been broken from foot to hanging for a short distance, the miners return, set up their machine on the broken ore, and again work their way forward. This is done two or three times until the broken ore is 15 or 20 ft. high, then the miners start again farther ahead and the shovelers begin to shovel the broken ore, first into the raise chute, until they reach the next ore chute. As soon as they have passed the ore chute, a temporary opening is made in the raise chute and waste is dumped into the chute from the level above and allowed to run on to the sheeting as long as it will; then a track is started about 6 ft. from the top of the stope, the waste is taken from the chute in a car and the filling of the stope continued to the first ore chute. The shovelers being now out of the way, the timbermen build up the ore chute one compartment wide and two in length, using small round stulls, 6 to 8 in. in diameter, to the level of the waste track. The waste-men then continue filling the stope, following the shovelers.

When the miners have gone as far as the stope is to be worked, they return and start another floor on top of the waste filling, which has been covered with a floor of 2-in. plank to keep the ore clean.

If at any time the ground changes and gets heavy, square setting can be started on the waste filling, and this is generally done when the stope has been worked out to within a floor or two of the level above.

For particulars regarding square-set timbering reference is here made to a series of articles by Claude T. Rice.[1] These articles were largely made up from data furnished by the author of this paper.

[1] *Mining and Engineering World*, vol. xxxvii, pp. 1079, 1133, 1175; vol. xxxviii, pp. 3, 243, 295, 379, and 429 (Dec. 14, 1912, to Mar. 1, 1913).

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.
[SUBJECT TO REVISION.]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Butte meeting, Aug. 18 to 21, 1913, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1913, when Vol. XLVI, of the *Transactions* will go to press. Any discussion offered thereafter should preferably be in the form of a new paper for publication in Vol. XLVII. (with suitable cross references in both volumes).

# Ore Deposits at Butte, Mont.

BY RENO H. SALES, BUTTE, MONT.

(Butte Meeting, August, 1913.)

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## INTRODUCTION.

THE geology of Butte possesses especial interest on account of the magnitude of the ore deposits, their extraordinary richness and persistence in depth. Since its discovery in the early 60's the district has yielded, in round numbers, 6,000,000,000 lb. of copper, 260,000,000 oz. of silver, 1,250,000 oz. of gold, and a large but not definitely known number of pounds of zinc. This enormous metal production has been derived from an ore tonnage estimated roughly at 65,000,000 tons. At the present day, most of the important mine shafts have reached a depth of 2,400 ft., while several are more than 3,000 ft. deep. At these great depths rich bodies of copper ore are being exploited, showing no diminution in metal content as compared with the ores of the higher levels.

The present (1913) daily rate of production is approximately 14,000 tons, of which 13,250 tons may be regarded as copper ore, although a small percentage of this is minable only because of the presence of

notable amounts of silver, and the remaining 750 tons are zinc ores. In addition, a small tonnage of straight silver ores is mined, chiefly by leasers, from abandoned silver mines. A small amount of copper is recovered in the precipitation plants treating the copper-bearing waters pumped from the mines.

In spite of their great importance as metal producers, the subject of the geology of these ore deposits received but scant attention at the hands of investigators until recent years. The Butte Special Folio,[1] the work of S. F. Emmons, W. H. Weed, and G. W. Tower, published by the U. S. Geological Survey in 1897, was the first publication to deal with the Butte district in detail. This was followed by a more comprehensive study undertaken by W. H. Weed[2] during the period extending from 1901 to 1905, the results of his work appearing in 1912. Other articles treating special features of the ore deposits have appeared from time to time. H. V. Winchell's paper[3] dealing with the artificial production of chalcocite or copper glance was timely and of great interest in connection with the problem of chalcocite formation. More recently Charles T. Kirk[4] has carefully studied the various phases of granite alteration found in association with the copper veins. He endeavored to determine if any genetic relation existed between these alteration phases and the deposition of certain copper minerals. He concludes from his investigations that the chalcocite of the Butte veins is almost entirely a product of descending sulphide enrichment, a view expressed also by W. H. Weed.

Weed's study was concluded in 1905 and since that time much development work has been done. New mines have been opened and the older ones have been greatly extended both vertically and laterally, bringing to light many new and interesting features concerning the vein and fault structure. The persistence of rich ore bodies to great depth has aroused a keen interest among geologists regarding the manner of formation of the abundant copper mineral chalcocite. Much information is available bearing directly upon this important question. In the presentation of this new material, together with such associated facts as are considered to be necessary and of general interest concerning the geology of these ore deposits, the writer hopes to find justification for the publication of this paper.

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[1] *Butte Special Folio* (No. 38), *Geologic Atlas, U. S. Geological Survey* (1897).

[2] Weed, W. H., *Professional Paper No. 74, U. S. Geological Survey* (1912).

[3] Winchell, H. V., Synthesis of Chalcocite and Its Genesis at Butte, Montana. *Bulletin of the Geological Society of America*, vol. xiv., pp. 272 to 275 (1902).

[4] Kirk, C. T., Conditions of Mineralization in the Copper Veins at Butte, Montana, *Economic Geology*, vol. vii., No. 1, pp. 35 to 82 (Jan., 1912).

## General Geology.

Butte is situated near the western border of a large granite area called the "Boulder" batholith, extending southwesterly from Helena to the Big Hole river in Beaverhead county, a distance of 70 miles. The general outline of the batholith is oblong, but it is extremely irregular in width, averaging about 20 miles. Many smaller isolated masses of similar rock occur along its western and northern borders. The granites of Marysville, Philipsburg, Clinton, Garnet, and several other mining districts are undoubtedly offshoots from a main parent magna. It is not improbable that further study will show a close relation between the Boulder granite area and the great granite batholith of central Idaho.

The Boulder batholith made its appearance subsequent to the great Rocky Mountain building period, marking the close of Cretaceous times. It is more recent than the large mass of andesite forming Bull mountain, near Whitehall, which is known to intrude upturned Palæozoic and Cretaceous rocks. There is an abundance of evidence to show that the Boulder batholith did not produce a doming effect on the sedimentary rocks now found along its borders; in fact, the orientation of these intruded rocks does not appear to have been visibly disturbed even where they are in direct contact with the granite. In nearly every instance where such contacts are open to observation, the sedimentaries are found to dip at a steep angle toward the granite; an exception, however, may be noted at Elkhorn, where dips away from the granite appear to be a coincidence and not an effect produced by its intrusion. In a broad way the batholith seems to occupy the trough of a great synclinal basin in whose dissected sloping sides may be seen remnants of the entire series of sedimentary rocks reaching from the pre-Cambrian slates and shales upward to the coal-bearing sandstones of the late Cretaceous. The manner in which this great displaced mass of sedimentaries made its escape is not known.

While the chemical composition of the granite is fairly uniform over the whole area there is a slight variation in physical character or texture, indicating the possibility of two or more distinct intrusions. The texture is usually granitic, in fact, typically so, but at times it is decidedly porphyritic, showing numerous imperfectly formed orthoclase feldspars an inch or more in length. Basic segregations composed of dark silicates are present in considerable amounts as a noticeable feature of the marginal areas of the batholith. These small dark masses, which are from 1 to 6 in. in diameter, are possibly more prevalent where the granite has the even texture and the porphyritic character is lacking. They are more resistant to weathering than the body

of the granite, so that they often stand out in relief over the surfaces of the big rounded boulders, giving a "warty" appearance to the rock.

These variations in texture may be seen in the mines of the Butte district, but there is little, if any, direct evidence to support the hypothesis of two or more separate and distinct intrusions. It is more probable that these variations in texture were developed during the process of cooling and were due to uneven temperature conditions within one great granite area.

An analysis of the typical Boulder granite is given by Weed as follows:

| | |
|---|---|
| $SiO_2$ | 67.12 |
| $Al_2O_3$ | 15.00 |
| $Fe_2O_3$ | 1.62 |
| $FeO$ | 2.23 |
| $CaO$ | 3.43 |
| $MgO$ | 1.74 |
| $K_2O$ | 4.52 |
| $Na_2O$ | 2.76 |
| $TiO_2$ | 0.48 |
| $MnO$ | 0.06 |
| $BaO$ | 0.07 |
| $SrO$ | 0.03 |
| $P_2O_5$ | 0.15 |
| $H_2O$ below 110°, | 0.09 |
| $H_2O$ above 110°, | 0.58 |
| | 99.98 |

The cooling of the main granite mass was accompanied in later stages by the segregation of great quantities of aplite, in the form of dikes or often as irregular masses of all sizes from a few yards to a mile or more across. These aplite areas are especially abundant near Butte, and also west and northwest of Boulder in Jefferson county. The texture of the aplite is usually fine grained, though often grading into a coarse pegmatitic structure or, in extreme stages, showing a development of pure glassy white quartz. The formation of aplite seems to have taken place at a very early period; its presence, therefore, does not influence in any manner the occurrence of ore deposits throughout the Boulder granite area.

A large body of dacite-rhyolite, partly intrusive and partly extrusive, covers a goodly portion of the western half of the Boulder batholith. Many smaller isolated plugs and dikes of rhyolite occur in various parts of the granite area, many of which are associated with ore deposits. A notable instance is at Butte, where the ore formation, however, is much older than the rhyolite. These rhyolites, which at one time probably covered a large portion of the granite

batholith, were intruded into the granite long after the latter had entirely cooled and at a time when the present-day physiographic features were strongly developed.

## Rocks of the Butte District.

### *Granite.*

Granite forms the main body of rock inclosing the Butte ore deposits. It is properly a quartz-monzonite, but the term "granite," or "Butte granite," has been so universally employed that it seems advisable to adhere to common usage in this paper. The Butte granite, which is in fact a local area of the Boulder batholith, is a dark gray-green rock of medium coarse granitic texture, but exhibiting at times a more porphyritic phase. This latter variation is especially prevalent in the Steward, Gagnon, and Colorado mines in the southwestern part of the district. The granite everywhere exhibits well-defined systems of joint planes, which, however, do not bear any definite relation to the later fracture systems. The jointings are old, and in many instances in the Butte mines show evidence of slight faulting movements along them, a condition brought about by faults or readjustments taking place within broad inter-fault blocks. While the joint planes have played no part in the determination of the larger features of the fracture systems, they have been of the greatest importance in making the granite adjacent to the fissures more permeable to ore-bearing solutions, and the existence of many of the large "stock-work" ore bodies of the Leonard mine was made possible through the presence of joint planes.

In the veins and intervening country rock in the copper vein area the granite has suffered intense alteration, so that often its original character is only in part preserved.

### *Aplite.*

Small irregular bodies and dikes of aplite are abundant in the northwestern portion of the Butte district. This rock has been noted in unusually large amonnts in the Green Mountain, Mountain Con, and Corra mines. In the main copper belt of Anaconda hill aplite is rarely seen. Immediately to the west of Big Butte it covers an area of several square miles. This exceptionally large body is separated in a general way from the copper zone by the intrusive rhyolite plug forming Big Butte. Where aplite occurs in the form of dikes, which may vary in thickness from a few inches to 50 ft. or more, there is no uniformity in dip or strike, although they generally lie at a low angle. The texture of the aplite is usually fine grained.

though pegmatitic phases are common. In general, it is not often seen as a wall rock of the ore bodies, for, by volume, it forms but a small percentage of the whole rock mass in which the veins occur.

*Quartz-Porphyry.*

In the copper belt of Butte the granite has been intruded by a series of roughly parallel quartz-porphyry dikes which extend in a general easterly and westerly direction across the district. These dikes are relatively narrow, from 10 to 50 ft. wide, but persistent, and they follow closely the general trend of the earliest system of copper veins, thus indicating a close genetic relation between the oldest veins and the porphyry. The dikes, however, are prior to any known vein formation and apparently intruded the granite before the latter had entirely cooled. The vein and fault fracture systems intersect the granite, aplite, and quartz-porphyry dikes alike, and all these rocks are similarly affected by the general alteration processes accompanying later vein formation.

*Rhyolite.*

Rhyolite, both intrusive and extrusive, occurs to the west and northwest of the district. Big Butte, a prominent topographic feature, is formed of intrusive rhyolite, as is the round-topped hill a mile to the south. From these two main bodies there are many offshoots and dikes having a general north and south course, some of which are known to break through silver and copper veins, thus fixing the period of the rhyolite eruptions at a much later date than that of the quartz-porphyry. Extrusive rhyolite extends northwesterly from Big Butte, covering a large area, and in itself is probably the remnant of a much larger rhyolite flow which formerly covered a large portion of the western half of the Boulder granite batholith.

*Andesite.*

Isolated areas of a porphyritic andesite occur to the northwest of the district, the age relations of which have not been definitely determined. They are believed to be comparatively old and probably prior to the granite, corresponding, therefore, to the early andesites of Bull mountain north of Whitehall, Mont., which are known to be intruded by the Butte granite.

*Sedimentary Rocks.*

The nearest sedimentary rocks are found in the Highland mountains, 15 miles south of Butte. Here Algonkian rocks, together with Palæozoic quartzites and limestones, are intruded by the Butte granite, which in turn intrudes also earlier andesite and a basic diorite.

### *Lake Beds.*

Three miles west of Butte is found the eastern limit of an extensive area of a geological formation termed Bozeman Lake beds by the U. S. Geological Survey. This formation is made up largely of sand, gravel, and water-bedded volcanic ash, and at some former geologic period it covered large valley areas in south central Montana. Near Butte these beds fill a long, narrow, pre-existing drainage valley, reaching from the town of Rocker to Melrose, a distance of 30 miles. A 700-ft. mine shaft sunk a half a mile northwest of Rocker discloses a depth of more than 1,000 ft. of these beds, with the total thickness still undetermined.

### *Valley Débris.*

The bottom of the level valley to the south of Butte is composed of coarse detrital sand formed from rapidly disintegrated granite. The greatest depth of this sandy material is unknown. Shafts sunk in the valley floor east of Meaderville have shown the former erosional surface to be buried to a depth of from 200 to 400 ft. in that section, or more than 200 ft. lower in elevation than the present bed of Silver Bow creek where it leaves the valley near the site of the Colorado Smelter.

The manner of occurrence of the Bozeman Lake bed formation near Rocker and the presence of the débris-filled valley south of Butte show conclusively that the former drainage level of the Butte district has been considerably disturbed, and it is probable that the present conditions were brought about either by faulting, volcanic flow coverings, or important crustal movements. These apparent drainage level oscillations are believed by some writers to have played a tremendously important part in the formation of the bonanza chalcocite ore bodies of the Butte district. This feature will be more fully discussed in connection with the problem of ore formation.

## General Geologic Structure.

### *Regional Faulting.*

It is difficult to recognize lines of faulting in the area of the Boulder granite batholith on account of the uniformity of the rock and the frequent masking of structural features by surface débris or late volcanic flow coverings. Numerous faults are known, many of which are mineralized, but the necessary evidence is lacking to fix definitely the direction of throw and amount of displacement along these fissures. Unless dikes or veins are intersected, no clue is afforded as to their magnitude other than the width, amount of fault clay, crushed wall rock, etc.

Taking a broad view of the Boulder batholith, it is of interest to note that fissuring is a common feature over the whole area. While no mapping has been done which permits of an accurate classification of these fissures, it is known that in general they may be grouped into two main series: (*a*) those having a general east-west strike, and (*b*) those having a northwest-southeast strike; both groups corresponding in a broad way with the two most important fracture systems of the Butte district. In general, however, these fissures are more or less localized, and it is not proper at this time to regard them as planes of movement resulting from great regional disturbances.

Of the recognizable faults of this class the only one known to be directly concerned with the geology of the Butte ore deposits is the Continental fault, lying in a north and south direction along the base of East ridge near the Pittsmont mine. There is but little doubt that this fault is of comparatively recent origin, and that it has had an important influence in determining the present topography of the district. The dip is to the west at a steep angle, Butte being on the downthrown, or hanging-wall side. The amount of vertical displacement is probably in the neighborhood of 1,500 ft., while the direction and amount of horizontal displacement have not been determined. A more detailed description of this important fault will be found on a later page. (See pp. 1549–1550.)

## Structure of the Butte District.

Locally, the rocks of the district are intersected by many separate and distinct periods of fissuring. These fractures cut alike the granite, aplite, and quartz-porphyry, but the rhyolite plug forming Big Butte has been found in many instances to be later, often cutting through veins belonging to certain fissure systems. These observed facts throw considerable light on the relations between the aplite, quartz-porphyry and the later rhyolite eruptions, but owing to lack of mine development in the western part of the district it is, as yet, impossible to correlate definitely the rhyolite intrusion with any particular period of fracturing or mineralization in the copper or silver veins. The rhyolite is known to be younger than the earliest vein system, but further than this the exact time of the eruption is not known.

Plate I. has been prepared to illustrate the structural relations of the various fissure systems. It is a horizontal section taken approximately at an elevation of 4,600 ft. above sea level, corresponding to an average depth, in the mines, of 1,500 ft. below the surface. Owing to the small scale of the map and the complexity of the vein and fault structure it has been possible to represent only the larger

and more general features. The Anaconda fractures, which are continuously mineralized, are denoted by solid black. The vein filling other than the crushed granite, in the later fissures is also represented by solid black whether it be of commercial grade or not. Indeed, in many instances, the indicated vein filling is not of commercial grade. The structural features are well developed by mine openings, and therefore the map may be regarded as only slightly idealized.

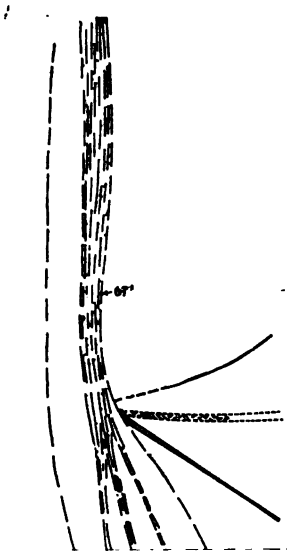
Plate II., a north and south vertical section through the copper district in the vicinity of the Anaconda shaft, indicates the structural relations of intersected veins and fissures on dip. Since the plane of section meets the Blue and Rarus faults at acute angles the dips indicated are flatter than the true dips.

*Classification of Fissures.*

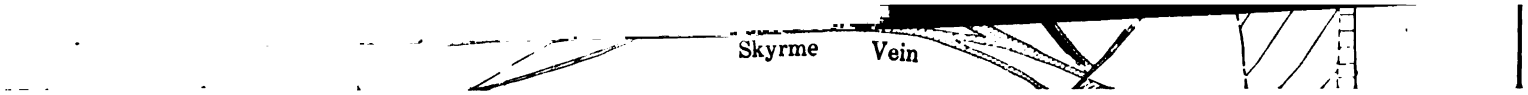
Grouped according to their relative ages the fissures of the Butte district may be divided into six distinct systems, as follows:

1. Anaconda or east-west system, comprising the oldest known fractures.
2. Blue system, the earliest fault fissure.
3. Mountain View breccia faults.
4. Steward system.
5. Rarus fault.
6. Middle faults.
7. Considered as a local feature the Continental faulting may be regarded as a seventh separate period of fissuring.

This classification is based on information gained through a close study of the intersections of the various fissures, aided to some extent by strike and dip and their mineralogical and physical characters. While the development by actual mine openings generally determines definitely the relative ages of intersecting veins or fissures, it is often desirable to learn in advance, if possible, the age relations of certain veins or fractures before such intersections are reached by mine workings. In these cases the vein characteristics must be relied upon for proper guidance. As a general rule the mineralogical composition offers but slight assistance, being a factor depending on geographical location rather than on relative geologic age. Where intersections are not available, the physical character of the fissure together with the determined strike and dip is of importance. As will later appear, the Anaconda fractures are more solidly and continuously mineralized and have a more nearly east and west strike than the later systems. The Blue system fissures are typical fault fissures, with ore occurring in disconnected shoots, and they have a remarkably uniform strike of about



Skvrme
Vein


Skyrme
Vein

. 60° W. The Steward fissures are characteristic faults, with a ightly north of east strike, and contain ore more sparingly than the lue fissures. The Rarus and later fractures are typical fault fissures f marked movement and contain no ore other than fragments or locks dragged from older veins.

When a typical fault fissure is penetrated by a mine opening and is ound to carry no vein mineral or ore where encountered, the strike nd dip are the factors used in provisionally assigning it to a recog-ized fissure system.

The Mountain View breccia faults have certain physical character-stics by which they are readily distinguished from all other fracture or ein systems.

Difficulties in classification are met with in certain areas, where the Anaconda fractures are but slightly mineralized and some secondary novement has taken place along them, resulting in a vein which closely resembles the later faults. Again, in the Mountain View, West Colusa, and Leonard mines, many of the Blue fissures are sol-dly mineralized over hundreds of feet, making them identical in phys-ical appearance with the older veins. A further complication arises in this section owing to the presence of many ore-bearing cross fractures, which, though parallel to the Blue fissures, are in fact older and belong to the Anaconda system.

A more general classification or grouping of these fissures may be made which is to some extent a genetic one, but largely one having to do with the physical nature of the fissures. This reclassification or grouping is offered here in an endeavor to set forth more clearly to the reader the physical differences between the earliest-formed veins and those of later age known as fault veins. A clear understanding of these differences will be of material aid in forming a proper conception of the main structural features of the faults and ore deposits, which at times are bewildering in their complexity.

From a study of the occurrence and nature of the various fissures of the district it is found that they may be arranged in three general groups, as follows:

Group A.—Remarkably continuous complex fractures or fissures of but slight displacement, with but little or no crushing of the wall rock; varying greatly in strike and dip, and exhibiting at times a tendency to develop highly fissured areas in which there may be found a multiplicity of transverse fractures more or less at right angles to the general direction or strike of the main fracture planes. The whole Anaconda fracture system falls within this group. As mineral veins they are in general uniformly and continuously mineralized.

Group B.—Persistent well-defined fissures of marked displacement and of later age than Group A. The fissures of this group are typically fissures of faulting, being invariably accompanied by crushed granite, attrition clay or gouge, etc. It is seldom that these evidences of movement are entirely obscured by later mineralization. In marked contrast to the fractures of Group A, these fissures are not continuously mineralized, the mineral bodies occurring in disconnected shoots. Included in Group B are the ore-bearing faults of the Blue and Steward systems, also the later unmineralized fissures such as the Rarus, Bell, Middle, and Continental.

Group C.—Breccia-filled angular cracks or fractures later in age than Group A, but prior to some of the faults of Group B. These angular cracks exhibit no displacement. They appear to have been formed by the simple drawing apart of the wall rocks. The Mountain View breccia faults and the ore breccias of the Gagnon mine are the important examples of this group.

### *Anaconda Fracture System.*

The Anaconda system includes the earliest-formed fractures of the district. It embraces, therefore, all fissures known to be earlier than the oldest known faults, which are those comprising the Blue system. It is not absolutely certain that all of the old fractures provisionally placed in the Anaconda system are exactly of the same age, but they are practically so. Such additional, recognizable fractures which are older than the Blue fissures and yet are later than the oldest known mineralization are of minor importance and not resolvable into a distinct system. Where such early fissuring is observed, cutting older fractures, it is believed to have resulted from continued movement or slight readjustments along the main fracture lines during the active period of mineralization.

The Anaconda fissures are the oldest geologically and the most important commercially in the district. This group has formerly been called the "East-West" or "Quartz-Pyrite" system of veins, both misnomers, strictly speaking, because the fractures exhibit wide variations from an east-west strike, and the mineral content of the Butte veins is not necessarily an incident of geologic age, but rather one of geographic position. This system is represented by a series of complex fractures, of a general east-west strike, extending across the district. In a broad way, in the copper-producing area they may be divided into two groups, one north of the other and separated from the latter by a relatively barren area. The northerly group embraces the Syndicate, Bell-Speculator and nearby fractures, and

the southerly comprises the Anaconda, Moonlight, O'Neill, and others.

*North Group.*—The most important fracture zone of this group is the Syndicate vein. In adjoining properties it is successively called the Yellow Jacket, Poulin, Buffalo, Mountain Con, Wake Up Jim, Middle vein, Bell-Speculator vein, etc. Westerly from the Mountain Con and Green Mountain mines the Syndicate vein is usually a well-defined, single, mineralized fissure, rarely branching excepting for short distances, forming "horses," or included barren granite blocks. Toward the east, however, it divides, and its various branches take a southeasterly course toward the Mountain View mine and split further into smaller veins until the identity of the major fissure is completely lost. These smaller branches form in a general way, no doubt, a connecting link with the great Anaconda complex in the vicinity of the Mountain View mine.

Other important fissures of the north section of the copper-producing district belonging to the Anaconda system are the Badger State, Berlin, North Wild Bill, Modoc, Mountain Con South, Eastwest Gray Rock, and the North vein of the Mountain Con mine. In addition to those above named there are numberless small cross fissures and cracks, associated with the larger fissures, and of geological interest, but not important as ore producers.

The main Syndicate vein in the Mountain Con and Buffalo dips 88° to the south. It is variable, however, at times dipping slightly north. Southeasterly it has a southerly dip of 60°. The North Wild Bill is vertical or slightly south dipping. The Badger State dips north at the surface, changing to a steep south dip in depth. The Eastwest Gray Rock and Modoc fissures dip 65° south, and the North vein of the Mountain Con dips 50° to the south.

*South Group.*—The south fracture complex known as the Anaconda vein in the Never Sweat, Anaconda, and St. Lawrence mines is of vastly more commercial importance than the north group. Beginning on the extreme west in the Gagnon mine, where it forms a compound fissure zone combined with later faulting, and passing easterly, it is known successively as the Gagnon, Original, Parrot, Anaconda, and Mountain View South vein, etc. Although intersected and displaced many times by later faults the identity of the main fissure is not lost. Easterly from the St. Lawrence mine this fracture exhibits great complexity. Passing through the Mountain View mine there is developed a great fissured zone bounded on the north by the Shannon vein, the easterly extension of which is the Colusa vein, and on the south by the Mountain View South No. 2 vein, the last named being

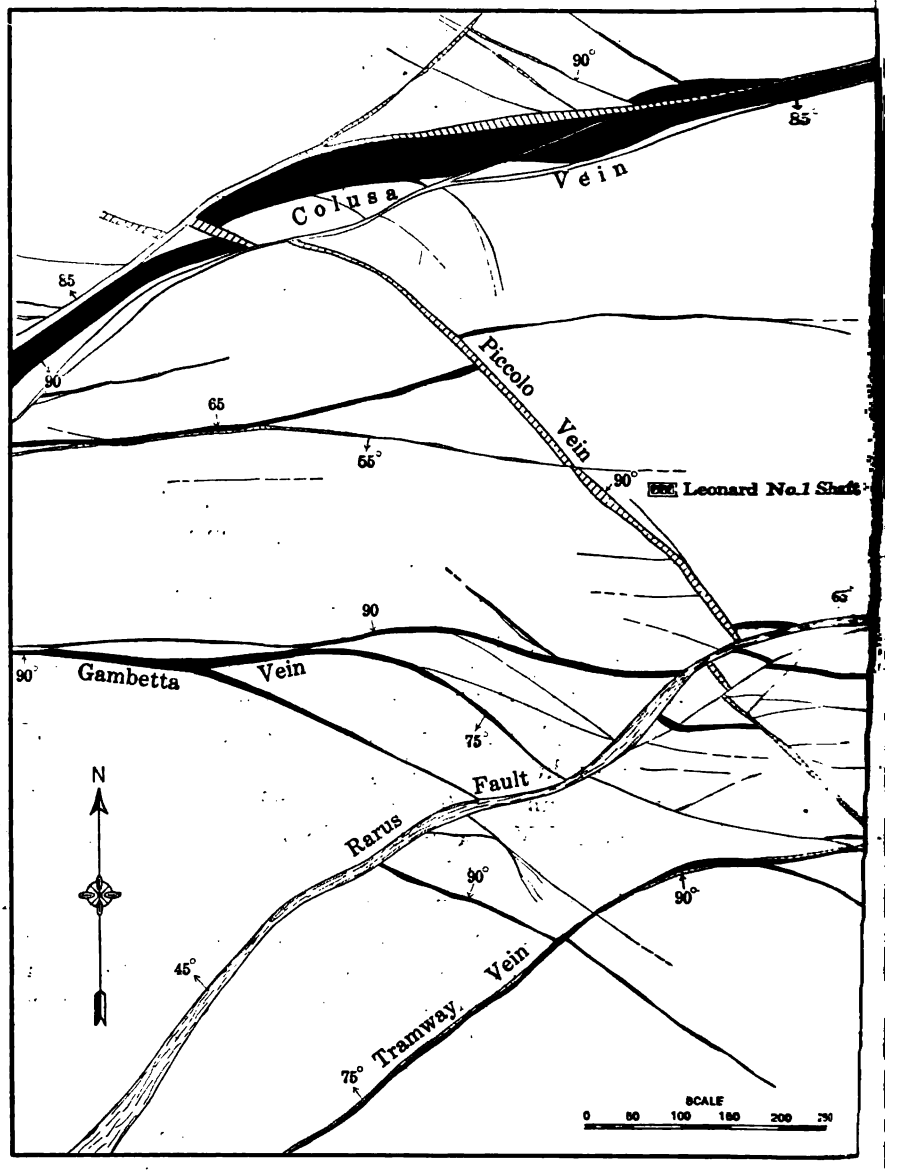



FIG. 1.—PLAN OF A PORTION OF THE 300-FT. LEVEL OF LEONARD MINE, SHOWING VEIN STRUCTURE.

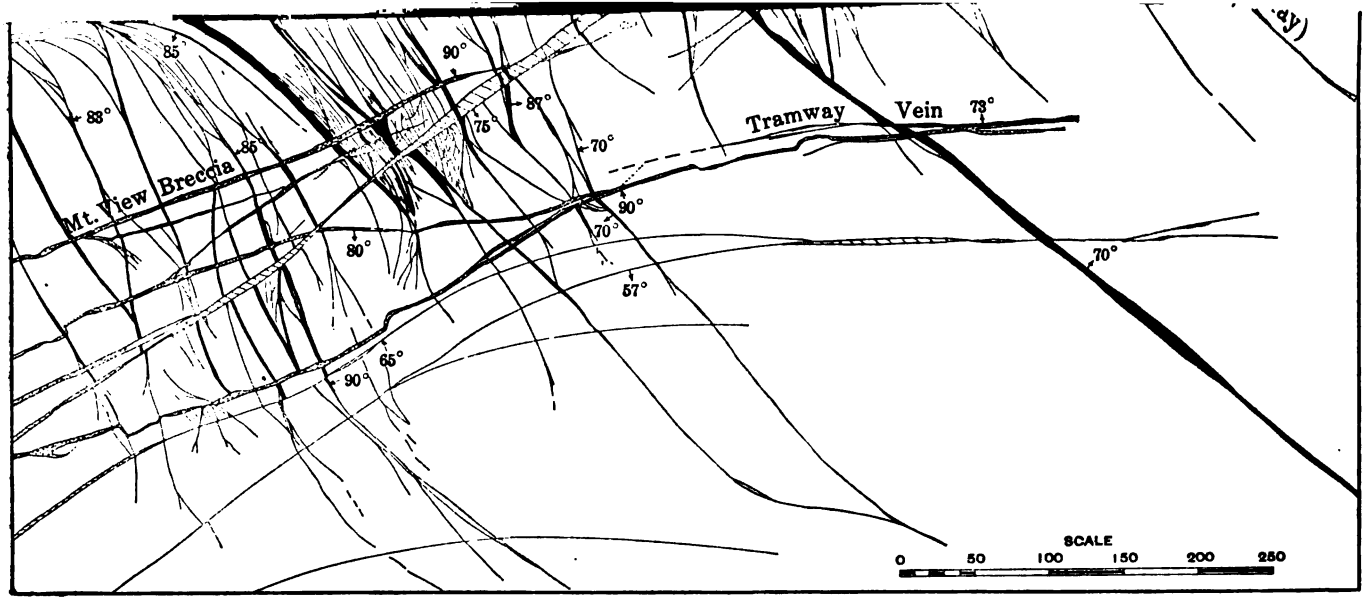



FIG. 2.—PLAN OF A PORTION OF THE 1,200-FT. LEVEL OF THE LEONARD MINE, SHOWING DEVELOPMENT OF TRANSVERSE FISSURING IN COLUSA-LEONARD VEIN.

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the faulted extremity of the main Anaconda fissure. The Shannon or north bounding fissure has a strike of N. 75° E., and a dip of 85° north, while the South No. 2 vein strikes S. 75° E., and dips 65° south. In its southeasterly course the latter branches into many fissures, forming many smaller veins of the Rarus and Berkeley mines; the identity of the principal fissure, however, is lost.

The distance between the limiting Shannon and South No. 2 fissures at the Mountain View shaft is 400 ft. For several hundred feet east of the Mountain View shaft there are a great number of connecting cross fractures between the Shannon and South No. 2 fissure. These cross fractures have a general strike of N. 20° W., with a westerly dip of about 75°. In many instances these cross veins have proved to be good producers of ore, although they are not of great width. The structure here is much complicated by the presence of several mineralized fissures belonging to the Blue system, being in fact the southerly extensions of the Gray Rock fault veins.

Owing to the divergence on strike of the Shannon and South No. 2 veins, the space between these limiting fissures becomes greater toward the east. There is an increasing tendency on the part of the Shannon vein to throw off north-south fissures in a southerly direction. In the West Colusa upper levels enormous stopes were made, using the north boundary of the Shannon fissure as the north wall, while the larger portion of the material mined out was mineralized granite intersected by a multiplicity of closely spaced fissures having a general strike of about N. 20° W., or nearly at right angles to the strike of the Shannon fissure. In the Leonard mine the north and south fissuring is not present in the upper levels. (See Fig. 1.) The first tendency toward a development of transverse fracturing appeared on the 600-ft. level. At greater depths the north-south fissuring became more intensified, while the east-west fissures became gradually less prominent, until it was found that below the 1,200-ft. level only the great areas of closely spaced north-south fissures remained, bounded on the north by the limiting north wall of the Shannon fissure, and forming the great "horse-tail" ore bodies of the Leonard, Tramway, and West Colusa mines. (See Fig. 2.)

Going northeasterly, the termination of the Shannon-Colusa-Leonard vein is remarkably abrupt and gives rise to a most peculiar geological condition. The throwing-off tendency of the Shannon fissure reaches a climax near the Leonard shaft, where the east-west fissuring ceases and the identity of the Shannon-Colusa vein is completely lost in a perfect maze of north and south fissuring without definite boundaries. (See Plate I.)

Lying to the south and parallel to the main Anaconda fissure are two important veins known as the Moonlight and O'Neill veins. The O'Neill fissure is distant about 1,000 ft. from the Anaconda, and it forms the general southern boundary of the great zone of alteration. The eastern extension of the O'Neill vein is known as the Pennsylvania No. 1 vein in the Pennsylvania mine, and as the Silver Bow vein farther to the southeast. The Moonlight fissure is prominent in the Moonlight and Anaconda mines, but dies out in an easterly direction before reaching the Pennsylvania shaft.

Many small unimportant fractures belonging to the Anaconda series occur south of the O'Neill vein, notably the Cambers, J. L. C., Glengarry, Silver Bow No. 3, and others, but these have been only slightly developed, owing to their non-productiveness in depths greater than 400 ft.

Southeasterly from the Rarus, Berkeley, and Silver Bow mine fissures of the Anaconda system pass into the property of the East Butte Mining Co., following in a general way the course of the quartz-porphyry dikes. These relations may be readily understood by referring to Plate I.

The principal veins of the so-called silver area are believed to be mineralized fractures of Anaconda age. Owing to the mining inactivity for many years past in this area, and through the general lack of definite information concerning the geological conditions, it is not possible to map with any degree of accuracy the vein structure in that section.

*Blue System of Fissures.*

In the Blue system of fissures is included a series of northwest-southeast fault fissures which cut and displace the fractures of the Anaconda system, but are themselves older geologically than the Steward faults. Their general strike is in the neighborhood of N. 55° W., with extreme variations within a range of N. 30° W. to N. 75° W. There is a notable uniformity in the arrangement or spacing of the important members of this series of faults in a northwest-southeast direction, or at right angles to the general line of strike. This feature is well illustrated in Plate I.

The important fissures of the Blue system, named in order of their occurrence, beginning on the southwest, are the No. 2, No. 1, Clear Grit, Blue, Diamond South or Dernier, High Ore, South Bell, Skyrme, Edith May, Jessie, Gem, and Crœsus, all of which are mineralized and most of which contain ore bodies of immense value.

In contrast to the marked uniformity in strike, permitting accurate projections over hundreds or even thousands of feet, the dip of the

Blue veins tends toward the opposite extreme. The fissures as a rule are steep, but the variations in dip aloug a single fissure may cover a wide range. It is not uncommon to note a change from a north dip to a south dip, followed by a change again to the north, all taking place within a vertical range of 1,600 ft. The Edith May vein, for example, dips 80° north from the surface to the 700-ft. level; it is vertical from the 700-ft. to the 1,600-ft. level, and has a south dip below the 1,600. The Jessie exhibits tendencies even more erratic. In the Modoc mine it is very steep or slightly north, dipping near the surface, changes to a 60° south dip at a depth of about 500 ft., retaining this flatter dip to the 700-ft. level or thereabouts, where it again assumes a vertical dip to the deep levels.

While the general tendency of the fissures of this system is to dip south, some of them do not follow this rule. The most southerly ones, the Clear Grit, Flat, and Blue, have decided south dips, from 45° to 65°. Those intermediate, the Skyrme and Edith May, are more nearly vertical, while on the extreme north the tendency is to dip steeply north in the uppper levels, with a change to the south at greater depths. The general variations in dip are well illustrated in Plate II. The Skyrme vein is of a particularly wavy habit on its descent into the earth.

The fissures of the Blue system are faults of marked displacement. Owing to the uniformity of the country rock in which the fissures occur it is difficult to determine accurately the direction of movement and amount of displacement. Where, however, a fault plane intersects two or more veins having different strikes and dips, these factors may be closely approximated. The amount of movement along the more important Blue fissures ranges from 150 to 300 ft. The High Ore fault vein displacement has been determined by Paul Billingsly to be in the neighborhood of 270 ft., with the line of direction of movement making an angle of 35° with the horizontal. In all of these fissures the south or hanging wall has apparently moved to the southeast and downward relatively to the foot wall and the direction of movement of the hanging wall has been along a line making an angle of 15° to 35° with the horizontal. The Blue faults more properly fall under the head of thrust faults, rather than normal faults as the latter term is commonly understood.

In cases where but one vein is cut by the fault plane there is a certain feature to be observed which offers a clue to the direction of movement. Large grooves, or striations, may be seen in the hard ore or the wall rock adjacent to the principal planes of movement. These grooves, or waves, measure from a few inches to a foot or

more from crest to crest, with depths of an inch or more. The writer is led to believe that they are good indications of movement because they exhibit a remarkable uniformity in dip or pitch along the walls where recognized in widely separated members of the series and they correspond in this respect to the larger fault vein structures earlier described. This belief is corroborated by the determination in the case of the High Ore fissure above noted. The small scratches and parallel lines so commonly seen in the soft fault clay are of no value whatever as indications of direction of important movements. In a single slab of clay an inch thick striations or lines may be found extending in every conceivable direction.

Viewed in a broad way the fissures of this system show a strong tendency to branch toward the southeast, a feature well marked in the Blue, Skyrme, Edith May and Jessie veins. Toward the west and northwest there is a noticeable swerving to a more westerly strike and many instances of unions on strike of important fissures are noted. A good example of this condition occurs in the Corra mine where the High Ore, South Bell and Skyrme veins unite forming a single fissure. (See Plate I.) This evident tendency to unite westerly on strike with a possible greater displacement along the large single fissure may have some significance when taken in connection with their origin.

*Mountain View Breccia Faults.*

A curious and interesting feature in connection with the copper veins is the occurrence of peculiar breccia faults (filled cracks) in certain localities within the copper-producing area. They are of frequent occurrence in the Mountain View and Leonard mines in the eastern part of the district and in the Gagnon mine on the west.

These breccia faults, or veins, are persistent angular fissures filled with fragmental material composed of country rock or with some fragments of earliest veins. The size of the fragments ranges from pieces a foot or more in diameter down to fine sand. Generally the breccia is a mixture of all sizes, although at times it is all fine material so arranged in the fissures that it resembles stratified sandstone. As a rule it has the general appearance of ordinary concrete showing angular rock fragments loosely set in a matrix of finer disintegrated granite grains. Often the larger pieces are rounded and look water-worn like stream pebbles. At or near the contact or intersection with older veins the breccia frequently contains many fragments of the vein filling, in sufficient quantities at times to constitute ore. This feature is especially important in the Original and Gagnon

mines, where big stopes have been made in ore breccia associated with older veins. In fact, from the surface to the deepest levels this has been a characteristic feature of the Gagnon-Original vein.

The general strike of the Mountain View breccia faults is about N. 75° E. They extend from the most westerly workings of the Gagnon easterly through the Original and Steward mines, thence northeasterly toward the Mountain View mine. At the present time there is not sufficient mine development to determine whether a general continuity exists between the breccia faults of the Mountain View mine and those of the Original-Steward mines. In the St. Lawrence mine they are found in abundance extending northeasterly into and through the Mountain View and West Colusa mines; most of them, however, die out in the vicinity of Leonard No. 1 shaft. While the general northeasterly course is well defined, locally and in greater detail, the strike is extremely irregular. They are angular and zigzag in plan, resembling streaks of lightning. A fissure from 12 to 18 in. in width may be found regular in course for several yards, when it will suddenly offset at right angles, following a joint plane or vein wall for a distance of several feet, after which it again resumes a normal course. These freakish tendencies are well illustrated in the general vein map, Plate I.

The breccia faults vary greatly in width, even within short distances along their strike and dip. Widths of from 6 in. to 2 ft. are common in the Mountain View mine, while thicknesses of from 10 to 30 ft. are prevalent in the Gagnon mine. The north cross-cut of the 1,900-ft. level of the St. Lawrence mine develops a body of breccia 135 ft. in width. A cross-cut on the 1,800 ft. level of the Mountain View mine has been extended 80 ft. into breccia, disclosing but one wall.

*Age of the Breccia Faults.*—The formation of these interesting faults undoubtedly followed the period of Blue vein formation. Without exception the breccia is found to cut through the ore of both the Anaconda and the Blue vein systems. Instances have been noted where secondary unimportant movements have taken place along the Blue vein fissures, intersecting and displacing slightly the Mountain View breccia. The Rarus fault cuts and displaces the faults, but the relation between the Steward faults and the breccia has not been satisfactorily ascertained. In the Gagnon mine the breccia is often much squeezed and faulted by northeast fault movements apparently of Steward age. On the whole the evidence indicates that these breccia faults were formed subsequent to the Blue fissures and later than the ore filling in the Blue vein fissures. They were formed, however, prior to the Rarus faulting and probably slightly antedate

the Steward faults. There are indications that the breccia veins were not all formed at the same time, but that in certain instances in the Mountain View some were formed as late as the Middle faults.

*The Steward Fissure System.*

In the Steward system is included a series of northeast-southwest fault fissures extending across the district. They cut and displace the quartz-porphyry dikes and the veins belonging to the Anaconda and Blue systems. The Steward fissures strike slightly more northeasterly than the veins of the Anaconda system, though the angle of intersection is very acute, often forming strike faults along them. Referring to Plate I., it will be noted that these fissures are more or less regularly spaced from north to south and that the strike does not vary much from a N. 65° E. course. The dip is uniformly to the south, ranging from 50° to 75°, with an average approximating 65°.

The most prominent members of this series, naming them in order from south to north, are, the Rob Roy, No. 16, Mollie Murphy, No. 6, Steward, Modoc, La Plata, and Poser. In addition to those named there are many smaller and less important fractures of Steward age, in part branches from the larger fissures, and in many instances sympathetic fractures of apparently limited lateral extent. The Bell fault in the Mountain Con and Diamond mines is a prominent northeast fissure paralleling the Steward faults, but of doubtful age. It is provisionally placed in the Steward system, but later developments may prove it to belong to a later period, possibly as late as the Middle faulting. The Bell fault, on the other hand, dips 65° to the south in the upper 500 ft., 85° to the north for the next 700 ft., and 80° southerly again in the deep levels. With the general dip to the south, the north or foot wall has moved downward relatively to the hanging wall. (See Fig. 3.) The Bell fault is not known to contain ore other than drag from older veins. No intersections between the Bell fault and undoubted Steward fissures have been developed by mine workings. It is therefore impossible to learn their true relationship, since both exhibit the characteristic fault structure.

The Middle fault of the Mountain View mine was formerly believed to be of the same age as the Steward and it was long considered a member of that system. More recent developments prove beyond question that the Middle fault is later. It is believed that the Middle fault is, in part, a strike fault, along an earlier fissure belonging to the Steward system. This feature will be more fully discussed in connection with the Middle fault.

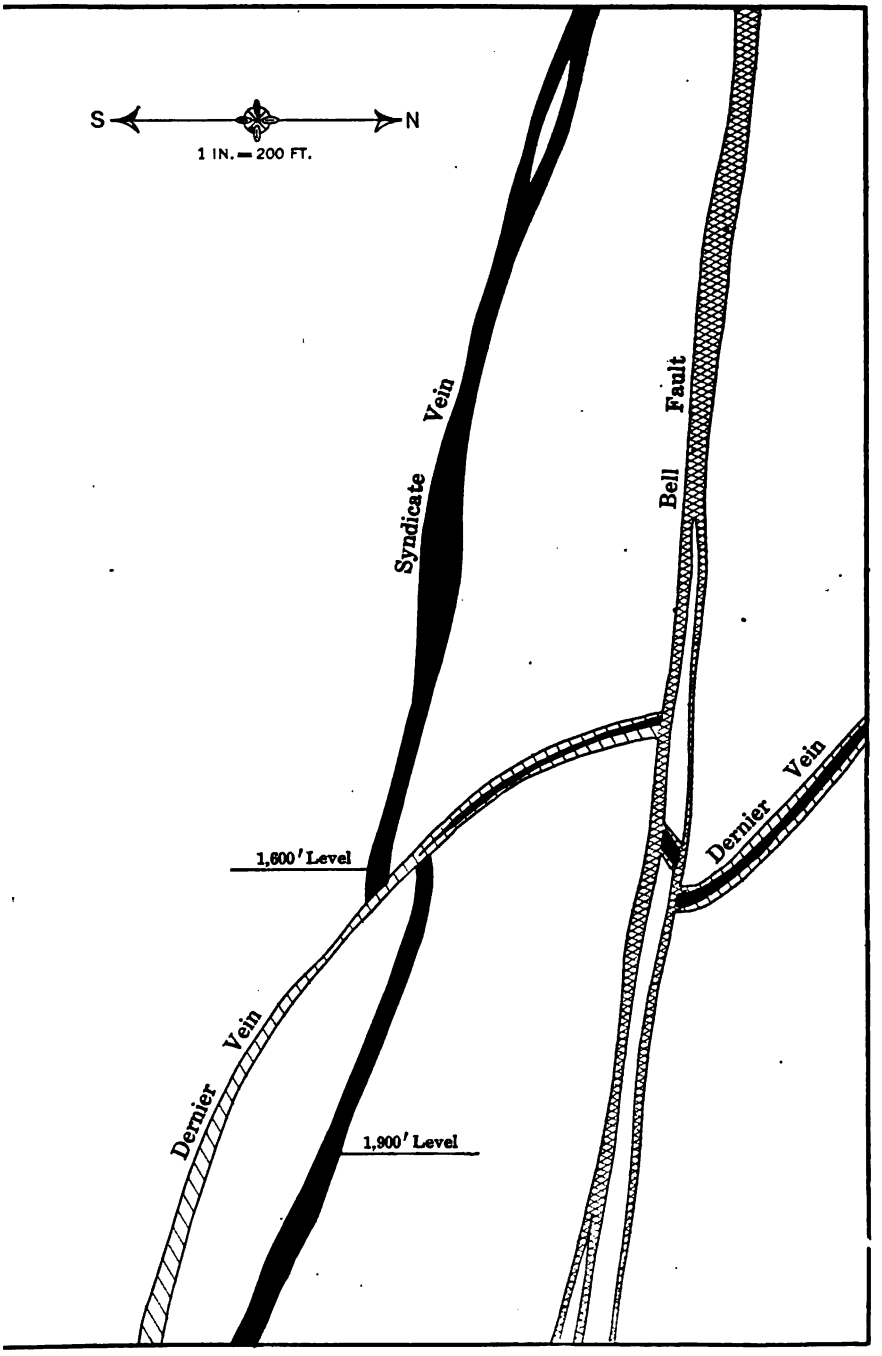



FIG. 3.—VERTICAL SECTION SHOWING DOWNWARD MOVEMENT OF FOOT-WALL COUNTRY ALONG BELL FAULT.

The Steward faults are characteristically fissures of movement. They are narrow zones, or belts, of crushed granite accompanied by well-marked seams of attrition clay. The widths vary from a few inches up to 10 ft., depending largely on the magnitude of the faulting. The total displacement along the faults varies from 50 to 150 ft. in the important fissures. The Steward fissures are normal faults inasmuch as the hanging wall has moved downward relatively to the foot wall. The movement, however, did not take place downward on a line normal to the strike, but along a line within the plane of the fault making an angle of approximately 70° with the horizontal.

With but few exceptions the Steward faults carry no ore; but it appears that in certain instances they have exerted an important influence on the veins of the earlier Anaconda system. This feature will be taken up later in the description of the Steward fissures as ore producers.

### *The Rarus Fault.*

*Age.*—The Rarus fault is a complex fissure of later age than the Anaconda and Blue systems of fissures. It is more recent also than the peculiar Mountain View breccia veins. It cuts and displaces the principal fissures of the Steward system, but the true age relationship of the Rarus fissure and the Bell and Continental faults is not known. In the St. Lawrence, Mountain View, and West Colusa mines the Rarus fault is faulted by a series of closely spaced south-dipping fissures called the Middle fault, as illustrated in Plate I. Northeasterly in the Leonard and to the southwest in the Moonlight mine the displacement along the Middle fault is slight and not readily recognized. Other northeast fissures in the Mountain View mine, parallel to the Middle fault, are also known to cut and displace the Rarus fault. It was first believed that the displacement of the Rarus was due largely to block readjustments locally in the Mountain View mine, where the whole body of the granite is intensely altered and there exists a multiplicity of fractures. Later evidence seems to indicate that these late movements have been more widespread than first realized, and it is not improbable that the Middle fault may mark but one of a more extensive series of movements taking place subsequent to the Rarus fault.

*Geographic Position.*—The Rarus fault has been opened up by numerous mine workings from the East Colusa mine on the northeast to the Belmont on the southwest, and on dip from the surface to the 2,800-ft. level. The dip is remarkably uniform throughout, varying but little from 45° to the northwest. The strike is variable, ranging from N. 30° E. to N. 80° E., the average being roughly N. 50° E. Although a well-defined compound fissure, broadly speaking, the Rarus fault is

detail is exceedingly complex. In places there are two separate limiting movement planes defined by heavy dark gray fault gouge from 1 to 8 in. thick, usually accompanied by from 10 to 30 ft. of finely crushed altered country rock, and ore fragments where in the vicinity of older veins. The perpendicular distance between the two limiting planes varies from 20 to 250 ft. Where the distance is less than 50 ft. the whole intervening rock mass is thoroughly crushed. With greater separation of the two planes the interfault ground becomes less and less affected, so that in extreme cases, as exhibited in the Rarus mine, the ground between the ultimate boundary fissures shows but little, if any, effect of fault movement. Generally, however, the included country is intersected by parallel or sympathetic fractures crossing diagonally from wall to wall. In the Rarus mine the limiting fissures are distinct and well marked and were early termed the Rarus "hanging-wall" and Rarus "foot-wall" faults.

*Displacement.*—The displacement along the Rarus fault is greater in the southwestern part of the district than toward the northeast; in fact, in the Leonard and Colusa mines the fissure becomes so split up that its identity cannot be established northeasterly from the Leonard shaft. It does not displace the Colusa vein in the upper levels and is believed to die out rather suddenly in this region. The movement along the fault plane is not normal to the strike at all points. The hanging wall has moved downward, the line of movement within the fault plane making an angle of 90° with the horizontal in the Belmont and 60° or even less in the Leonard. In the Belmont the displacement is 350 ft., in the Rarus mine 240 ft., and in the Leonard less than 120 ft.

An interesting feature in connection with the Rarus fault is the shifting of the movement from hanging wall to foot wall, or *vice versa*, through the medium of diagonal movement planes intersecting the interfault granite. This is well illustrated in Fig. 4, which shows the faulting effect on certain veins of the Pennsylvania mine. At times the whole fault displacement is so evenly distributed between the limiting boundaries by means of small movement planes that no actual cut-off of the intersected vein occurs. In certain instances the movement has been entirely taken up by interior adjustments within the granite blocks so that no visible movement planes are developed. (See Fig. 4.) The phenomenon of step faulting of veins is a result of the frequent shifting of the movement from the wall through the intervening country rock and veins. The cause seems to be found in the fact that the direction of downward movement of the hanging wall was not normal to the strike at every point.

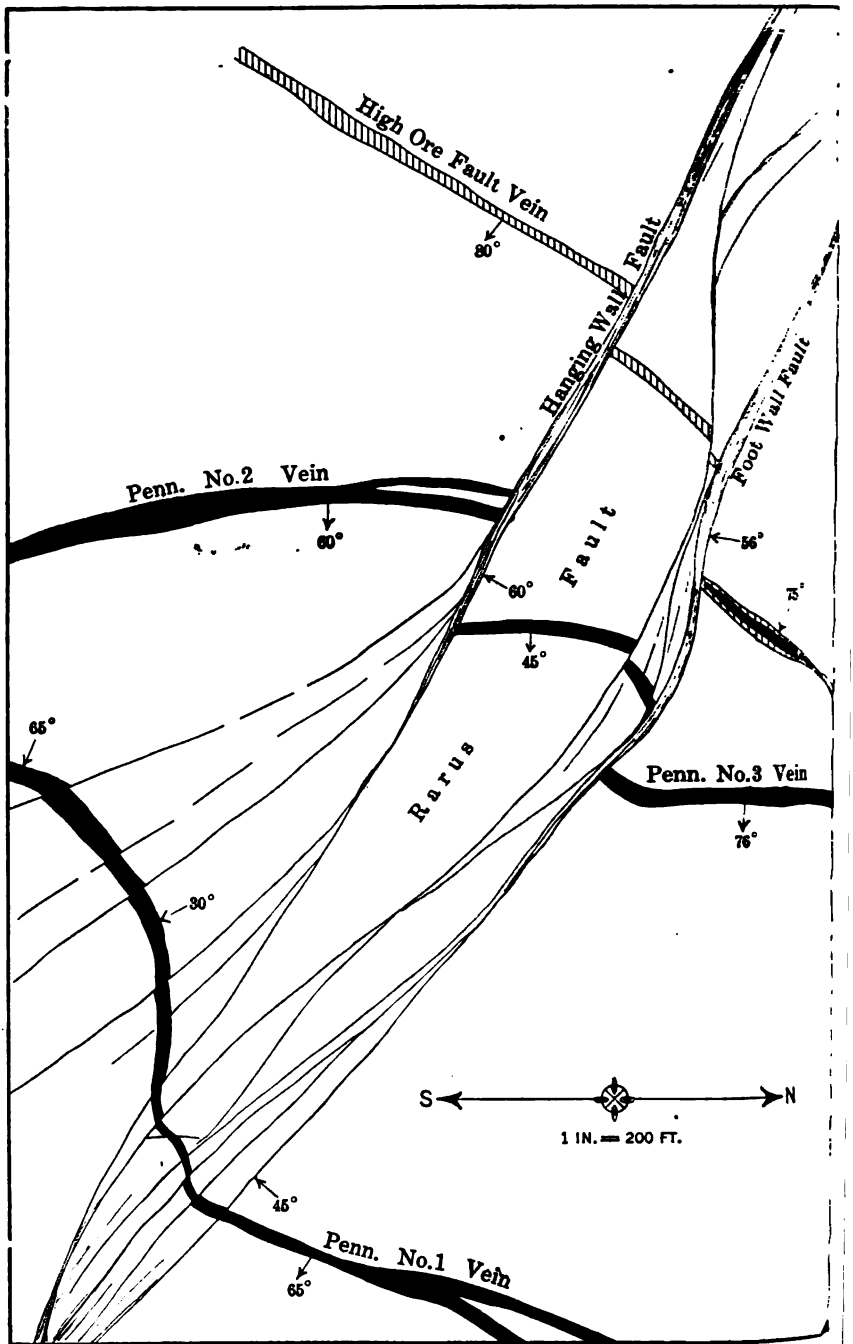



FIG. 4.—PLAN OF 500-FT. LEVEL OF PENNSYLVANIA MINE, SHOWING EFFECT OF RARUS FAULT ON DIFFERENT VEINS.

*Mineralization.*—The Rarus fault has not been found to carry ore other than fragmental ore dragged from older formed veins. These included blocks or fragments varying in size from small bits, or pebbles, etc., up to large blocks or slices of veins which reach from wall to wall of the fault. In thickly veined areas it frequently happens that segments of one vein are moved within the fault to a position where the ends are brought opposite an entirely different vein lying without the fault zone. This phenomenon formerly gave rise to the belief by some geologists that no displacement occurred along the Rarus fault, for it was found to be possible in rare instances to extend a drift entirely through the fault zone on vein, although the faulted portions of three separate veins were required.

Mine developments so far indicate that the only indigenous minerals of the Rarus fault are quartz and pyrite in sparsely disseminated amounts. The influence of the Rarus fault on intersected veins will be further treated under vein descriptions.

In the Corra and Gray Rock mines a small fault fissure, known as the Corra fault, having a normal displacement of 40 ft., striking N. 75° E. and dipping 50° to the north, is believed to be of Rarus age. It cuts the Anaconda, Blue, and certain fissures of the Steward system. The age relation between the Corra and Bell faults has not been determined. The Corra fault carries no ore.

### *The Middle Fault.*

Formerly it was believed that the Rarus fault was the latest geologically in the entire district, with the possible exception of the Continental fault at the base of East ridge. Recent mine developments, however, have disclosed the fact that the Middle fissures of the Mountain View mine represent a period of movement later than the Rarus. On account of its strike, dip, and general physical character, the Middle fault was early assigned to the Steward fissure system. It was thought that the No. 16 vein of the Rarus mine, a Steward fissure, was the faulted segment of the Middle fault, lying beneath the Rarus fault. It has been abundantly shown, however, by the recent development of many intersections of the Rarus and fissures of Middle fault age that the latter are post-Rarus.

Owing to the extreme complexity of fissuring in the region of the Mountain View mine the relation between the numerous fractures has not been satisfactorily worked out. The available evidence indicates that the Middle fault is coincident with, and, in fact, forms a strike fault along the segment of the No. 16 vein lying above the Rarus fault. (Fig. 5.)

According to the best information available, the Middle fault exhibits the greatest displacement in the Mountain View mine and vicinity, where it approximates 75 ft., with a lessening amount of movement both to the northeast and southwest. There is certain evidence indicating that the Middle fault is the manifestation of a

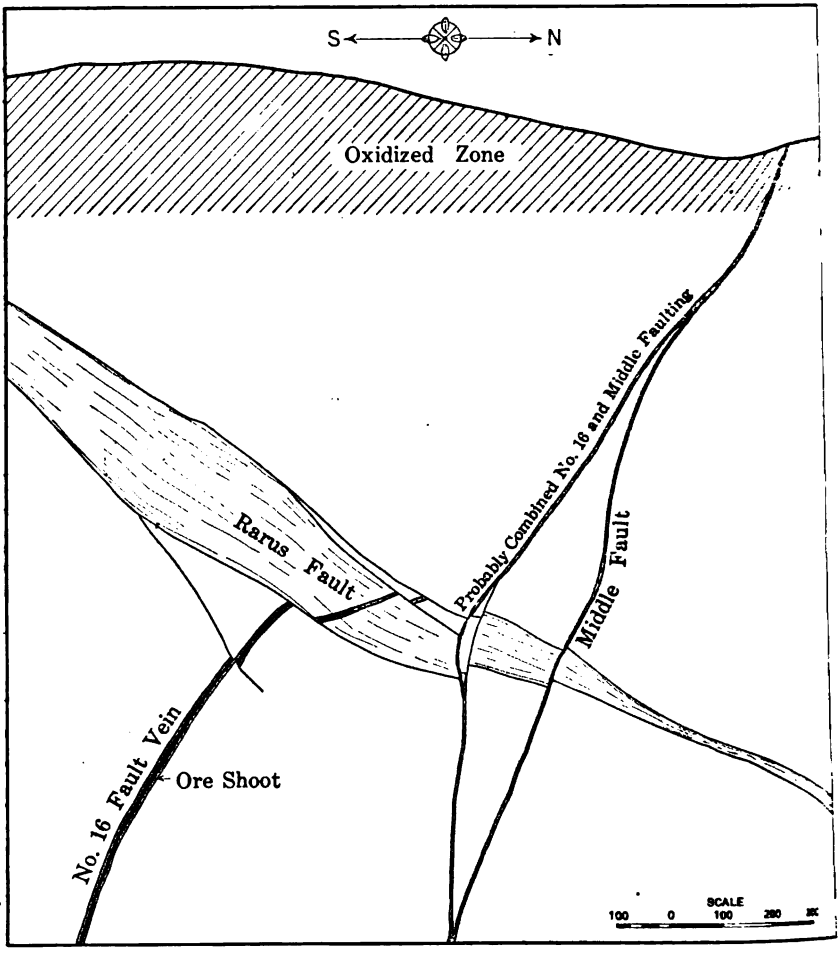



FIG. 5.—VERTICAL SECTION THROUGH THE MOUNTAIN VIEW MINE, SHOWING THE RELATION BETWEEN CHALCOCITE ORE SHOOT OF NO. 16 VEIN, OF THE STEWARD SYSTEM, AND THE RARUS AND MIDDLE FAULTS.

sagging tendency within this particular section, caused by unusual weakness in the earth's crust due to extreme alteration and intense fissuring.

The Middle fault fissure is of the same general character as the typical faults of the district. It carries no ore and exhibits no min-

eralization whatsoever. The strike is N. 65° E. with a general southerly dip, although in deep levels and to the northeast it dips slightly north. The average dip down to the 1,200 Mountain View is 65° to the south, below which level it quickly steepens to a vertical or slightly north dip, which is maintained to the deepest mine levels.

### *The Continental Fault.*

The Continental fault lies almost entirely without the copper-producing district. It is a complex fissure zone from 200 to 1,000 ft. wide, having a general north and south strike following close to the base of East ridge. It may be seen outcropping in the Great Northern railroad cuts at Horseshoe Bend, from which point it passes southerly through the Six o'Clock, Greenleaf, Bullwhacker, Montgomery, and Amazon-Butte properties. Continuing to the south it follows closely the base of East ridge in a direction toward Nine Mile canyon.

The general dip of this fault is to the west at an angle of 75°. Within the ultimate boundaries of the main fissured zone, wide variations in dip are commonly met. In the Greenleaf mine many of the important movement planes are vertical or slightly west dipping. It appears that the western or hanging wall has moved south and downward, the amount of vertical throw being unknown, but possibly in the neighborhood of 1,500 ft. The horizontal throw is probably several hundred feet, but the evidence is not sufficient to even approximate the amount. The movement of the fault is rarely concentrated along a single plane, but is distributed over a series of roughly parallel fissures, accompanied by a variable amount of breaking or crushing of the intervening country rock. In the Six o'Clock mine a thickness of 4 ft. of tough, dry gouge, dipping 67° to the west, marks the foot-wall fissure. At this point the fault movement is much more concentrated along a single fissure than at points farther south.

No primary mineralization has been recognized in this fault. The intense alteration of the granite, so characteristic of the Blue and Steward faults, is absent. The movement planes of the Continental fault are marked by a tough gouge composed of finely comminuted, unaltered granite. The waters now found within and along the fissure are no doubt of meteoric origin and have effected but slight changes in the crushed country rock. It would seem reasonable to infer from these observed facts that the Continental fault is of comparatively recent origin, and that, at great depths, it did not reach a source of primary vein-forming waters.

The age of the Continental fault relative to the various fractures

of the district is proved by evidence obtainable in the East Butte mines, where two north and south fractures, undoubtedly as a part of the Continental complex, cut and displace veins of the Anaconda and Blue systems, and in one instance a northeast south-dipping fissure, probably of Steward age. The relatively recent date of this fault is also suggested by Plate IV., a vertical section taken along the general course of the Anaconda vein showing the probable relation between the fault and the oxidized and sooty chalcocite zones.

## Rock Alteration.

Extensive alteration of the rocks has taken place in the Butte district. There are two principal zones or areas associated with the copper veins in which the rocks are altered to an unusual degree. These areas are closely related to the more important developments of the earliest-formed veins, or those of the Anaconda system. One of these alteration zones follows rather closely the Syndicate and other early veins in the Mountain Con, Gray Rock, and Diamond mines. In the two last-named properties the zone reaches a maximum development where intersected by many fault veins belonging to the Blue system. The outline of this zone is so irregular that it cannot be mapped with any degree of accuracy. On Plate III. an attempt has been made to indicate the general conditions. The northerly zone is seen at this elevation to be composed of three or more disconnected areas, all of which merge into one more or less continuous area at greater depths. When referring to the map, Plate III., it should be kept in mind by the reader that only the more important areas of rock alteration are shown by the cross hatching; furthermore, there has been a marked alteration of the granite along nearly every vein and fissure shown on the map, but usually not extending outward for appreciable distances from the immediate influence of the fissures, at least not extensive enough to be accurately represented on a map of this scale.

The largest and most important area of altered granite is in the vicinity of Anaconda hill. In this part of the district a great altered belt extends easterly from the Parrot mine through and including the Never Sweat, Anaconda, Mountain View, West Colusa, Berkeley, and Silver Bow mines, and within this whole area it is next to impossible to find a hand specimen of rock which has not undergone marked chemical and physical changes. As in the case of the north belt above described, there is an unmistakable close genetic relation existing between the widespread rock alteration and the Anaconda vein system, a fact more readily understood by reference to Plate III.

In these two general zones intense alteration has taken place n

only within and along the veins and faults, but the entire rock mass, whether granite, aplite, or quartz-porphyry, has been invaded by active metasomatic processes, resulting in a product differing markedly both in chemical and physical character from the original rock. The hard, dark-colored granite has been changed to a whitish or mottled gray, less firm rock, usually peppered generously with iron pyrite. The quartz-porphyry has been altered from a grayish green color to a yellowish white rock studded with glassy quartz phenocrysts. When subjected to further action by descending meteoric waters the rocks became weaker, more porous, and there is slightly more uniformity of structure. In extreme cases in newly opened workings of the upper levels the rock breaks in great slab-like pieces, the fracture planes extending directly across and without regard to joint planes.

In the areas between and outside the principal altered zones of the district the country rock generally shows but slight change, excepting within or adjacent to important veins, fissures, and shear zones, and to a less degree along joint planes and cracks. It is not uncommon to find, within general areas of unaltered rock, cracked or crushed zones in which there has been but slight chemical change in the rock. These broken zones, which may vary in width from a few inches up to several feet, appear not to have reached a source of active water circulation. Observations seem to indicate that the rock alteration is becoming more widespread as greater depths are attained. This is particularly true in the region of the Diamond, High Ore, and Leonard mines.

### *Causes of Alteration.*

The alteration in the Butte rocks may be traced to three general causes: (1) vein formation; that is, alteration effected by solutions which were primarily responsible for the formation of the ores; (2) common hydro-metamorphism, or effects produced by the action of descending meteoric waters; and (3) oxidation.

These processes are, in fact, superimposed in part. The chemical and physical changes resulting from vein-forming processes are modified in the higher levels by meteoric waters, and a further change takes place when such altered rocks are brought by erosion or other causes into the zone of oxidation.

As already stated, the great alteration zones are distinctly associated with veins and fractures belonging to the oldest known vein period, or that embracing the Anaconda system. In regions much faulted and crushed one might naturally expect to find the greatest alteration, because of the apparently greater permeability of the broken country rock to solutions traversing the fissures. That such is not

necessarily the case is believed to be due to the following conditions:

1. The earliest ascending waters and gases were much more active chemical agents than later solutions, owing to higher temperature and pressure conditions, and to some extent, possibly, to their chemical composition.

2. The earliest fractures were fissures of but slight dislocation, therefore they were unaccompanied by impervious crushed granite and fault clay. The solutions and gases were thus accorded easy access to the wall rock at all points along the fissures. In later faults exhibiting much movement the circulating solutions were often closely confined within impervious fault-clay seams, and as a result unaltered wall rock is commonly found within a few feet of extensive ore bodies.

3. The regions traversed by the oldest fractures have been subjected to the action of vein-forming processes over a much longer period than regions adjacent to later fault veins.

There can be no question but that chemical agents have been greatly aided in their attack upon the granite and other rocks by dynamic processes. The breaking and crushing of the country rock in the Butte district has been on a profound scale. Dynamic agencies have not only furnished the avenues of travel within the rock for the gases or highly heated water, but, through accompanying crushing and mashing, the rocks are made more susceptible to attack by solutions. The true nature of the changes in the rocks, other than crushing, wrought by dynamic agencies, has been in a large measure obscured by subsequent metasomatic processes accompanying vein formation.

*Vein-Forming Processes.*—The alteration from this cause took place along and outward from fractures which acted as channels for the uprising solutions. These waters, of deep-seated origin, traversed not only the main trunk channels and associated fractures, but in highly fissured areas they followed also the joint planes, and penetrated, by slow stages no doubt, the whole mass of the granite in the more highly fissured areas. The remarkable activity of these thermal processes is evidenced by the development of the extensive altered zones accompanying the oldest fracture systems.

The initial stage of these changes, which were metasomatic in their nature, seems to have been the development of chlorite accompanied by the formation of pyrite. The uprising thermal waters or gases attacked first the iron silicates, augite, hornblende, and biotite, forming chlorite, epidote, secondary silica, and iron pyrite. Pyrite was developed also from the iron of the magnetite, the sulphur in this case, as in the former, being furnished by the attacking thermal waters, in

which it existed as hydrogen sulphide or as an alkaline sulphide. Plagioclase and orthoclase feldspars give way to continued attack, resulting in the formation of sericite and secondary silica. The result of the continued action of these metasomatic processes upon the granite has been the development of "pyritized" granite. The early-formed chlorite and epidote largely disappear; practically all of the iron of the original granite is converted into pyrite; and the feldspars are broken up into sericite and quartz. "Pyritized" granite, therefore, where representing the extreme development of the sericitic stage, consists principally of sericite, quartz, and disseminated pyrite.

Since the abundant fractures of the various systems form the medium through which thermal waters, or vein-forming solutions, traverse the body of the rock, the alteration necessarily proceeds, generally speaking, outward from these channels. The alteration is therefore usually more intense immediately along and within the fractures or fracture zones. Vein-forming processes have altered large areas of rock within and tributary to the Anaconda fissures; the Blue veins show less alteration of the wall rock than the Anaconda fissures, and similarly the Steward faults exhibit less alteration of the wall rock than the Blue veins, although in every case alteration has been intense within the fissures themselves.

Charles T. Kirk's[5] studies indicate that these various alteration phases are seldom free from the presence of kaolinite, a mineral of uncertain origin. It may be here stated that the relative quantity of kaolinite present in the deeper levels is insignificant when compared to the amount present near the surface, where it is known to result from the action of cold meteoric waters on pyritized or sericitized granite. There seem to be excellent reasons, to be later given, for believing that a large part of the present-day existing ground-water, even to great depths, is of meteoric origin, carrying appreciable quantities of iron sulphates and sulphuric acid. In the presence of such waters a slight development of kaolinite might be reasonably expected, while an important actual movement of these waters need not be inferred.

*Common Hydro-Metamorphism.*—Cold meteoric waters penetrating and passing downward through areas of unaltered Butte granite have effected but slight changes. The chemical activity of such solutions has apparently been slightly increased where preceded by crushing, and notably increased where preceded by rock alterations caused by vein-forming waters. The action of meteoric waters has, therefore,

[5] Kirk, C. T., Conditions of Mineralization in the Copper Veins at Butte, Montana, *Economic Geology*, vol. vii., No. 1, p 60 (Jan., 1912).

been most intense in veins and in the great alteration zones associated with the Anaconda fractures earlier described. The reasons for this increased activity in veins and in regions of altered granite are many. The oxidation of the pyrite of the veins and the disseminated pyrite of the altered granite results in the formation of iron sulphates and free sulphuric acid, which readily attack the already altered granite below, converting the sericite largely to kaolin. This process develops greater porosity in the granite, thus affording a more ready passage for the surface waters to greater depths.

The most noticeable effects upon the pyritized altered granite of descending meteoric waters have been kaolinization and chalcocitization, accompanied by greater porosity. In general, the ordinary meteoric waters have had no noticeable chemical effects upon the normal Butte granite at depths greater than the vertical thickness of the oxidized zone, which is seldom more than a few feet in unaltered granite.

*Depth Reached by Meteoric Waters.*—The maximum depths reached by meteoric waters in the Butte district cannot be definitely determined. It is probable that they have descended to greater depths than any yet reached by mine workings. A study of the physical and chemical changes that have taken place in the veins and country rock known to be due to the presence of waters of meteoric origin offers the only criteria for a partial solution of this problem.

The oxidized zone, resulting from the oxidizing influence of meteoric waters, varies in depth from 10 to 500 ft., an exceptional case in the Mountain View mine measuring over 900 ft. from the surface. The average depth, however, along the Anaconda vein is not more than 300 ft. That surface waters move downward to depths much greater than the lower limit of the oxidized zone is proved by the occurrence of an abundance of minerals in the veins and country rock known to be of secondary origin. Of these, chalcocite and kaolinite furnish the most reliable indicators of meteoric water influence. In veins and pyritized granite wall rock below the zone oxidation, the processes of chalcocitization and kaolinization go hand in hand, with the important difference, however, that while chalcocitization is always accompanied by kaolinization, the reverse is not necessarily true. Kaolinite, probably resulting from meteoric water action, is found at much greater depths than undoubted secondary chalcocite, and from this fact it is believed that down-seeping sulphate solutions continue to act on altered granite, forming kaolin, long after the last trace of copper has precipitated out as chalcocite in the zone of secondary chalcocite at higher levels.

Since primary chalcocite occurs in great abundance in the Butte veins, the mere presence of mineral chalcocite is not indicative of the presence of meteoric waters. The presence of the "sooty" chalcocite, however, which is known to be a product of descending meteoric waters, may be generally taken as proof of surface water action.

As a reliable means of determining more or less accurately the amount of vertical descent of meteoric waters, secondary chalcocite loses much of its importance when it is considered that the depth of the zone of secondary chalcocite is dependent upon many variable factors such as topography, mineralogical and physical character of the vein, etc. The depth of the zone of secondary chalcocite varies from 50 to 200 ft. in the fault veins to a maximum of 1,200 ft. in the veins of the Anaconda system. With the aid of a reliable method for distinguishing between primary and secondary chalcocite when in the massive form, the value of this mineral as a criterion of descending sulphide enrichment will be greatly increased.

If kaolinite is characteristically a product of meteoric water action, it is safe to conclude that surface waters have descended to depths greater than any yet reached by mine shafts. However, if, as Gregory[6] holds, kaolinite may also result from hydrothermal action at great depths, the small quantities of kaolinite present in the deep Butte levels may not be properly regarded as proof of the presence of waters of meteoric origin.

*Oxidation.*—Oxidizing processes acting upon veins and altered granite tend to transform the sulphides into oxides and native metals. The results of these processes acting on Butte ores and rocks are briefly described in the chapter following.

## Superficial Alteration of the Butte Veins.

### *Outcrops of Copper Veins.*

The mantle of "wash," or débris from disintegration and weathering, often masks the intersection of the veins with the surface of the bed rock. This condition is especially prevalent in areas of intense granite alteration, notably eastward from the Parrot and Moonlight and in the vicinity of the Diamond mine. The thickness of the surface wash varies from 2 to 10 ft., although an exceptional thickness of from 200 to 400 ft. occurs in the valley in the vicinity of the Pittsmont smelter. (See Plate IV.) This unusual thickness, however, has probably resulted from certain conditions brought about by the Continental fault movement, rather than from natural processes of erosion and decay.

[6] Gregory, J. W., Criteria of Downward Sulphide Enrichment, *Economic Geology*, vol. v., No. 7, p. 680 (Oct.-Nov., 1910).

A number of copper veins, notably the Anaconda, Syndicate, and Colusa, have more prominent outcrops characterized by unusual amounts of strong iron-stained quartz and vein matter projecting well above the wash. As a general rule, however, the position of the copper vein outcrops, especially within the alteration zones, cannot be accurately determined without the aid of shafts or tests pits, although the general location and direction may be known by the presence of detached fragments of the veins, or "float." The degree of prominence of vein outcrops appears to be governed largely by their mineralogical content and physical character, and also by the nature of the inclosing wall rock.

The outcrop of a typical copper vein of the Anaconda system is marked by altered granite, quartz, and oxides of iron. There may also be present a small band of oxidized clay or crushed granite within or along the vein. The granite included within the vein boundaries or adjacent to the vein is irregularly seamed and stained with iron oxides. The quartz commonly exhibits the well-known honeycomb structure.

The outcrop of a fault vein of the Blue or Steward systems consists of a slightly iron-stained mass of soft, crushed and altered granite with one or more seams of fault clay of a blue-gray or yellowish color. Where a fault-vein ore shoot outcrops, the composition is very similar to that of the Anaconda veins, with the possible addition of one or more well-defined bands of fault clay.

Almost without exception the copper veins are practically barren of copper at the outcrop and in the zone of oxidation below. There are no visible copper minerals, and it rarely happens that an assay of the oxidized material yields more than a trace of copper. (Compare Table I., following.) Some exceptions, however, may be noted. Small quantities of carbonates and oxides of copper were found in the outcrop of the Gagnon-Parrot vein, also in the Syndicate vein and others. The discovery shaft of the Mountain Chief claim, located on the southeasterly extension of the Jessie vein, was sunk in a rich body of red oxide of copper, running high in silver. The oxidized zone, however, proved to be very shallow in vertical extent, the rich oxide ore changing abruptly to chalcopyrite at less than 30 ft. in depth. (See Plate V.)

Taking a broad view of the entire copper producing area, it may be said with emphasis that there is but little, if any, evidence of a positive character to be found in the outcrops or the oxidized zones of the Butte veins to indicate the existence of copper in commercial quantities at greater depths.

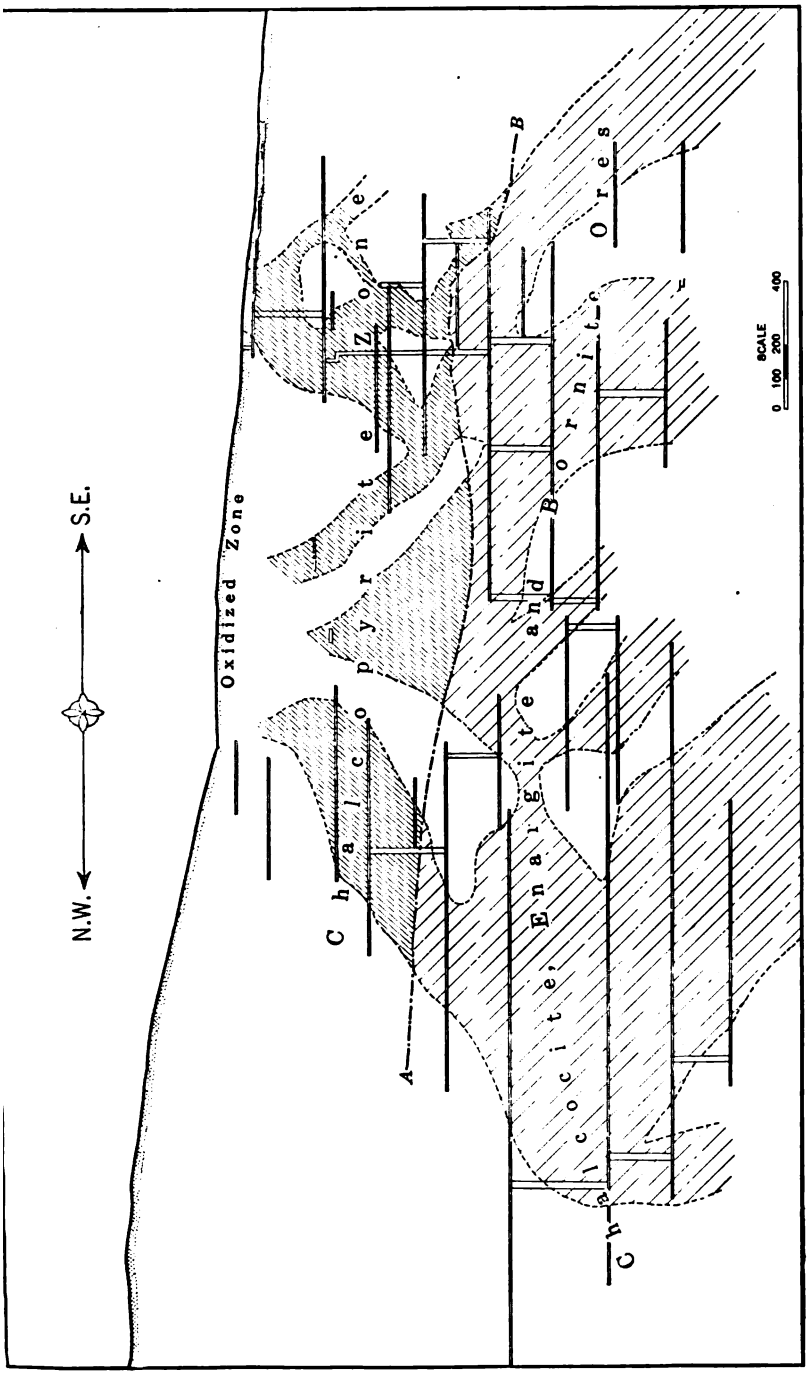



PLATE V.—LONGITUDINAL PROJECTION OF A PORTION OF THE JESSIE VEIN, SHOWING ARRANGEMENT OF ORE SHOOTS.

Line *A–B* marks approximately the dividing line between chalcopyrite and rich chalcocite-enargite ores below. Both above and below line *A–B* quartz and pyrite are abundant as gangue minerals. Above line *A–B* rhodochrosite is commonly present.

The following table gives the analyses of a series of samples taken from oxidized portions of producing copper veins:

TABLE I.—*Analyses.*

| | I. | II. | III. | IV. | V. | VI. | VII. | VIII. | IX. | X. | XI | XII. | XIII. |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| $SiO_2$ | 48.0 | 84.2 | 65.5 | 41.1 | 79.2 | 67.4 | 71.7 | 68.3 | 73.3 | 26.6 | 58.4 | 51.9 | [illegible] |
| Fe | 4,6 | 3.3 | 15.0 | 30.6 | 5.9 | 12.3 | 8.9 | 10.2 | 5.4 | 19.8 | 12.5 | 16.5 | [illegible] |
| $Al_2O_3$ | 25.3 | 5.9 | 4.3 | 5.4 | 5.5 | 7.4 | 4.1 | 6.4 | 11.5 | 25.9 | 5.5 | 8.5 | [illegible] |
| Cu | 0.05 | tr. | 0.12 | 0.13 | tr. | 0.15 | 0.05 | 0.05 | tr. | 0.05 | tr. | 0.06 | [illegible] |
| $As_2O_3$ | 1.71 | 0.14 | 0.79 | 0.52 | 0.22 | 0.38 | 0.32 | 0.56 | 0.38 | 0.88 | 5.24 | 1.2 | [illegible] |
| Zn | tr. | tr. | tr. | tr. | tr. | tr. | tr. | tr. | tr. | tr. | tr. | tr. | tr. |
| Ag, oz. per ton | 17.7 | 3.7 | 5.8 | 1.0 | 4.1 | 3.1 | 5.7 | 10.9 | 1.2 | 1.1 | 2.8 | 4.2 | [illegible] |
| Au, oz. per ton | 0.05 | 0.008 | 0.005 | 0.005 | 0.07 | 0.01 | 0.015 | 0.02 | 0.01 | 5.007 | 0.03 | 0.01 | [illegible] |

I. Lloyd tunnel 250 ft. below outcrop of vein.

II. Well-defined quartz vein 5 ft. wide; sample taken 10 ft. below outcrop.

III. Oxidized quartz vein of Anaconda system; sample taken 15 ft. below outcrop.

IV. Soft dark red oxides along fault clay; sample taken 15 ft. below outcrop.

V. Sample of 4-ft. siliceous vein taken 20 ft. below outcrop.

VI. Oxidized enargite-chalcocite ore taken 2 ft. above sulphides and 400 ft. below outcrop.

VII. Oxidized vein; sample taken 20 ft. above sulphides, St. Lawrence mine, and 100 ft. below outcrop.

VIII. Anaconda vein, Mountain View mine; much enargite in ore below; sample taken 500 ft. below outcrop.

IX. Anaconda system, South vein; taken above secondary chalcocite ore, 260 ft. from outcrop.

X. Soft oxidized clayey vein, 20 ft. below outcrop.

XI. Blue vein system; taken above enargite ore, 360 ft. below outcrop.

XII. From fault vein enargite ore, Mountain View mine, 450 ft. below outcrop.

XIII. From south vein, Mountain View mine, 600 ft. below outcrop.

The above samples are from veins lying well within the alteration zone in the vicinity of the Rarus mine. The most significant feature is the high content of $As_2O_3$, indicating that not all of the arsenic is extracted from the vein in the process of oxidation. Further investigations are now under way to determine if possible the relation between the arsenic present in the oxidized to the total arsenic in the unoxidized vein below.

*Outcrops of Manganese-Silver Veins.*

In marked contrast to the ill-defined copper-vein outcrops are the bold projecting outcrops of the manganese-silver veins, which may be traced for hundreds or even thousands of feet over the surface. Many of them, like the Rainbow, Silver Lick, Ancient, Emma, and numberless others, project in bold relief from 1 to 10 ft. above the ground surface. Where they do not project above the wash, their presence is generally indicated by an abundance of characteristic float rock.

The outcropping portion or the oxidized zone of a typical manganese-silver vein consists chiefly of quartz and oxides of manganese and iron. As an original mineral of the unoxidized vein, pyrite is not as abundant as in the copper veins, nor is it so universally present.

The Black Rock vein, an important producer of zinc, is a member of the manganese-silver vein series belonging, it is thought, to the Anaconda fracture system. The developments in the Black Rock vein show that many of the great sphalerite ore bodies first appear several hundred feet below the surface, and that the upper portion of the vein is composed principally of quartz and rhodochrosite. The outcrop is mainly quartz and oxide of manganese.

In the outcrops and oxidized zone of the zinc-producing veins the zinc is entirely removed in the processes of oxidation, and not more than a trace remains to indicate the presence of zinc at greater depths.

*Oxidation and Disintegration of the Granite.*

In the great zones of altered granite associated with the copper veins there are but few, if any, actual outcroppings of solid granite or bed rock, the latter being effectually concealed by the covering of disintegrated rock. This surface wash, or débris, consists of iron-stained altered granite in the form of angular fragments, and finer incoherent grains, together with fragmentary oxidized vein quartz, aplite, and quartz-porphyry. The clusters of rounded boulders so characteristic of the weathering of normal granite are entirely absent.

The unaltered granite is apparently more resistant to the action of atmospheric agencies than the altered granite, and owing to this fact veins or fault outcrops found within normal granite areas are conspicuous as belts or zones of finely disintegrated rock, readily traceable over the surface. Generally in the manganese-silver vein area and all of that portion of the Butte district lying to the north of the Speculator, Tuolumne, and Corra mines, there is but little alteration of the granite, excepting within and along the vein or fault fissures. Under these conditions the positions of the veins and faults are indicated by smooth surfaces, while the intervening areas of normal granite are marked by

the presence of rounded granite boulders, a feature characteristic of normal granite weathering over the whole Boulder granite area.

*Zone of Oxidation.*

By the expression "zone of oxidation" is meant those portions of the veins and country rock which have been oxidized through the action of descending ground-waters. It embraces in vertical extent all of the veins and country rock lying above the irregular boundary plane marking the upper limit of the sulphides. (See Plate IV.) The depth of the zone of oxidation in the Butte district is extremely variable in different localities, and in exceptional instances wide variations occur within small areas. For example, in the Mountain View mine, No. 4 vein, the first sulphide ore was met at 250 ft. below the surface, or 150 ft. above the first level, while at a point 400 ft. south of the main shaft the South vein is oxidized for a short distance along its strike to a depth of over 900 ft. below the surface. Extreme local variations of this nature, however, are not the rule. The depth of the zone of oxidation in a particular locality is largely dependent upon the character of the veins and the inclosing country rock (see Plate II), to a greater extent, in fact, than upon the topographic features. The deepest oxidation is found in the great zones of altered granite described on pages 1550-1551, inclusive, where it runs from 100 to 400 ft., averaging in the neighborhood of 250 ft. (See Plate IV.) Under similar conditions of granite alteration the heavily mineralized veins of the Anaconda system show deeper oxidation than the fault veins, owing to the greater permeability of the quartz vein to downward-seeping waters and to the greater abundance of pyrite, the source of the all-important oxidizers, ferric sulphate and sulphuric acid. In regions unaffected by the widespread hydro-thermal processes accompanying the early vein formation, the depth of oxidation along the veins of the Anaconda system averages about 75 ft., and that of the fault veins 20 ft., with frequent variations in both classes of veins.

Examined from the surface downward the oxidized portion of a copper vein will show but little change in physical character and mineral composition between the outcrop and the sulphide ore below. The line of separation marking the change from oxidized to sulphide ore is extremely sharp. Above this contact plane there is no mixture of oxides and sulphides, the oxidation is complete. The entire change, as shown at any single cross-section of a vein, takes place within a vertical distance of a few feet. Frequently, near the upper limits of the sulphide ore, the proximity of the zone of oxidation is indicated by slight changes in the relative abundance of certain secondary minerals,

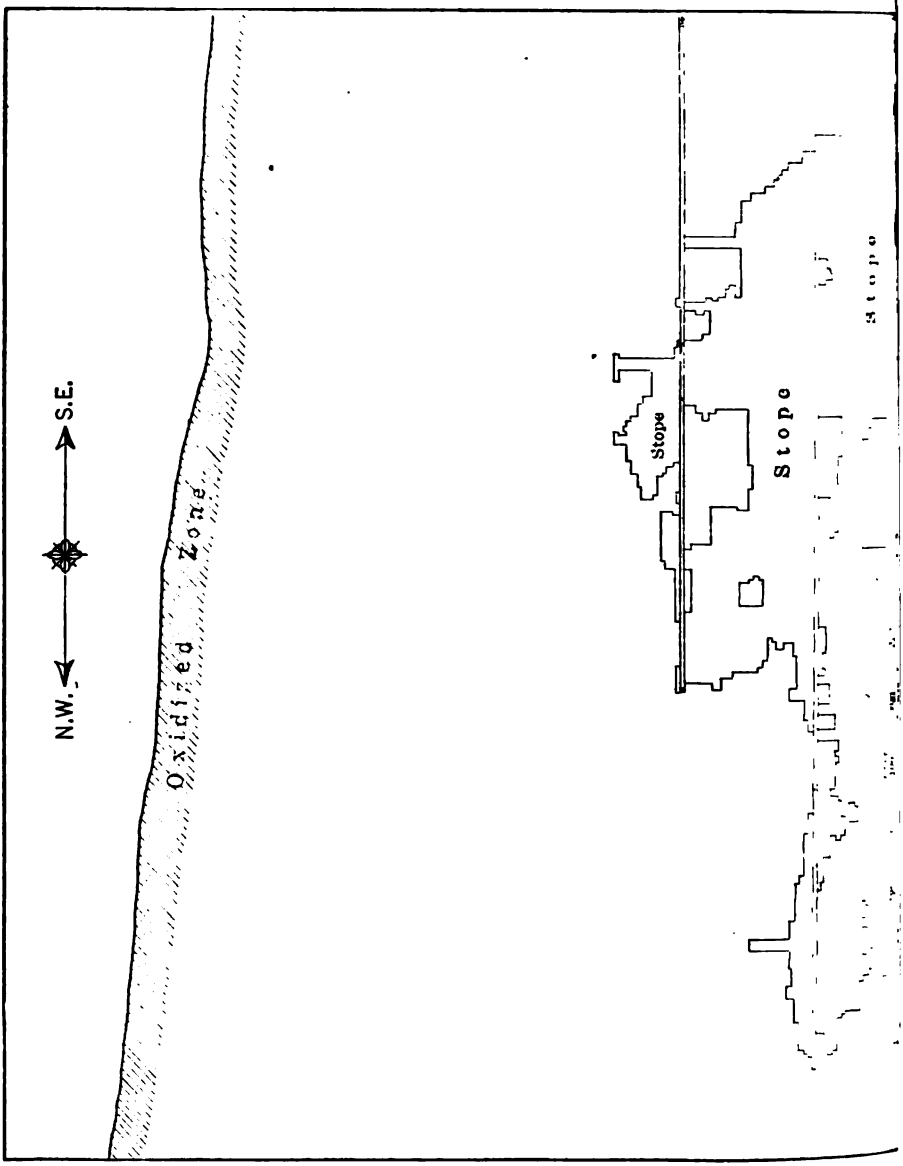
but in the case of the oxidized vein there is seldom any change indicating nearby sulphides.

In the quartz-pyrite veins of the Anaconda system within the copper-producing area, the appearance of sulphides at the lower limit of the zone of oxidation almost always means the beginning of commercial copper ore. This feature of the heavily mineralized veins is in striking contrast to the fault veins of the Blue and Steward systems, where the ore shoots are often separated vertically by hundreds of feet of barren unoxidized vein matter which begins at the base of the shallow oxidized zone below outcrop. (See Figs. 6 and 6A.)

Unusual conditions of oxidation are found in the Bullwhacker, Butte & Duluth, and adjoining properties, situated along the line of the Continental fault outcrop east of the Pitsmont mine. It appears that the unoxidized granite of that section carries a small percentage of disseminated chalcopyrite. The shearing and crushing along the fault zone caused a rapid disintegration of the granite under the action of atmospheric agencies, during which process the copper of the chalcopyrite was oxidized and carried downward by surface waters and redeposited as chrysocolla, red oxide, or as a carbonate. The granite exhibits but slight alteration. The resulitng ore is therefore a green-stained granite with the more important accumulations of copper carbonate along the cracks and joint planes. As mined the ore runs from 1.5 to 4 per cent. The absence of pyrite in the original disintegrating granite explains the formation of ore of this character. In the process of oxidation of chalcopyrite, insufficient ferric sulphate or sulphuric acid is formed to prevent the formation of the insoluble oxides, silicates, and carbonates of copper. Under such conditions copper migrates but short distances.

## Ground-Water.

Extensive mine developments have disclosed many interesting facts concerning the distribution of the underground waters in the veins and rocks of the Butte district. In areas of intense rock alterations there is approximate saturation of the rocks and veins, a feature in striking contrast to areas of normal granite, where alternate wet and dry zones are the rule. In the great zones of rock alteration not only are the veins wet, but the intervening country carries water as well. The granite in one part of a cross-cut through an area of altered granite may be wetter than at other points, but no part of it will be absolutely dry. The inclosed veins and faults may or may not be wetter than the granite, the more important trunk channels not being readily recognized as in the zones of unaltered granite.


N.W.
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Oxidized Zone
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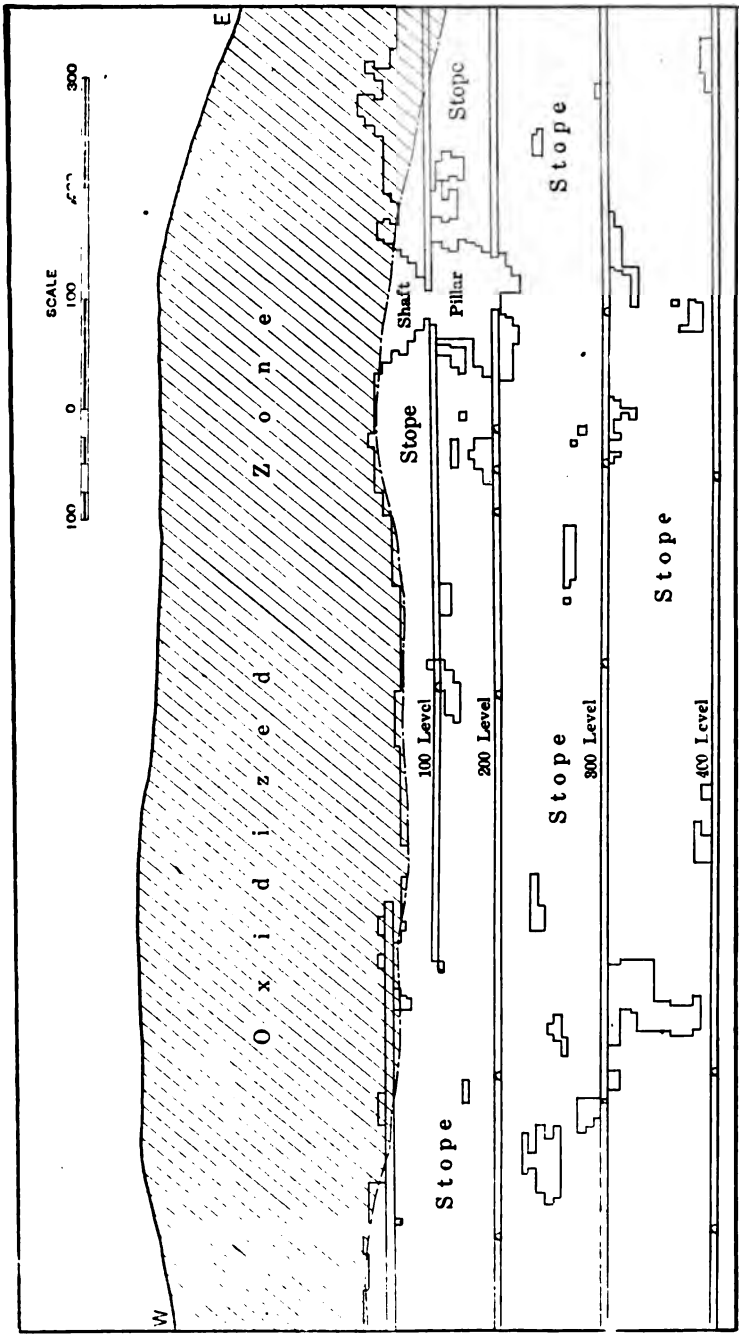


FIG. 6A.—VERTICAL PROJECTION OF A PORTION OF PENNSYLVANIA VEIN, A MEMBER OF THE ANACONDA SYSTEM, SHOWING THE RELATION BETWEEN THE OXIDIZED ZONE AND THE COMMERCIAL ORE BELOW.

In this connection it is not implied that in areas of altered granite water moves freely through the actual rock. It is believed, however, that the water does follow rather freely, comparatively speaking, the numerous joint planes, from which it spreads outward into the altered solid rock much more readily than in the case of unaltered granite. The saturated condition of the altered granite is therefore due to the ease with which the water finds its way along every insignificant joint plane or crack. The altered areas are more fractured than the areas of normal granite, because they were, in the first place, zones of the most intense early fissuring. This early fissuring being followed by marked alteration, a weakening of the rock naturally resulted. Because of this weakening, the faults of later systems commonly spread out over wider zones and were accompanied by greater crushing than in areas of normal granite.

In areas of normal or unaltered granite the ground-water is confined to the definite channels or water zones afforded by shear zones, cracks, faults, or veins. Generally the unaltered granite between the main circulation channels is dry. From these facts it appears that the unaltered granite is quite impervious to ordinary underground waters under normal conditions of temperature and pressure.

In the early days of Butte mining the water level was encountered in the shafts in the region of the contact between the oxidized and sulphide ores. This was true, however, only in areas of altered granite or where the opening was made within a vein. A shaft sunk in the normal unaltered granite between faults and veins, perchance missing a water course, might extend to great depths as a dry shaft. On the other hand, a shaft put down in the altered granite areas became a wet shaft as soon as the general water level was reached, whether a vein was encountered or not, the whole mass of the altered rock being practically saturated.

It has been repeatedly observed that the well-developed fault fissures do not serve as the most important channels for the movement of the present day ground-waters, although the influence of the fault movements in determining the distribution of these waters has been very marked. For example, the Bell, Rarus, and Middle faults seldom contain water in unusual quantities; on the contrary, they are often dry. They have broken and cracked older veins, making them quite permeable to solutions. Of much greater importance as water carriers are the veins of the Anaconda system, fissures, cracks, and shear zones of relatively slight displacement. The Anaconda veins are frequently of a porous nature and contain many cavities, vugs,

and water courses. They are especially good water carriers where cracked or broken by late fault movements.

The ore-bearing faults of the Blue and Steward systems are alternately wet and dry along the strike. It is believed that originally the uprising solutions spread out along these fissures throughout the entire length, but continued earth movements along the fissure planes developed zones of impervious clay and crushed granite, and the circulating solutions were directed along the more open zones now occupied by the ore shoots. The dry zones have remained so, and the waters now found in association with the fault veins are encountered in the more porous ore shoots, or in cracks and broken granite of the walls of the fissure. Frequently the ore shoots contain no water, and it is probable that they have been dry ever since the close of the period of thermal water activity, when the ores were deposited.

The Mountain View breccia veins are practically impervious to water. It appears that they were never connected with active uprising waters, although in part, at least, they are older than the ores of the Steward fault veins. The breccia filling of these cracks is often horizontally bedded, showing characteristic water action. It is probable, therefore, that during the period of filling, the fissures were filled with water practically stagnant.

### *Source of the Ground-Water.*

There are two possible sources of the water now found in the rocks and veins of the Butte district: (*a*) uprising waters of deep-seated origin, and (*b*) meteoric water derived principally from atmospheric precipitation in the form of rain or snow.

*Primary or Juvenile Waters.*—The primary ores of the district were deposited from ascending waters presumably of deep-seated origin. The deposition of primary minerals and ores apparently stopped some time prior to the Rarus faulting, but it is not known how much longer ascending waters continued to traverse the channels of circulation after ore deposition ceased. It is reasonable to assume that during the period of primary ore formation the ascending waters were very active and plentiful and they constituted the bulk of the waters then occupying the veins and adjacent granite. A rapid waning in the activity of the ascending waters took place, no doubt, at the close of the period of primary ore deposition, after which time downward-seeping water of meteoric origin more and more predominated, until at the present day it is extremely doubtful whether any appreciable quantities of the mine waters found above the 3,000-ft. level have been derived directly from deep-seated sources.

*Meteoric Waters.*—The annual precipitation in the district varies from 12 to 20 in. Of this amount a certain part evaporates, some finds its way to the streams as run-off, and the remainder sinks into the earth and by means of cracks, veins, porous rocks, etc., reaches to considerable depths, forming the ordinary ground-water. On account of the extreme porosity of the surface wash covering the district, an unusually large proportion of the rainfall sinks to the contact between the wash and the solid rock below. This factor is offset in a large measure by the steepness of the slopes, permitting a proportionately large run-off.

*Underground Circulation.*—Actual underground observations of the action of water in the veins and rocks as disclosed by mine openings are not especially important in determining the immediate source of the ground-water. Where an opening such as a cross-cut or drift penetrates undrained water-bearing rocks or veins a pervasive dripping from the top or roof of the opening, due to gravity, is certain to follow, regardless of the original source of supply. Irregular cracks and water courses, of which there are many, both in veins and in the hard country rock, may appear in the top, side, or bottom of the opening, exhibiting either a downward or an upward flow of water according to whether the water channel first appears in the bottom or in the back of the opening.

The flow of water encountered in a new bottom level decreases rapidly, but does not entirely cease until additional openings are made at greater depths in the immediate vicinity. While this observed condition indicates a water-soaked condition of the rock and included veins or fissures, it does not afford a reliable clue concerning the immediate source of the water. Whether the source of supply is from above or below, the flow from the opening will be maintained until a certain rock volume surrounding the opening is drained. The rate of flow, however, will vary from a maximum when the opening is first made, to a certain minimum after the opening is made, the length of time elapsing depending on many factors, such as volume of ground to be drained, porosity of the rock, etc.

The fact that the water flow decreases rapidly from newly opened ground indicates that the normal ground-water of the district is extremely sluggish, if not practically stagnant, particularly in the deep levels. In higher levels incontrovertible proof of a certain amount of ground-water activity is afforded by the zone of oxidation and, in slightly deeper zones, by the presence of sooty chalcocite, an undoubted product of descending water action. After the altered rocks and veins of the great altered zones and the veins of the unaltered gran-

ite areas become waterladen, it is certain that a small supply only need be added to maintain saturation. The fact must be kept clearly in mind that the water flow encountered by mine openings penetrating new and undrained country in the Butte district does not represent the rate at which water is being supplied, but instead merely the rush of water into the opening caused principally by force of gravity.

*Ground-Water Largely of Meteoric Origin.*—The facts and conditions heretofore outlined seem to indicate a meteoric origin for the greater part of the present mine waters of the district. The frequent occurrence of dry ore shoots in deep levels, known to have been saturated at a former time with ascending water, indicates that the period of intense activity of uprising water has long ago ceased. In certain instances, such as in the Tramway deep levels, where unusually high water temperatures are noted, one is led to suspect a possible dying ember of a former active hot-water circulation. There are no active hot springs and no waters have been encountered in the mines to which one might ascribe with any degree of certainty a deep-seated origin. The series of temperature observations given below, however, indicate that the waters encountered in areas of intense granite alteration are slightly hotter than waters in unaltered areas at corresponding elevations. The waters of the Tramway mine traverse intensely altered granite zones, and they are, furthermore, in close proximity to certain of the high-grade primary chalcocite ore shoots of the No. 16 vein belonging to the Steward system, known to be the most recent of the primary copper ore bodies.

TABLE II.—*Temperature Observations on Butte Mine Waters.*

| Mine. | Mine Level. | Elevation above Sea Level. | Temperature Degree F. | Remarks. |
|---|---|---|---|---|
| Tramway | 2,800 | 3,400 | 102 | Water encountered in shaft station bottom level. |
| Tramway | 2,200 | 3,571 | 100 | Water in drift newly opened. |
| Tramway | 2,000 | 3,770 | 104 | Drift on north-south vein near No. 16 fault vein. |
| Tramway | 2,000 | 3,770 | 100 | Water from cross-cut. |
| Tramway | 1,700 | 3,917 | 94½ | Drift on vein, new work. |
| St. Lawrence | 2,800 | 3,395 | 97 | Average of four observations at different points bottom level. |
| Pennsylvania | 1,800 | 3,940 | 92 | Water from diamond-drill hole in altered granite area. |
| Pennsylvania | 1,800 | 3,940 | 90 | Water from diamond-drill hole partly in altered granite area. |
| Pennsylvania | 1,800 | 3,940 | 87 | Water from diamond drill hole outside of altered granite area. |
| Pennsylvania | 1,800 | 3,940 | 94 | Water in face of drift on No. 1 vein in altered granite area. |
| Gagnon | 1,900 | 3,955 | 83 | Average of 8 observations in new cross-cut in normal granite area. |
| Original | 2,800 | 3,400 | 99 | Water in face of drift on Gagnon-Original vein. |
| Original | 2,800 | 3,393 | 93 | Water in face of drift on Gagnon-Original vein. |
| Diamond | 2,800 | 3,393 | 95 | General average of observations, bottom level. |
| Diamond | 2,600 | 3,590 | 94 | General average of observations. |
| Badger | 1,800 | 4,430 | 76 | New drift in new territory on east-west vein. |
| Badger | 1,800 | 4,430 | 74 | Water running from diamond-drill hole in area of normal granite. |

The Tramway, St. Lawrence, and Pennsylvania mines are within the great altered zones of Anaconda hill. The Gagnon and Original are in an area of normal granite, although a prominent altered belt is associated with the main Gagnon-Original vein. (See Plate III.) The Diamond area shows a less general altered condition than the main altered zone. In the Badger mine granite alteration is prominent only within and along the veins.

## Mineralogy of the Veins.

It is not intended in this chapter to describe in detail all of the minerals found in the district, but to note only those of importance in connection with the ore deposits.

The important copper minerals of the Butte ores, named in order of their relative abundance, are; chalcocite, enargite, bornite, chalcopyrite, tetrahedrite, tennantite, and covellite. Of the oxidized products, chrysocolla, malachite, cuprite, and native copper are the most common, but taken as a whole they have contributed but little to the total copper produced. The gangue minerals are principally quartz and pyrite, occurring in about equal amounts. Sphalerite is abundant in the border zones of the district, in some instances being the predominating constituent of the vein filling, as in the Black Rock mine, where zinc ore is the chief product.

The less common minerals found in the copper veins are hübnerite, galena, barite, rhodochrosite, fluorite, and calcite. Passing from the copper to the silver veins, minerals characteristic of the silver vein area, rhodochrosite, galena, and barite become more and more abundant, the quartz and rhodochrosite finally forming the chief constituents of the gangue, although pyrite and sphalerite are present in notable amounts.

The great bulk of waste matter in the ore as mined and sent to the reduction works is altered granite, which forms from 50 to 70 per cent. by weight of the ore. The occurrence and association of the common minerals of the veins will be described in greater detail below.

### *Chalcocite.*

Chalcocite is the most important copper mineral of the Butte district. As a common constituent of the copper ores it has been the source of not less than 60 per cent. of the total copper produced to date. It occurs in ore-producing veins of all ages and is abundant at all levels from the upper limit of the sulphides down to the greatest depths yet reached, extending, in many instances, more than 3,000 ft. below the surface.

Broadly speaking, the chalcocite of the Butte veins occurs in three

distinct forms: (*a*) as "sooty" glance, so-called, in which form it appears as a dull black coating on iron pyrite or other sulphides, or frequently developing a black amorphous powdery substance resembling "soot," where it has resulted from the replacement of the precipitant sulphide; (*b*) as massive steel-gray chalcocite; and (*c*) in the form of crystals. Of the three varieties named, the first two are of great economic importance. Chalcocite crystals are of frequent occurrence but they have no commercial significance.

*Sooty Chalcocite.*—Sooty chalcocite is widely distributed as an ore-forming mineral, but it is confined particularly to those portions of the veins lying immediately below the zone of oxidation. (See Plates II. and IV.) This "sooty" glance zone, or belt, is the "chalcocitization zone" of Lindgren at Morenci. In Butte it varies in vertical extent between wide limits, depending primarily on the mineralogical and physical character of the veins and the depth of the zone of oxidation. The veins of the Anaconda system, or the quartz-pyrite series, are deeply oxidized as a rule, and in these veins the sooty chalcocite reaches its greatest development.

In the fault veins the zone of oxidation is usually of but slight vertical extent, and since the primary zones seldom extend upward to the surface, the development of sooty chalcocite is of but little importance. The barren portions of fault veins consisting of clay and crushed granite, occupying long stretches on the strike of the vein between ore shoots, do not offer an adequate source for copper to form sooty glance ores in quantity below the zone of oxidation. The majority of the big chalcocite-enargite shoots of the fault veins do not extend upward to within 500 ft. of the lower limit of the oxidized zone. Where fault-vein ore shoots reach the surface they are found to be oxidized, and they are accompanied by the development of sooty chalcocite ores within the boundaries of the primary-ore shoot similar in every respect to the corresponding secondary enrichment belt of the quartz-pyrite zeins.

In the veins of the Anaconda system, particularly in the great altered granite belts, sooty chalcocite is very important commercially. It is found usually as a coating, or as a partial replacement of pyrite, and, less commonly, sphalerite, enargite, and chalcopyrite. It replaces, completely or in part, the pyrite of the veins, also the stringers, veinlets, and fine disseminations of pyrite in the altered granite, not only within and along the veins, but in the intervening altered country rock. The chalcocitization of the altered pyritized granite lying between the more important veins has resulted in the development of a low-grade material similar in general character to the disseminated porphyry

ores of the Southwest, although in the Butte district such material is seldom rich enough to mine except when adjacent to the well-defined veins.

In the veins of the Anaconda, or quartz-pyrite, series (see Plates II. and IV.) the sooty chalcocite occupies a zone from 200 to 1,000 ft. in vertical extent, measured from the bottom of the zone of oxidation. The lower limit of this chalcocitization zone is ill-defined and extremely irregular, which is in marked contrast to the sharply defined lower limit of the zone of oxidation. The mineral is most abundant in the highest levels of the sulphide zone. As depth is gained it becomes less prominent, finally disappearing. On the accompanying maps, Plates II. and IV., the lines marking the lower limit of the sooty chalcocite zone are intended to mark approximately the elevation below which but little, if any, sooty chalcocite has been observed.

*Massive Chalcocite.*—Chalcocite in massive form is of wide distribution in the Butte veins, occurring in veins and masses of great purity. In color it is steel gray, exhibiting the usual conchoidal fracture. In the veins of the Anaconda system it is found in great abundance at all levels from the bottom of the oxidized zone to the greatest depth reached by underground workings. In the ore shoots of the Blue vein system chalcocite ores seldom extend upward to within 500 ft. of the surface. In the later Steward fault veins the rich chalcocite ore bodies were first encountered at a depth of from 1,000 to 1,200 ft.

As an ore mineral chalcocite is found filling fractures in quartz, pyrite, or other older vein minerals, and in irregular veinlets, seams, and stringers in altered granite within or along the veins. Chalcocite plays an important part in the formation of the great stock-work ore bodies of the Mountain View, West Colusa, and Leonard mines. In these properties the granite has been highly altered and much fissured. (See Plate I.) The shattering and alteration was followed by mineralization of the fissures and joint planes, chiefly by pyrite, quartz, chalcocite, and enargite, with occasional covellite. In the massive quartz-pyrite veins, such as the Anaconda and Syndicate veins, chalcocite is commonly found associated with quartz, pyrite, bornite, and enargite filling fractures or vugs in the earlier vein filling. Both in the older quartz-pyrite veins and in the later fault veins massive chalcocite is found in close association with pyrite, bornite, enargite, and quartz, forming an extremely complex mineral aggregate.

In the ore shoots of the Blue vein and Steward vein series, chalcocite frequently occurs as fine disseminations, seams, and splotchy masses mixed with iron pyrite, quartz, enargite, and bornite. It

commonly replaces the attrition clay and altered granite of these veins, often retaining the grooves, striations, and other markings characteristic of the original clay. Bodies of massive chalcocite are found in the ore shoots of No. 16 vein in the Rarus and Tramway mines. In these high-grade shoots, which are regarded as primary, enargite, pyrite, and quartz are also present intimately associated with the chalcocite. The massive chalcocite of the Blue and Steward veins and the major part of the chalcocite lying below the zone of sooty chalcocite in the Anaconda veins are believed to be of primary origin.

*Enargite.*

Enargite is of wide distribution both vertically and laterally in the Butte veins. It has been the source of from 25 to 40 per cent. of the copper output of the district. It is particularly abundant in the eastern portion of the district in the region of the Pennsylvania, Rarus, West Colusa, and Leonard mines. In the deep levels of the Gagnon and Original mines enargite is the predominating copper mineral and forms wonderfully rich and persistent ore bodies. Enargite is largely a product of a comparatively old mineralization period. It is found in copper-producing veins of all ages from the earliest up to and including the No. 16 fault veins of the Rarus and Tramway mines. Enargite is found in great abundance in the higher as well as in the lower levels of the Anaconda vein and other important veins of the oldest vein system, contrary to a former published statement [7] that it did not appear in the Anaconda vein higher than the 2,000-ft. level.

As a mineral of the veins, enargite usually occurs in interlocking aggregates of imperfectly outlined crystals in size from 0.25 to 1 in. across forming a solid mass, the individual growths always exhibiting characteristic cleavage surfaces. Beautifully formed crystals are frequently found from $\frac{1}{8}$ to 1 in. in length, but the larger sizes are usually less perfect than the smaller ones.

The more massive enargite is found occupying fractures in older quartz-pryite veins, or replacing the altered granite of the vein in the form of veins and stringers and as splotchy masses along the cracks and joint planes of the granite. It is often more or less intimately mixed with either pyrite, quartz, chalcocite, or bornite, and less commonly with covellite. Intimate mixtures of enargite, pyrite, and barite crystals contemporaneous in origin are frequently seen lining vein cavities.

Enargite is chiefly a product of primary ore deposition. In the

[7] Weed, W. H., Geology and Ore Deposits of the Butte District, Montana, *Professional Paper No. 74, U. S. Geological Survey*, pp. 77 and 107 (1912).

Shannon Windlass, Enargite, South, and other veins of the eastern district it is found in great abundance in the upper mine levels, the greatest development occurring immediately below the lower limit of the oxidized zone and extending downward for from 400 to 600 ft. Chemical analyses show appreciable amounts of arsenic (see Table I, p. 1558) in the oxidized portions of these rich enargite veins, but no marked development of chalcocite occurs in the sulphide zone immediately underlying these gossans. It seems not unreasonable under such conditions to assume that a part, at least, of the enargite is of secondary origin.

*Bornite.*

Bornite is one the commonest of the copper sulphide minerals. It is almost universally present in the copper ores, associated with chalcocite, enargite, chalcopyrite, and other sulphides, but usually in subordinate amounts. As a source of copper it is especially important in the Steward and Original mines and in the mines of the North Butte section. The bornite occurrence in Butte is particularly interesting from the fact that while it seldom forms the chief copper mineral of the ore, it is rarely absent even in hand specimens.

In physical character the bornite of the Butte vein exhibits on fresh surface the characteristic horse-flesh color, tarnishing rapidly on exposure to a blue or green color. It is a frequent associate of chalcocite, chalcopyrite, enargite, and other sulphides.

Bornite occurs in copper veins of all ages at all levels from the oxidized zone to the greatest depths yet reached. It is largely a product of primary ore deposition along with enargite, chalcocite, and other associated primary minerals.

*Chalcopyrite.*

Although described in former years as the chief copper mineral of the primary ores of the Butte copper veins, chalcopyrite has proved to be of comparatively small importance as a source of copper. It is characteristically a mineral of the border zone of the central copper area. Chalcopyrite is practically unknown in the Anaconda, Mountain View, and other mines of the eastern part of the district. It is common in the extreme west end of the Gagnon mine, in the Lexington, West Gray Rock and in the veins of the Speculator mine. It is of frequent occurrence in the Elm Orlu mine, Alice, and other silver properties. Chalcopyrite is commonly a product of late vein-forming action, being frequently found in cracks, vugs, and cavities within older vein filling or as a thin coating or replacement of older sulphide minerals, particularly sphalerite, pyrite, enargite, and covellite. It has been noted in the form of small imperfect crystals in rare in-

stances, occurring in cracks in older vein filling and finely disseminated throughout the unaltered granite, particularly in the Altona and Amazon shafts in the eastern part of the district. The unusual occurrence in the Jessie vein of chalcopyrite-pyrite vein filling capping rich chalcocite-enargite ores at greater depths is of special interest and will be further described in the discussion of the formation of chalcocite. (See p. 1620.)

In vertical distribution chalcopyrite is found at all levels. In certain localities, as in the Gagnon and Steward mines, it is more abundant in the deep levels than in the upper levels, in marked contrast to the chalcopyrite ores of the Jessie above mentioned.

Chalcopyrite in the Butte veins is largely a product of primary mineralization, although in certain instances it is believed to be possibly of secondary origin. Frequently it is the most recent mineral of the veiu in which it is found, occurring as fine crystals in cracks or as a replacement of enargite, covellite, or chalcocite.

### *Covellite.*

Covellite occurs as massive mineral and more rarely as crystals of the characteristic indigo-blue color in certain veins of the northern and eastern parts of the district. It was found in abundance in the Gray Rock and Edith May veins, particularly at comparatively shallow depths from the 700 to the 1,200 ft. levels. It occurs in considerable amounts in the Leonard vein from the 1,200 to the 2,000 ft. levels. The covellite of the Leonard vein frequently shows a partial replacement of chalcopyrite. An interesting occurrence of covellite was noted in the Skyrme vein of the High Ore mine, where on the 2,400-ft. level a large ore boulder broken open was found to be composed principally of covellite intimately associated with enargite, some bornite and large amounts of later chalcocite. This entire mass was in large part inclosed by a 0.5-in. covering of pyrite. At several points on the surface of the boulder the pyrite, enargite, chalcocite, and covellite alike were being altered and replaced by chalcopyrite as the latest mineral.

In the Butte veins covellite is believed to be largely of primary origin, if not entirely so. Its occurrence bears no relation to the surface or to the zone of oxidation, and the intimate association with enargite, bornite, and pyrite lends strong support to the primary view.

### *Tetrahedrite.*

Tetrahedrite is unimportant as an ore of copper, although it has been found in small quantities in nearly every mine in Butte. It

varies in physical appearance from a dull gray to a bronzy metallic luster. The latter form occurs in considerable amounts in the Leonard mine, where it is remarkable for its high gold content. Lots of 50 tons have been known to run as high as $20 a ton. It is especially abundant in the stopes of the Gem vein above the 900-ft. level. The common variety has a characteristic cherry-red streak, which immediately distinguishes it from chalcocite.

*Tennantite.*

Tennantite is of frequent occurrence in the veins of the northeastern part of the district in the Badger, Speculator, and Gem mines, but is not of great importance as a source of copper. In massive form it is of a deep gray-black color, having a faint reddish tint. It is of more common occurrence as minute gray glistening crystals of a steel gray color, filling or coating the walls of fractures in older vein filling. These crystals are believed to have formed during the alteration of enargite to chalcopyrite, marking possibly the regeneration of the mineral from the copper and arsenic lost in the transformation of enargite into chalcopyrite.

*Sphalerite.*

Sphalerite is a common mineral of the Butte veins. It is of wide distribution in all of the veins of the district except in that part of the copper-producing area extending easterly from the Never Sweat to the Leonard, Berkeley, and Silver Bow mines. (See Fig. 7.) Sphalerite is especially abundant in the Colorado, Gagnon, Poulin, Lexington, West Gray Rock, and Corra mines, and to the north in the veins of the old silver producing area. Large bodies of sphalerite have been developed in the Elm Orlu and Black Rock mines, where it occurs chiefly in veins belonging to the Anaconda system. It is here found in unusual purity, the principal gangue minerals being quartz, rhodonite, and rhodochrosite. In marked contrast to the veins of the copper area, pyrite occurs but sparingly in the zinc ores of the Black Rock mine, running generally less than 2 per cent.

Sphalerite is found in abundance in veins and faults of the Anaconda, Blue, and Steward systems. It occurs at all levels from the oxidized zone to the greatest depths yet reached by mine workings. It is a primary vein mineral deposited contemporaneously with pyrite, quartz, galena, enargite, chalcocite, bornite, and other copper minerals, in veins of all ages. As is the habit with other sulphide minerals in the Butte veins, it is more abundant in higher levels in certain parts of the district, and relatively more abundant in deep levels of other localities. The relative proportion of sphalerite to copper content is

widely variable in different veins or at different sections within the same vein. In some instances veins have been found to contain an ore rich in sphalerite and low in copper in the higher levels with increasing proportions of copper as depth is gained. On the other hand, many veins with much zinc in the upper levels show no marked improvement in copper contents down to great depths.

Sphalerite is frequently found in intimate association with pyrite, bornite, chalcocite, galena, and quartz, and but rarely with enargite. Like enargite, sphalerite is often corroded or eaten away, the dissolving action being followed in many cases by the deposition of chalcopyrite, bornite, or chalcocite. The age relations of the sphalerite havè not been worked out. It probably began to deposit at an

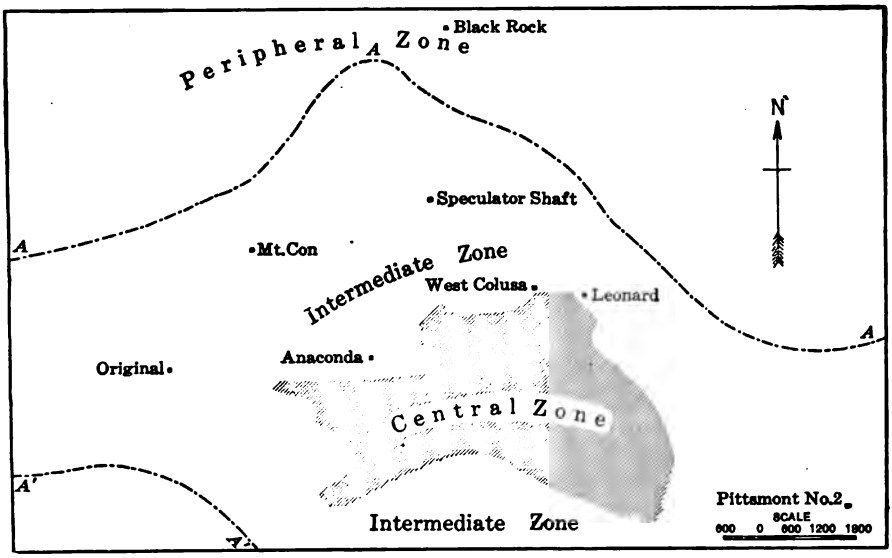


FIG. 7.—PLAN MAP AT ELEVATION 4,600 FT. ILLUSTRATING GENERAL DISTRIBUTION OF ORE TYPES WITH REFERENCE TO THE CENTRAL COPPER ZONE. (See pp. 1578 to 1581.)

early period of vein formation, and undoubtedly appeared in large quantities in certain parts of the Anaconda fissures long before the advent of the Blue system of fissures.

### *Galena.*

Galena is found sparingly in the intermediate zone surrounding the central copper area. It is particularly abundant in the region immediately north of the Mountain Con mine in the Old Glory, Lexington, and Gray Rock veins. It has been noted in the Anaconda, Blue, and Steward fissures in the intermediate and peripheral zones. Ga-

lena is a common, but not necessarily plentiful, constituent of the manganese-silver veins and generally of the veins of the outer zones.

*Silver.*

Silver is universally present in the Butte veins. In the copper area silver is more plentiful in the veins carrying zinc. The enargite veins of the Rarus, Berkeley, and Silver Bow mines have the lowest silver content of any in the district. Silver increases toward the border area, where the proportionate amounts of bornite, chalcocite and sphalerite become greater. The mineralogical nature of the silver occurrence has not been definitely determined. It is frequently seen in native form associated with chalcocite and bornite. Argentite has been noted in the veins of the silver area.

*Gangue Minerals.*

*Quartz.*—Quartz is the most abundant vein mineral of the district. It forms the chief constituent of the gangue of the copper and zinc veins, occurring in both massive and crystal form. Quartz is a product of vein-forming activity of all ages from the earliest known veins following the appearance of the quartz-porphyry dikes up to the close of the primary ore-forming period marked by the Steward-vein ore. Several generations of quartz are readily observed in the vein filling of Anaconda veins, all of which are of primary origin. Secondary quartz as a product of meteoric water circulation has not been observed, excepting possibly within or near the lower limit of the zone of oxidation. Quartz probably forms 70 per cent. by volume of the vein filling of the Butte veins.

*Pyrite.*

Pyrite is the most widely distributed of the sulphide minerals. It occurs both massive and crystalline in veins and fissures of the Anaconda, Blue, and Steward systems. Pyrite occurs sparingly as a primary constituent of the Butte granite, and abundantly in a finely disseminated condition in altered granite resulting from the attack of thermal waters upon the dark silicates of the original rock. In the veins of the district pyrite is more abundant in the early copper veins belonging to the Anaconda system than in later fault veins, and much more abundant in the veins of the central copper area than in the manganese-silver veins of the border areas. In vertical distribution it is doubtful if there has been a measurable change in the total pyrite present between the higher and the deep levels of the mines. In the regions of sooty chalcocite pyrite has been to some extent replaced by secondary chalcocite. Below the zone of sooty chalcocite no marked change has been noted.

### *Manganese Minerals.*

Rhodonite and rhodochrosite are found in great quantities in the veins of the peripheral zone surrounding the copper-producing area of the district, but rarely in the important copper-producing veins. These minerals are also abundantly associated with the sphalerite of the Black Rock veins.

### *Other Minerals.*

Hübnerite, fluorite, barite, and calcite are of frequent occurrence in the copper area. *Hübnerite* has been noted in the Steward, Leonard, Mountain View, and many other mines, where it appears as one of the earliest-formed minerals. It is found in considerable quantities in certain veins, 2 miles east of the Pittsmont mine. In the Mountain View mine blade-like enargite after hübnerite has been noted from the High Ore vein, a northwest-southeast fault of the Blue system.

*Barite* in characteristic tabular crystals is occasionally seen projecting into cavities in the veins. It is pale brown in color. Excellent crystals have been found in the Parrot mine.

*Calcite* is rarely seen in the copper veins, but it is commonly formed in veinlets along joint planes of the granite in areas of fractured or crushed character, where alteration processes have but slightly affected the rock. It is unknown in areas or zones of altered granite. Where found in copper veins it has always proved to be the most recent mineral, often filling cracks through chalcocite, chalcopyrite, and other copper minerals.

*Fluorite* has been found in the Parrot vein, in the Blue and Steward veins, and in considerable quantities in the Black Rock vein on the 1,200-ft. level of the Black Rock mine. In the Parrot mine on the 1,200-ft. level fluorite and enargite were intimately associated and apparently contemporaneous. Later chalcopyrite was observed replacing the enargite.

## The Ore Deposits.

The ore deposits of Butte are essentially of the fissure-vein type. They have resulted from the mineralization of fissures accompanied by replacement of the country rock. The fissure systems, as previously described, belong to at least six, or possibly seven, distinct periods of fracturing, and many examples of extreme complexity are presented. The oldest or first-formed fractures, composing the Anaconda system, have been continuously mineralized and are remarkably free from sudden changes in vein filling, a feature which presents a marked contrast to the later mineralized fault fissures of the Blue and

Steward systems. In the latter the ore occurs in great lenses, shoots, with intervening stretches of barren vein characterized by crushed country rock and fault gouge. The fissure-vein structure is the rule throughout the district. The ore-bodies display rather well-defined boundaries, when broadly considered. Important exceptions, however, are found in the Leonard, West Colusa, Rarus, and Tramway mines, where the largest ore bodies are more in the nature of mineralized highly fissured granite, having boundaries which are often commercial rather than geological The mineralization of the early complex fracture systems, followed by later faulting, has resulted in a most complicated arrangement of ore bodies. Reference to Plates I. and II. will convey to the reader a more comprehensive understanding of these structural relations than can be conveyed by detailed description.

The valuable metal content of the ores is chiefly copper, with subordinate but important amounts of silver, gold, and zinc. As mined, 60 to 80 per cent. by weight of the ore is altered granite, which usually, though not always, carries sufficient quantities of valuable minerals in seams, impregnations, or disseminations to constitute ore. Irregularities in the vein boundaries, horses, pinches, and included granite, necessitate stoping widths often in excess of the actual thickness of ore streaks. The copper ores invariably carry commercially important quantities of silver. The typical manganese-silver ores contain only traces of copper; and the newly developed zinc ores of Elm Orlu and Black Rock mines carry considerable silver, but rarely appreciable amounts of copper.

### *Distribution of Ore Types.*

An interesting geological condition is found in the unmistakable concentric zonal arrangement of certain ore types, based on mineral composition, around a central copper zone. It was observed in the early days of mining that ores from different mines exhibited considerable variation in mineralogical composition, but it has been only in the more recent years that underground developments have progressed to such an extent that the apparent orderly arrangement of the various types can be approximately outlined. From the information now available, the writer has formulated certain generalizations concerning these features, and on the accompanying map, Fig. 7, an attempt has been made to indicate as accurately as possible the zones or belts which are typified by characteristic mineral associations.

In brief, these zones may be grouped as follows:

1. A main or central copper zone occupying largely the great area

of altered granite in the vicinity of the Mountain View mine, in which the ores are characteristically free from sphalerite and manganese minerals. This zone is represented by the shaded area in Fig. 7.

2. An indeterminate zone of irregular width nearly surrounding the central copper zone, in which the ores are predominantly copper, but are seldom free from the mineral sphalerite, and near the outward boundaries, *A–A–A* and *A′–A′*, the manganese minerals rhodonite and rhodochrosite are of frequent occurrence.

3. An outer or peripheral zone of undetermined width bordering the intermediate zone, in which copper has not been found in commercial quantities. The vein filling is chiefly quartz, rhodonite, sphalerite, pyrite, and rhodochrosite. In this zone are included the manganese-silver veins of the Alice, Moulton, and Magna Charta mines on the north and the Emma, Ophir, Travonia, etc., on the south.

It should be kept in mind that the dividing lines between these three zones shown on the map are from necessity of an arbitrary nature. The passing of one zone into another is gradual, and cannot, as a matter of fact, be correctly represented by a mere line. Underground developments will never be of sufficient extent to permit of this extreme refinement.

*The Central Copper Zone.*—Within the central zone represented by the shaded area in Fig. 7 are found the typical copper ores in which the copper minerals are predominently chalcocite and enargite in a gangue of pyrite and quartz. Bornite is also present, but in proportionately small amounts. Covellite is uncommon, having been found only in the Leonard and Mountain View mines associated with chalcocite, enargite, and later chalcopyrite. Chalcopyrite and sphalerite are extremely rare, having been noted in but few instances. The manganese minerals rhodonite and rhodochrosite are unknown, as is galena. Silver is a universal constituent of the veins of this area, but in less quantity than in the ores of the intermediate zone. The ratio between the silver and the copper is approximately $\frac{1}{8}$ oz. of silver to 1 per cent. of copper.

It is of special interest to note that the central copper zone coincides in part with the great zone of rock alteration, and again a feature of interest is found in the fact that within the area there is no essential difference in mineralogical composition of the vein filling in veins of different ages, although there may be great variations in the relative amounts present. In one locality a vein of the Anaconda system may contain proportionately more enargite than a nearby vein of the Blue system, while in other localities the reverse is true. As a general con-

dition, however, it is believed that the later veins of the Blue system have more chalcocite and bornite, and less enargite proportionately than the veins of the Anaconda system. In the ores of the Steward veins the chalcocite ratio is especially high.

*The Intermediate Zone.*—In the intermediate zone surrounding the central copper area there is a noticeable change in the mineralogical composition of the veins. This change consists chiefly in the addition of the mineral sphalerite to the general type of vein filling of the copper zone above described. Outwardly from the central copper area the sphalerite does not appear suddenly in great quantity, but it comes in rather gradually. This feature is variable in different localities; for example, in the Moonlight mine there is but little sphalerite in the veins of the east end of the mine, but westerly even in the same veins, within a distance along the strike of 1,000 ft., the ores become very rich in zinc. In the Anaconda vein westerly from the Never Sweat mine sphalerite increases in amount slowly and gradually, and unusual quantities of zinc are found only at distances of 3,000 ft or more from the general outside limit of the central copper zone.

In addition to the changes noted in zinc content, other mineral variations take place that are of considerable interest. The manganese minerals rhodonite and rhodochrosite begin to appear in small quantities toward the borders of the intermediate zone, and increase perceptibly toward its outer limits. Among the copper minerals, as compared with the central copper zone, there is less enargite in proportion to the total copper present, but proportionately greater quantities of bornite, chalcopyrite, tetrahedrite and tennantite. Chalcocite apparently remains about the same relative to the total copper present as in the central copper zone. The silver content increases materially, the ratio to copper content being 1 oz. of silver to 1 per cent. of copper as a general average. It should be remembered that the facts above outlined are generalized conditions and that locally extreme variations occur. The net result, however, is a decrease in copper content with an increase in silver, zinc, lead, and manganese. Quartz as a gangue mineral does not vary perceptibly, but pyrite is less abundant toward the outside limits of the intermediate zone than in the central copper zone and certainly much less common in the peripheral zone than in the intermediate and central zones. In the intermediate zone the veins of different systems carry the same variety of minerals, but in widely different proportions. The veins are more zincky in some localities than in others. Those of the Anaconda system are exceptionally high in pyrite and quartz in certain localities, and of

ther places sphalerite is unusully abundant. There are certain areas n the Mountain Con and Diamond mines where the ores closely resemle those of the central copper zone, and, curiously enough, these reas are closely associated with the earliest vein development which ıad attended intense rock alteration.

*The Peripheral Zone.*—As with the central and intermediate zones, he line of division between the intermediate and the outer zones is argely an arbitrary one. It marks the general outside limit of presıt known commercial copper deposits, and therefore it is in nowise ntended to be construed as an attempt to mark the possible limit of copper ores. Up to the present time the ores of this zone have been valuable principally for silver, gold, and zinc. The vein filling is characterized by the abundance of the manganese minerals rhodochrosite and rhodonite. Sphalerite is present in great abundance, forming in some instances ore bodies of value. Copper is sparingly present, chiefly as chalcopyrite, tetrahedrite, tennantite, and rarely chalcocite and bornite. Pyrite is common, but in relatively much less quantities than in the two zones above described. Quartz is the most abundant gangue mineral. Galena is also present in considerable quantities intimately associated with sphalerite and of the same age. The width of this zone is indefinite and irregular and there is a noticeable change in the mineralogical character of the vein filling at greater distances from the central copper zone. Extending outward from the peripheral zone the fissures appear to become less mineralized; the manganese, pyrite, zinc, and other sulphides largely disappear, the vein filling consisting principally of quartz with scattered pyrite; and, curiously enough, arsenopyrite is noted at extreme distances, 2 miles or more from the central copper zone.

The zonal arrangement of ore types above described is repeated on a smaller scale in certain ore shoots of the Blue veins in the northeastern part of the district. In the shoots of the Jessie and Edith May veins, for example, the mineral composition differs in the central part of the shoot from that at the ends and along its walls. High-grade chalcocite-enargite-bornite ores form the central part of the ore shoot with but little sphalerite and chalcopyrite and shade almost imperceptibly into zincky ore with chalcopyrite toward the ends of the shoots.

## Vein Systems.

Of the seven distinct periods of fissuring in the district only three are known to be ore producing: namely, the Anaconda system, the Blue system, and the Steward system. The Anaconda system is the

dition, however, it is believed that the later veins of the Blue system have more chalcocite and bornite, and less enargite proportionately than the veins of the Anaconda system. In the ores of the Steward veins the chalcocite ratio is especially high.

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continuously stoped they are frequently composed of two or more closely spaced ore streaks, called the "foot-wall streak" and the "hanging-wall streak." It is not unusual to find a stope face showing four or five roughly parallel ore streaks varying in width from an inch up to several feet. The included granite and wall rock are highly altered and commonly netted with smaller seams of mineral similar in a general way to the large ore streaks. The variable mineralogical nature of the many ore seams, veinlets, etc., is a notable feature not only of the veins of the Anaconda system, but of all of the veins in the district. For example, a drift face may show a well-defined foot-wall streak 2 ft. wide composed almost entirely of quartz and pyrite assaying less than 2 per cent. of copper. The hanging-wall vein, separated from the foot-wall streak by perhaps 2 ft. of highly altered granite, may be very rich, consisting largely of chalcocite, enargite, bornite, and pyrite, assaying 25 per cent. of copper. However, as stopes are made above the drift, the rich streak may show only enargite and pyrite or only chalcocite, while the lean foot-wall vein in the stopes may become richer by the appearance of enargite, chalcocite, or bornite. These extreme local variations in veinlets and ore streaks are extraordinarily common to veins of all systems, although the general average valuable metal content of the ore as mined in a particular locality may not show much general longitudinal or vertical variation over hundreds of feet. It is a notable fact that while the variety of minerals present in a given vein seldom suffers marked changes within short distances, the relative proportions of minerals reach extremes.

The component ore or vein streaks may or may not be included within fairly well-defined boundaries. Usually one of the ore bands is wider, more prominent, and more persistent than the rest, the smaller streaks maintaining a rough parallelism separated by vein granite from the main fissure. The most important variation from the common type of vein above described is found in the Shannon-Colusa vein. The stoping width of the Anaconda veins varies from 5 up to 100 ft., with a general average of 20 ft. The Anaconda and Syndicate veins average 30 ft. or more. The stopes of the Leonard mine within the big stock-work ore bodies often run as high as 400 ft. from north to south. Within this extreme width, however, there are many bands or ribs of waste, and as a rule the actual stope is not solid ore, from 10 to 30 per cent. of the ground broken being sorted out and rejected. Among the less important veins 10 ft. is a common stoping width. In most cases the actual vein filling represents only

a ractional width of the stoped material, the remainder being mineral-bearing altered granite.

*Minerals of the Anaconda Vein.*—The veins of the Anaconda system carry more quartz and pyrite than the later veins, though the term "quartz-pyrite," formerly much used to designate the veins of this system, is, in the strict sense, a misnomer. Quartz and pyrite are present in great quantities in the Blue veins and form the chief gangue minerals, but in less quantities in proportion to the copper minerals present than in the earlier veins. The greater continuity and uniformity of the hard vein filling, its freedom from fault clay and crushed granite, together with the proportionately greater amounts of quartz and pyrite, gave rise to the early usage of the name "quartz-pyrite" veins as distinguished from the typical fault fissure not known to contain ore bodies. That "quartz-pyrite vein system" is not, strictly speaking, a proper designation, but often misleading, may also be readily understood by a casual inspection of representative ores from the different veins of the district.

The general mineralogical character may be best illustrated by a more detailed description of some of the more prominent veins belonging to the Anaconda system. Of those described below the Anaconda is typical of the important well-defined continuous compound fissures. The Shannon-Colusa illustrates fissuring of remarkable complexity and ore bodies of an unusual type. The Syndicate vein illustrates certain structural features of interest, while the Gray Rock vein is of interest as a copper vein well toward the border of the copper-producing area.

*The Anaconda Vein.*—Beginning in the Gagnon mine on the extreme west, the principal copper minerals are chalcocite, enargite, bornite, and chalcopyrite, named in order of their abundance. The gangue minerals are quartz, sphalerite, and pyrite, with occasional occurrences of rhodochrosite and hübnerite. In vertical arrangement there is possibly more chalcocite and less enargite and bornite in the upper levels than in the lower levels of the Gagnon mine. The sphalerite is only slightly variable as between the higher and lower levels, probably less in the latter. The most notable change in depth is the remarkable development of enargite in the bottom levels, accompanied by unusual quantities of later chalcopyrite, largely a replacement of enargite. In the Gagnon mine the silver tenor of the ore has decreased in depth. The oxidized zone here is from 100 to 200 ft. deep, accompanied by a considerable development of sooty or secondary chalcocite whose vertical extent is in the neighborhood of 400 ft.

Easterly from the Gagnon mine, in the Original and Steward, the oxidized zone with the accompanying zone of sooty chalcocite shows no marked change from that just described. Of the copper minerals, enargite and chalcopyrite decrease, while chalcocite and bornite increase in amount. Quartz and pyrite increase, especially the latter, but sphalerite shows a marked decrease. The silver content is slightly lower than further west. Rhodochrosite and fluorite have been noted in the Steward and Parrot mines.

Taking the next step eastward into the Never Sweat, Anaconda, and St. Lawrence mines, the enargite and bornite have continued to increase slightly, chalcocite increases, silver content becomes lower, sphalerite and chalcopyrite disappear completely, while on the whole, the vein becomes much more massive. The gangue, instead of being granitic, is hard, composed chiefly of quartz and pyrite. Hübnerite is noted occasionally; rhodochrosite and fluorite are unknown. In the Never Sweat-St. Lawrence section the oxidized zone is of unusual depth, averaging 300 ft. Below, the secondary chalcocite zone shows extensive development, having a vertical extent of from 600 to 800 ft.

Easterly from the St. Lawrence the Anaconda is faulted north by the High Ore vein into the Mountain View mine, where the mineralogical character does not differ materially from that shown in the Never Sweat-St. Lawrence region. The copper minerals are enargite and chalcocite, with unusually small amounts of bornite in a quartz-pyrite gangue. Southeasterly, where the main Anaconda vein apparently forks and in a large measure loses its identity, there is no noticeable change in mineral content, except as to relative quantities present, until the region of the Pittsmont or East Butte property is reached, where the only change is in the reappearance of sphalerite in considerable quantities. Chalcopyrite does not again appear except in rare cases, chalcocite and enargite forming the chief source of the copper. The silver content decreases slightly southeasterly from the Mountain View. The rarer minerals noted are fluorite in the Parrot mine, barite occasionally at different points in the vein, and hübnerite in the Mountain View mine.

On the whole, the veins of the Mountain View, Rarus, and Pennsylvania mines have more enargite but very much less bornite in proportion to the total copper present, than those of the Steward, Anaconda, Moonlight, and St. Lawrence mines.

*The Shannon-Colusa Vein.*—The Shannon-Colusa vein is a strongly mineralized fissure forming the north boundary of the complex Anaconda fracture zone previously described. In the Mountain View mine it has been developed as a compound fissure striking

slightly north of east, composed usually of a single vein in the upper levels, but of two or more branches at greater depths. The vein dips from 80° to 85° north. There are, occasionally, nearly north and south veins branching toward the south, but the north wall is generally clean cut and well defined. This comparatively simple vein structure is maintained eastward into the West Colusa mine, where it suddenly becomes exceedingly complex. Great mineralized fissure zones forming remarkable ore bodies break suddenly to the south nearly at right angles to the main foot-wall fissure. The granite between and forming the country rock of the complex fissure aggregates is richly mineralized with chalcocite and enargite as disseminations or veinlets following the joint planes of the granite.

In the Leonard mine the development of the north-south vein structure does not appear higher than the 600-ft. level. Figs. 1 and 2 illustrate the relation between the veins of the Leonard 300-ft. level and the fracture complex of the 1,200-ft. level. On the 300 there are four well-defined veins having approximate east-west strikes. The Colusa, the most northernly one, is a large well-defined vein, averaging 40 ft. in width, having an 85° dip to the south. At a depth of 600 ft. it experiences a sudden change to a flatter dip, due in part to faulting by the Rarus fault, causing it to appear on the 700-ft. level, 320 ft. south of its position on the 600-ft. level. From the 700-ft. level downward it meets the vertically dipping Gambetta veins, and it is below this level that the development of the north-south fissuring begins. The north-south veins are often well defined and carry wonderfully rich bodies of chalcocite-enargite ore, with occasional admixture of covellite associated with later chalcopyrite.

The Gambetta veins are intersected between the 300-ft. and 500-ft. levels and are displaced northward by the Rarus fault. Above the fault they carry but little commercial ore except near the Gambetta shaft. Below they are notably rich in enargite, often showing from 2 to 5 ft. of nearly pure enargite, with numerous veinlets of coarse enargite penetrating the vein walls. Where intersected by a crosscut on the 800-ft. level the Gambetta vein showed more than 6 ft. of solid, coarsely crystalline enargite, with little or no chalcocite, quartz, or pyrite. The oxidized zone of the Gambetta veins is slight, being not more than 60 ft. and usually much less. There is but a slight development of sooty chalcocite below; in fact, not enough to make the vein filling of commercial grade. The zone of oxidation in the Colusa vein, which varies from 100 to 150 ft. in depth, is accompanied by a marked enrichment of the veins below by sooty chalcocite. This enrichment zone, however, barely reaches the 600-ft. level. In the big

flat stopes connecting the 700-ft. and 600-ft. levels and in the big ore zones of deeper levels, the valuable copper minerals are chalcocite and enargite, with occasional bunches of covellite associated with chalcopyrite.

*The Syndicate Vein.*—The Syndicate vein of the Mountain Con group of mines is a massive quartz-pyrite vein with a curving strike from N. 70° E. in the Buffalo mine to a more easterly and southeasterly strike eastward from the Mountain Con mine. (See Plate I.) In general habit it does not differ materially from the main Anaconda vein above described. There are the common splits, or branches, on strike and dip, including horses of granite. The granite of the walls is highly altered and contains the usual minor mineralized fissures roughly parallel to the main vein. The Syndicate is cut and displaced by the Nappa, Dernier, Midnight, and other fault veins belonging to the Blue system. The Bell, a strong northeast fault, crosses the Syndicate vein at an acute angle, at times following it as a strike fault for several hundred feet.

In mineralogical character there is a notable difference between the Syndicate and the main Anaconda vein. The former is more massive, with a larger proportion of pyrite as a gangue mineral. There is more bornite and less enargite in proportion to total copper present than in the Anaconda. Chalcocite is abundant. There is the usual oxidized zone of from 150 to 250 ft. thick, accompanied by important secondary enrichment below. There is less chalcopyrite than in the Gagnon mine, but more than in the neighborhood of the Mountain View mine. The Syndicate vein contains large amounts of sphalerite in the Poulin, slightly less in the Buffalo, Mountain Con, and eastward. In the immediate vicinity of the Mountain Con shaft for a short distance on strike sphalerite is almost entirely absent. There is a tendency on the west to develop minerals characteristic of the manganese-silver vein area to the north. Barite, galena, tetrahedrite, and tennantite are frequently seen in the ores of the Poulin mine. The Old Glory vein of the Buffalo carries unusual amounts of galena

*West Gray Rock Vein.*—The West Gray Rock vein has been worked only in the Gray Rock mine. It is a well-defined vein of the Anaconda system, exhibiting the usual characteristics as to general strike, continuity of mineralization, etc. It appears to split into several branches, dying out quickly both east and west. It is a typical hard quartz-pyrite vein having more quartz and less pyrite than the Syndicate. Sphalerite is always present. Galena, chalcopyrite, and barite are occasionally noted. Chalcocite and bornite are the chief copper

minerals. Engarite is less important that in most of the copper veins.

It is faulted by the High Ore, the South Bell of the Blue system, and the La Plata of the Steward system, and the Corra fault, a north-dipping fissure possibly of the same age as the Rarus.

*Physical Changes Effected in Anaconda Veins by Faults.*—Where in contact with the intersecting fissures of the Blue vein system the Anaconda veins are usually shattered and broken, often exhibiting step faulting with a disturbance of orientation of the vein as the fault is approached. There is frequent strike faulting locally along the intersected vein, due to the general disturbance of the hanging and foot wall country-rock near the fault. Blue vein fault fissures often split where passing through older veins and develop the phenomena of step faulting.

The physical changes caused by faults meeting the Anaconda veins at acute angles or strike faults is more marked than that caused by faults like the Blue and Rarus crossing at obtuse angles. With strike faults along other veins, of which the Gagnon, Moonlight, and Bell-Speculator are notable examples, the movement may take place along the foot or the hanging wall of the vein or both. If the movement is extensive the older vein filling becomes more or less broken. If ore-bearing solutions are introduced by the later faulting, or if ore-forming waters are traversing the older vein, the new fractures and cracks in the old vein filling may become filled with mineral again forming in solid veins. This process of breaking and re-cementing of older veins by new primary ores is a common feature of the Butte veins, and where the process is many times repeated the chronological sequence of mineral formation cannot be satisfactorily determined.

Often where a strike fault crosses older vein filling diagonally from hanging wall to foot wall, the displacement along the fault causes the two segments of the vein to become separated on strike by a barren stretch of the fault fissure, giving the general effect of separate ore shoots, a feature early described by R. G. Brown.[8]

*Mineralogical Changes Due to Appearance of Faults.*—The changes in mineral composition of the Anaconda veins effected by later fault influence cannot be fully determined. It is certain that the Anaconda veins were well formed and existed as solid quartz-pyrite veins containing unknown quantities of copper minerals at the time the Blue fault fissures appeared, as shown by included blocks of drag ore found at intersections with Blue veins. The extensive mineralization of the

[8] Brown, R. G., The Ore-Deposits of Butte City, *Trans.*, xxiv., 555 (1894).

Blue fissures, identical in character with that of the Anaconda veins, indicates that there was no substantial period of quiescence separating the two vein series. With an overlapping of these periods, marked possibly by a slight change only in the character of minerals deposited, the problem concerning the influence of the Blue veins on earlier-formed veins becomes a complex one and difficult of solution.

Certain facts have been observed which tend to throw some light on this question. It is found that the crossing of older veins by fault veins of the Blue and Steward is not always accompanied by undoubted enrichment of the former. In fact, the stope and assay records show that as a general rule there has been no marked local enrichment at these intersections. There are many exceptions, however, where, at certain elevations along the line of intersection, between veins of the Anaconda system and later fault veins, both the old vein and the later fault fissure contain unusually rich ores. Careful observations of a number of such examples of enrichment show that they mark not merely the intersection of the two fissures, but the intersection of the older vein and an ore shoot lying within the plane of the fault fissure. The enrichment, therefore, bears no relation to the surface or secondary influence, but it occurs where the ore shoot happens to cross the older vein segments. A good example is found in the Parrot mine where a Blue vein ore shoot meets the Parrot vein between the 1,200-ft. and the 1,500-ft. levels.

It has been suggested that the Blue veins have been prominent zinc ore channels and that the Black Rock and other sphalerite veins probably of Anaconda age, of the north district, owe their unusual zinc content to the Blue vein fissures. The Black Rock vein appears to be of Anaconda vein age, as it is cut and displaced by northwest fissures of the Blue system, and no doubt its mineral content has been influenced by these intersecting veins. But it might be said with just as good reason that since these same northwest faults of the Blue system contain copper ore farther south, they were responsible for the copper in the earliest copper veins. The fact is, the Blue veins in the zinc regions contain sphalerite, quartz, and manganese minerals just the same as the other veins of that particular section. In the central copper area, however, the Blue veins carry no sphalerite, neither do the Anaconda or Steward veins; they all contain the ore and gangue minerals common to that locality. Intermediate between the copper and zinc areas, the Blue veins, which extend directly and continuously from one area to the other, contain ores of intermediate mineralogical composition, getting more zincky with less copper toward the zinc area, and less zincky with more copper toward the

copper area. Finally toward the north the copper minerals fail with increased sphalerite, manganese, and quartz, and less pyrite, while toward the copper area, sphalerite and the manganese minerals drop out completely, while pyrite and the copper minerals, chalcocite, enargite, and bornite, increase.

The above general statements apply with equal force to the fault-veins of the Steward system or any faults or fissures which have served as channels for uprising primary ore solutions. The Steward veins, like the Blue, exhibit the same variations in mineralogical composition which are believed to be functions of geographical position. (See pp. 1578–1581, inclusive, Distribution of Ore Types.) The direct effect of the Steward faults on the mineral composition of the Anaconda veins is a more obscure problem than in the case of the Blue veins, because the Steward fissures are so scantily mineralized that, even where in direct contact with older veins for long distances on strike, there are logical reasons for doubting that any primary enrichment whatsoever has resulted directly from these faults. In certain instances, however, it is believed that older veins have been enormously enriched by faults of the Steward system. Important examples may be cited, such as the Moonlight vein in the Anaconda mine, the Gagnon vein in the Original and Gagnon mines, the Bell-Speculator, and others. The No. 16 vein in the Rarus and Tramway mines contains remarkably rich primary chalcocite ore where it possibly acted in part as a feeder, exercising a marked influence on the mineral content of the big ore bodies of the Leonard, Tramway, and Rarus mines.

Since fault-vein ore occurs in separated shoots along the strike of the fissures, it is not improbable in the case of strike faulting that the crackled and reopened older vein filling has supplied important new channels connecting the widely separated ore channels of the fault veins, in which case the enriching influence upon the old vein filling might be very great.

*Influence of Faults on Secondary Enrichment.*—The influence of faulting on the secondary enrichment processes which have been active in the Anaconda veins is not easily determined. The breaking and shattering, by faults, of the old massive vein filling has undoubtedly permitted a more rapid and extensive oxidation of the vein and a correspondingly greater enrichment below. As in the case of primary minerals, the strike faults have exerted more influence on the older veins than the cross faults; that is, where the coincidence of the old vein and the strike fault occurs in levels relatively near the surface. In the St. Lawrence and Mountain View mines secondary enrichment

has reached unusual depths, owing to intense fissuring and a highly altered and mineralizing condition of the granite, and to faulting of Anaconda veins.

Weed's statement[9] that the early quartz-pyrite veins belonging to the Anaconda system were too low grade and unworkable except where fractured and enriched through the action of descending sulphate waters, has not been corroborated by the writer's own observations. The assertion that workable ores in the veins of the Anaconda system are localized and occur only at or near intersections with later fissures is entirely disproved by mining operations.

The Rarus fault has not influenced the mineral content of the ores of older veins, either in regions near the surface or at greater depths. The segments or blocks of ore included within the walls of the Rarus fault are no richer than the original vein. There has been no primary enrichment traceable to Rarus fault influence and it is believed that this fault has exerted but slight influence even in the case of the sooty chalcocite enrichment.

*Vertical Distribution of Ore Minerals in Veins of the Anaconda System.*—The upper parts of the Anaconda veins are oxidized to depths varying from 50 to 400 ft. The mineralogical composition of the oxidized portions of the veins is fully described under the subject of superficial vein alteration on preceding pages of this paper. Without exception in the copper district there is a development of sooty chalcocite immediately below the zone of oxidation in the Anaconda veins, which is found to be of variable vertical extent, depending primarily upon many factors, such as depth of the zone of oxidation, physical character of the vein, copper content of the primary ore, etc. Since the Anaconda veins were, in many instances, rich with primary copper minerals such as enargite, bornite, and chalcocite, apparently before being subjected to secondary influences of importance, the addition of a large amount of copper in the form of secondary chalcocite resulted in marvelously rich ores extending many hundreds of feet below the zone of oxidation. In the large quartz-pyrite veins, such as the Anaconda, Colusa, and Syndicate, the wonderful bonanzas of the upper levels were examples of the secondary enrichment of already rich primary vein filling. Below the zone of secondary enrichment there are no great variations in the mineralogical composition of the veins on the dip at any particular vertical section taken through the vein, but no general rule holds good for all veins. For

[9] Weed, W. H., Geology and Ore Deposits of the Butte District, Montana, *Professional Paper No.* 74, *U. S. Geological Survey*, p. 95 (1912).

example, in certain veins, such as the Johnstown, Shannon, Enargite, Windlass, and many others of the Mountain View mine, the predominant copper mineral in the higher levels is enargite, while in the Gagnon and possibly numerous other veins enargite is much more abundant in the deep levels than near the surface. Analogous conditions hold in the case of primary chalcocite, bornite, and sphalerite. In the "stock-work" ore bodies of the Leonard mine primary chalcocite is vastly more abundant at great depths than above the 800-ft. level. The same is true of the Moonlight, Original, and other veins.

Concerning the presence of the gangue minerals, quartz and pyrite, much difficulty is encountered in an endeavor to arrive at definite conclusions as to increase or decrease with depths. As indicated by the uneven distribution of these minerals along the veins, it is not improbable that they are principally deposited in irregular zones or shoots within the plane of the vein, so that observation on a single vertical cross-section through a vein does not accurately determine the general vertical distribution of either mineral. As a general observation on this point, however, it is believed by the writer that proportionately to the total gangue present, there is more quartz and less pyrite in the higher levels in the Anaconda veins than at great depths. Not enough data have been collected, however, from which reliable deductions can be made.

In the manganese-silver veins belonging to the Anaconda system but little information can be had on the matter of vertical distribution of minerals, owing to the inaccessibility of the mine workings.

### *Blue Vein System.*

The veins belonging to this system have resulted from the mineralization of the Blue fissure system previously described. (See pp. 1538–1540.) The veins known to be of greater or less importance as ore producers are the No. 1, Clear Grit, Blue, Diamond South, High Ore, South Bell, Skyrme, Adirondack, Edith May, Jessie, Gem, and Crœsus. In the earlier days of copper mining in Butte the presence of these well-defined copper-bearing fault veins was not recognized. Even for a long time after they were known as faults, their importance as ore carriers was not known. Their tardy development was due to two facts, later determined: (*a*) the ore bodies rarely reach the surface so as to come within the observation of the prospector, and (*b*) the early copper vein developments were along quartz-pyrite veins of the Anaconda system, little attention being paid to cross fissures, known to the miners as breaks, pinches, etc., which, as a

matter of fact, were usually barren where in contact with the older veins. (See Plate I.) But little encouragement was therefore offered by these veins, either to the surface prospector or to the miner underground. Many valuable mining claims of the district covering certain veins of this system remained idle and were considered as having but little value until they were cross-cut at deep levels from adjoining properties.

*Structure of the Veins.*—The Blue veins are typical fault fissures of marked displacement, with variable dips, but fairly uniform strike. (See Plates I. and II.) There is frequent splitting, or branching, on strike, the branches commonly re-uniting along the course of the vein after inclosing horses of granite. (See High Ore vein, Plate I.) In certain localities the branches are diverging and remain separate, as far as known from present developments. In closer detail the fault zone, which may be entirely included within a width of from 5 to 25 ft., is often composed of two or more well-defined movement planes characterized by seams of fault clay from 0.25 to 2 in. thick. The clay seams are usually accompanied by crushed granite of variable thickness, often occupying the entire width of the fault zone. The so-called crushed granite does not as a rule represent a cracked condition of granite adjacent to the movement planes, but more precisely, it is a distinct zone of finely comminuted granite, sharply separated from the fairly solid country rock by a wall or a clay seam marking a plane of movement.

*Mineralogy of the Blue Veins.*—The principal copper minerals are chalcocite, enargite, and bornite, together with small amounts of chalcopyrite, covellite, tetrahedrite, and tennantite, named in order of their importance. The gangue minerals are quartz and pyrite, with widely variable amounts of sphalerite and rhodochrosite, the latter minerals being present in certain localities only. Variations in the mineralogical composition of the ores of the Blue veins are common; in fact, are the rule. In certain ore shoots chalcocite predominates, in others enargite is the chief source of copper. Both are usually present, and bornite always but in less important quantities. Covellite is of small importance in the Gray Rock and Skyrme veins, but generally absent from the other veins of the Blue system. Chalcopyrite is of widespread occurrence, but in insignificant amounts, excepting in the Jessie vein, where it formed the chief ore mineral in the upper levels of the Jessie and Mountain Chief mines. Tetrahedrite and tennantite have been of little commercial importance, although of frequent occurrence, particularly in the northeast section. The gangue minerals are chiefly quartz and pyrite, with large

amounts of sphalerite in certain localities and minor amounts of galena, rhodochrosite, fluorite, etc.

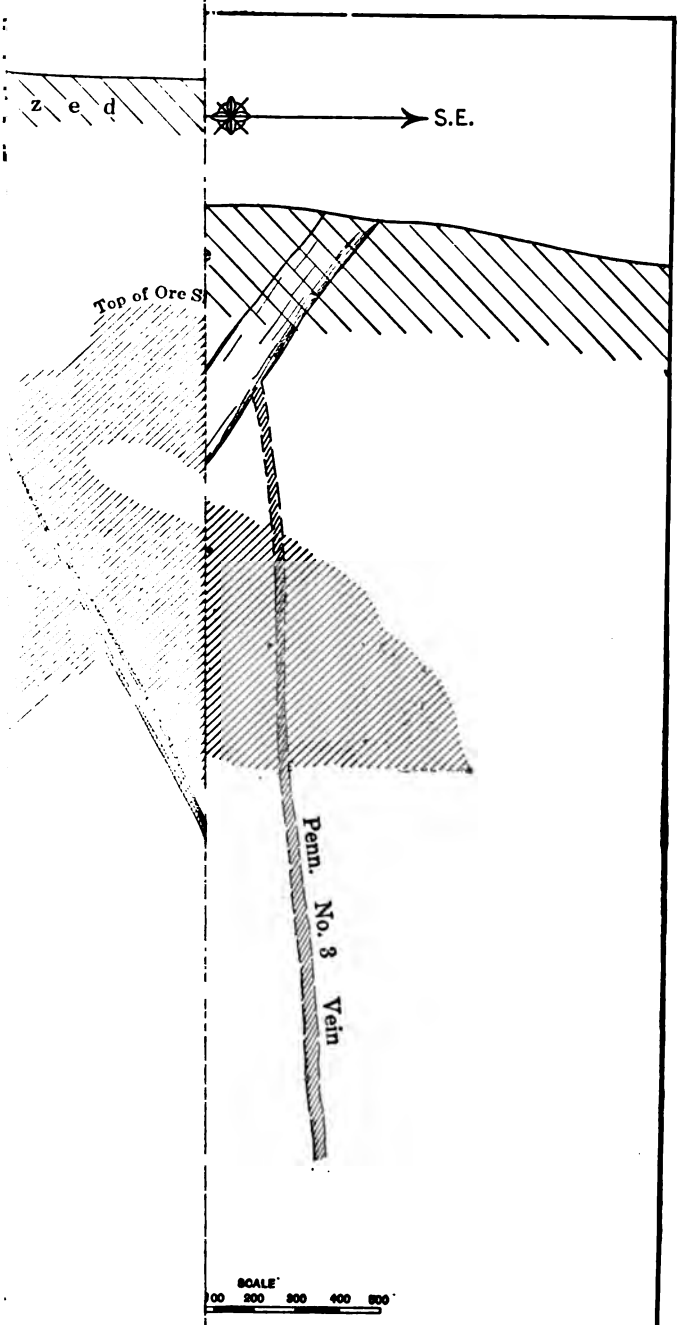
Like the Anaconda veins, the mineral variations are matters pertaining largely to locality. As far as the primary copper minerals are concerned there is no possible mineralogical distinction between the ores of the Blue veins and those of the Anaconda system. The Blue veins lack the zone of sooty chalcocite, so common to the older veins. The minerals composing the ore bodies are believed to be almost universally of primary origin. Where shoots of primary ore reach the surface they are oxidized and secondarily enriched below the zone of oxidation in a manner similar to the Anaconda veins.

*Ore Bodies.*—The ore occurs in irregular shoots, either along the main fissure or along the various branches comprising the fault zone. On the strike of the fissure the ore shoots are separated by barren stretches of typical fault material consisting of crushed granite intensely altered and one or more clay seams. The altered granite usually carries abundant disseminated pyrite and frequently small wavy quartz-pyrite veinlets of small lateral extent.

The general distribution of the ore shoots is shown in plan on the accompanying map. (See Plate I.) It will be seen that they are irregularly spaced along the fissures, and, at this particular elevation at least, they appear to be arranged wholly without regard to Anaconda or other vein intersections. Typical examples of ore shoot occurrences are shown in Plates V. and VI., longitudinal projections of the Jessie and High Ore veins, respectively, upon which have been outlined the ore shoots (shaded areas) as far as known from present developments. The pitch of the ore shoots is usually to the southeast, but not uniformly so. The stope length of the shoots varies from 100 to 2,000 ft. The width of the ore runs from nothing up to 30 ft. or more. Stoping widths of from 10 to 20 ft. are common.

*Structure of the Ore.*—As a rule the vein filling of the Blue vein ore shoots is less massive than that of the Anaconda veins. A certain amount, however, of mineralization and replacement of firm wall rock has taken place along the Blue vein fissures, as well as in the older veins, resulting in hard, massive, complex aggregates of ore and gangue minerals similar to the typical "quartz-pyrite" ore of the Anaconda veins. Where, as is commonly the case, the ore is a complete or partial replacement of crushed granite, by quartz, pyrite, and copper minerals, it may be readily distinguished from ore formed by replacement of hard granite by reason of retaining the brecciated structure. Blue-vein ore frequently consists of crushed and altered granite netted by stringers, bands, and tiny seams of the copper

SALES.—Plate VI.

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Note that the shoots

minerals, chalcocite, enargite, bornite, and chalcopyrite. In such cases quartz and pyrite may be present in limited amounts, intimately associated with the copper minerals forming the ore seams. Crushed highly altered granite carrying disseminated chalcocite and bornite, are common features of Blue-vein ore shoots.

The structural details of the ores are perhaps not as simple as might appear from the above general description. The order of deposition of the minerals has been much obscured by faulting, which took place along the fissures during the period of mineralization. The ore shoots often show rude banding due, in the main, to replacements along closely spaced fissures or planes of movement, or to shearing of the early-formed ore masses.

*Influence of Vein or Fault Intersections upon Mineralogical Character of Blue Veins.*—The intersection of a vein of the Anaconda system by a vein of the Blue system, or the faulting of a Blue vein by later fissures, has not resulted in marked changes in the mineralogical character of the Blue vein ore shoots. There is a possibility that at great depths the uprising solutions followed east-west fractures and at times found their way into the intersecting Blue fissures, through which they continued to ascend along the more open zones eventually forming the ore shoots. Developments are not of sufficient extent to determine this point conclusively.

The ore bodies of the Blue veins do not occur at intersections with older veins with any more frequency than in the wide stretches of barren country rock separating ths older veins. There is no evidence whatsoever that secondary ores have been formed in Blue veins through the action of descending meteoric waters in adjacent older vein segments, through which process the surface waters are supposed by Weed[10] to have taken up copper and, on their descent into the earth, spread out along the older vein and into intersecting Blue fissures, forming notable bodies of chalcocite. Plate VI., a longitudinal section of the High Ore vein typical of the Blue system, shows beyond question that the ore shoots bear no genetic relation whatsoever to the positions of the intersected segment of the Anaconda vein.

### *Steward Vein System.*

The mineralization of the Steward fissures has not been of such great economic importance as that of the older vein systems just described. The ore bodies of the No. 16 vein of the Rarus and Leonard mines are the largest yet found in this system of fissures. The La

[10] Weed, W. H., Geology and Ore Deposits of the Butte District, Montana, *Professional Paper No. 74, U. S. Geological Survey*, p. 103 (1912.

Plata vein north of the Syndicate vein in the Mountain Con mine yielded a small amount of silver ore from shallow workings. In depth two small shoots in the Buffalo mine were worked for copper; however, they have thus far proved to be of little commercial importance. In the Gagnon mine the Gagnon South vein, apparently belonging to the Steward system, has yielded a considerable tonnage of copper-silver ore. This vein may possibly be the western extension of the No. 6 fault of the Parrot mine.

*Occurrence of the Ore Bodies.*—The ore of the Steward fault veins occurs in shoots similar in form and general character to those of the Blue vein system. There are many instances where Steward fissures are strike faults along veins of the Anaconda system, forming what might be properly termed compound veins. In examples of this class it is next to impossible to determine with any degree of certainty the influence of the fault fissure on the mineralogy of the older veins. Usually the fault fissure follows along the vein for a time on one wall, then crosses diagonally to the opposite wall. If the vein filling of the old vein is wide there may be but little breaking or crushing. As a general rule, the old vein itself is stronger than the adjoining altered wall rock, so that the strike fault occupies the plane of least resistance, which is usually the contact between the vein filling and wall rock. Further movement crushes the weaker altered granite, forming a zone of fault material following the solid ore. Where a small ore vein is overtaken by a strong strike fault both walls of the older vein may form planes of movement. The older vein filling thus inclosed within fault zones has the appearance of the usual fault vein ore and it is generally impossible to correctly classify such mineral masses without additional corroborative evidence. Even where the evidence proves conclusively that the major part of the vein filling belongs to a geologically older vein system, the influence of the strike fault on the mineral composition may be determined with difficulty, if at all. The mineral mass may exhibit later-filled fractures in primary vein filling, or replacements of the first-formed minerals, etc., but the phenomena of successive periods of mineral deposition is so frequently met with in veins of all ages that such evidence cannot be given great weight. The existence of minerals indigenous to the fault fissure itself in the vicinity of the mineral mass in question, identical with that found filling fractures in the older vein filling, offers satisfactory evidence of the fault influence.

The most important ore shoots found in fissures of Steward age and known positively to be indigenous are those of the No. 16 vein in the Rarus and Leonard mines. Although existing as a single well-defined

ore-bearing fault fissure in the Rarus mine, in the Leonard mine the No. 16 vein fissure splits into two or more branches, both of which contain important ore bodies. The ore is especially high grade, often showing from 1 to 4 ft. of reasonably clean chalcocite ore, containing also enargite, quartz, and pyrite, with little or no sphalerite, chalcopyrite, or bornite.

*Depth of Ore Shoots.*—No ore was found in the No. 16 vein higher than the 1,000-ft. level of the Rarus mine, or more than 1,100 ft. from the surface. In the Leonard mine the first ore was encountered at more than 1,200 ft. below the surface. These ore shoots have persisted to the greatest depths yet reached by mine workings.

*Influence of Later Faults.*—There is no evidence that the quality or mineralogical character of the Steward ores has been influenced by later faulting. The high-grade chalcocite-enargite shoot of the No. 16 vein in the Rarus mine is cut sharply by the Rarus fault unaccompanied by enrichment. (See Fig. 5.) Where the ores occur at great depth, as in No. 16 vein below the oxidized zone, it is doubtful if any appreciable amount of secondary enrichment has taken place. The chalcocite and other copper minerals of these deep ore bodies are believed to be universally of primary origin.

*Minerals of Steward Vein Ores.*—The mineral composition of the Steward ore bodies corresponds to that of the Anaconda and Blue veins, but depends on the particular locality in question. In the Rarus and Tramway mines the copper mineral comes principally from chalcocite, which occurs in great purity. There are usually present in small quantities enargite and bornite, but no sphalerite. There are also less quartz and pyrite than in the older veins. In the Gagnon district the ore shoots contain a large amount of sphalerite similar to older vein ores. The same is true of the La Plata ore shoots, where the ore contains sphalerite and rhodochrosite, both characteristic of the border zones.

In structural appearance the Steward ore shoots are similar to the previously described Blue vein shoots. They are, however, less complex both mineralogically and structurally, and it does not appear that the faulting movement continued to any great extent during the period of ore deposition.

## Contrasting Features of Vein Systems.

### *General Summary.*

*Forms of Ore Bodies.*—The Anaconda fissures are solidly and continuously mineralized, forming ore bodies which may be continuously stoped over thousands of feet without showing any great variations

in valuable metal content. Anaconda veins seldom exhibit evidences of extensive fault movement so characteristic of the Blue and Steward veins.

The ore of the Blue and Steward fissures occurs in the form of "shoots" which vary greatly in size and extent. These ore shoots are irregular in outline and are separated on the strike of the fissure by hundreds or even thousands of feet of barren crushed granite and fault clay composing the fissure zone. The walls of the ore are seldom free from fault gouge or other evidences of extensive movement.

*Oxidation and Enrichment.*—The vein filling of the Anaconda fissures within the copper area extends with great strength upward to the oxidized zone. It is deeply oxidized and invariably shows a marked development of secondary chalcocite below the zone of oxidation. which has proved in most instances to be of immense commercial importance. (See Fig. 6A.)

In the Blue and Steward fissures. the ore shoots seldom extend upward to within 500 ft. or more of the zone of oxidation. Except in the zones of highly altered granite the zone of oxidation in fault veins is shallow, the vertical extent being not more than from 20 to 40 ft., and often as low as 10 ft. The disseminated pyrite so universally present in the crushed granite of the fault fissure may show slight secondary chalcocite enrichment immediately below the zone of oxidation, but it is of no commercial importance. (See Fig. 6.)

*Physical Character of the Ores.*—As regards physical character, the ore of the Anaconda veins is usually harder, more massive, and it seldom exhibits the breccia structure so often characteristic of replacement of fault breccia of the fault veins. The solid ore streaks of the Anaconda veins are wider, and the metasomatic action has been sharper and more complete than in fault veins where vein-forming process were often disturbed by fault movements.

*Mineralogical Differences.*—There are no notable differences in the mineralogy of the veins of the three systems in any given area. As noted on a previous page, the veins of a certain limited area contain minerals characteristic of that particular area, regardless of geological age. The Blue veins are later than the Anaconda veins and they have less pyrite but more quartz in proportion to the total vein filling present. There is also the same general order of sequence in mineral deposition in the various veins. The earliest minerals are quartz and pyrite, followed by enargite, bornite, and chalcocite in the central zone. In the intermediate zone the order is quartz and pyrite. enargite, sphalerite, bornite, and chalcocite, with chalcopyrite probably forming contemporaneously but under different conditions in

border zones. In the peripheral zone, the order among quartz, pyrite, galena, and sphalerite has not been worked out owing to inaccessibility of mine workings.

## Genesis of the Ore Deposits.

### *Source of the Ores.*

The evidence points directly to the conclusion that the ores have been derived primarily from rocks of igneous origin. The veins are found entirely within igneous rocks, with no sedimentary rocks in quantity within 15 miles of the district. Formerly sedimentaries may have covered the granite of the Boulder batholith, but there is no evidence indicating that such rocks played any part whatever in the formation of the Butte ores. The ore deposits of the district are so centered or concentrated within a relatively small area that causes which led to their formation must be sought within, or in close proximity to, the area in which the ores are found.

Compared with other mining districts situated within the borders of the Boulder granite batholith, the chief point of difference in the geologic structure is the presence in Butte of the quartz-porphyry dikes in close association with the veins. In the Corbin, Wickes, Rimini, Basin, Clancy, and Alhambra mining districts no quartz-porphyry has been found, although rhyolite intrusions are common, corresponding, no doubt, to the rhyolite intrusion forming Big Butte at Butte. The districts above named, however, have developed no important copper deposits, and it is believed, therefore, that the many rhyolite eruptions occurring at a comparatively recent date within the Boulder granite area, played a minor part in the deposition of the copper ores at Butte and elsewhere. The rhyolite at Butte is not directly associated with the ores in any way. The copper veins become poor going westerly, and the earlier vein systems are known to be older than the rhyolite.

There are three or more well-defined quartz-porphyry dikes traversing the Butte district in close association with the copper ores of the main central zone. Those dikes extend in an easterly and westerly direction, coinciding in a significant manner with the general habit of the veins of the Anaconda system. (See Plate I.) It will be observed that in going easterly from the Anaconda mine the Anaconda veins bend to the southeast toward the Silver Bow mine, while a similar bend may be noted in the quartz-porphyry dikes. Farther to the east a change in strike from southeast to northeast in the dikes is accompanied by a similar variation in the course of the Anaconda veins. These last changes in strike in both the dikes and the veins

are accompanied by corresponding changes in dips from south to north.

Another feature of interest in this connection is found in the fact that the areal extent of the quartz-porphyry appears to be increasing in depth. Where the upper levels of the Mountain View mine show but one small dike of an average width of 30 ft., the 2,200-ft. level discloses three dikes, having a combined width of 150 ft. The main St. Lawrence dike, found no farther west than the Nipper shaft on the surface, has been cut in the Gagnon, 1,900 ft. level, 2,500 ft. west of the Nipper. Again, in the Mountain View, West Colusa, and Leonard mines, areas of the most intense fissuring of Anaconda age are associated with quartz-porphyry dikes, and dikes wholly unsuspected at the surface are appearing in deep levels.

A certain significance might also be attached to the marked zonal arrangements of ore types about the central copper zone in which the more important quartz-porphyry dikes occur. This fact tends to support the belief that the Butte ores were derived from a demonstrable centralized source.

Notwithstanding these apparent close genetic relations between the copper veins of the central zone and the quartz-porphyry, it does not follow necessarily that the quartz-porphyry magma was the immediate and direct source of the ore minerals now found in the veins. The association of the quartz-porphyry dikes and the Anaconda veins seems to have greater significance when considered in connection with the origin of these early fissures than when referred to the source of the vein-forming waters. There is no evidence indicating a direct transfer of vein-forming waters from the quartz-porphyry dikes into the adjacent granite, nor does it appear that any notable alteration of the granite took place prior to the appearance of the Anaconda system of fractures. The dikes are altered only in areas where the granite is altered in connection with the veins. The Modoc dike extends northwesterly from the East Colusa mine through a long stretch of unaltered granite, in which the dike itself shows no alteration whatever. Other similar instances have been noted.

It is a reasonable assumption that the quartz-porphyry was derived primarily from the parent granite magma; and also that the ultimate source of the ore minerals was the granite magma. The principal part played by the quartz-porphyry has apparently been the opening of the way for vein-forming waters of deep-seated origin to reach the higher regions where the ore deposits are now found. While the ultimate source of the metals of the ores was probably the original

granite magma, the direct source may have been the same magma locally which furnished the quartz-porphyry, the latter rock following the earliest fracturing and at the same time stimulating an upward movement of the ore-bearing waters.

### *Ore Deposition.*

*Ore-Forming Agents.*—There is little doubt that the chief transporting agent at work in the formation of the primary Butte ores was water. Vein-forming waters are believed to have been derived from the same fluid magma which earlier produced the quartz-porphyry dikes, and possibly also, at a much later period, the rhyolites. Such waters, of deep-seated origin, either contained primarily the elements which now go to make up the miuerals forming the ores, or they gathered them from the wall rocks in the course of their upward journey. In regions open to observation the almost entire absence from the normal granite of many of the most abundant elements of the ore-forming minerals, such as copper, arsenic, sulphur, zinc, etc., points to the conclusion that these elements could not have been derived from the rocks adjacent to the avenues of travel, but that they were probably constituent parts of the solution when the upward journey began.

*Composition of Vein-Forming Waters.*—Certain general conclusions may be drawn as to the probable composition of these uprising waters when they reached the regions in which the ore deposits are now found, through a study of the character of the vein minerals known to be of primary origin, and the altered condition of the rocks directly associated with the veins. In the Butte district enormous quantities of copper, sulphur, arsenic, zinc, and manganese have been added, together with notable quantities of lead, antimony, silver, tungsten, barium, fluorine, etc.; most of which are very minor constituents of the normal granite rock and some of them have not been detected in the original granite by chemical analyses of fresh rock. In addition, the abundance of pyrite and quartz in the veins indicates the presence in the solutions of large quantities of iron and silica, both of which, however, are essential constituents of the original granite. Although purely a matter of speculation, it seems unnecessary to assume that the uprising solutions originally contained notable quantities of either iron or silica, both of which could have been readily derived from the granite wall rock at great depths through chemical processes.

The variable mineralogical nature of the ores of different veins presumes, apparently, many changes in the composition of the vein-forming waters from time to time. However, such a statement re-

quires some modification. It is conceivable that from uprising waters carrying constant proportions of ore-forming elements, varying quantities of certain minerals might deposit at different localities within the same vein, or at different periods in the same locality, due to changes in temperature, pressure, or other controlling factors. Analyses of the altered granite in association with the veins show that large amounts of sodium and calcium have been extracted from the granite. These elements were undoubtedly present in the vein-forming waters in varying quantities along with the metallic elements which were later deposited as minerals in the vein.

The probable high temperature and vigorous chemical activity of the earliest solutions passing upward through the fissures of the Anaconda system is indicated by the intensely altered condition of the wall rock in the region of such fissuring. The composition of these solutions can be judged in a general way only from a study of their effects. The widespread development of pyrite both in veins and as disseminated pyrite in granite, in the absence of sulphur as a constituent of the original granite, requires the addition of sulphur in large quantities. The extraction of the sodium and calcium in this process with addition of sulphur, forming pyrite, may be accounted for by the presence originally of an abundance of $H_2S$. Since this activity was probably more intense at great depths, it is reasonable to assume that the ascending waters in the upper regions now open to observation carried not only hydrogen sulphide but alkaline sulphides as well.

*Processes Involved.*—The ore-forming solutions were gathered at great depths and were transported upward and deposited largely through metasomatic replacement of the country rock along fissures so as to yield the ores now being mined. The factors which influenced mineral deposition at certain horizons are believed to be mainly those of temperature and pressure. However, another important factor entered into the vein-forming processes which is believed to account in no small way for the variable nature of the ore minerals in different localities and at different points in the same vein. The association of the great areas of altered and pyritized granite with the massive quartz-pyrite veins belonging to the oldest vein system indicates that the early solutions were extremely active chemically, and it is not improbable that the first processes were in certain measure solfataric in their action. These solutions or gases readily attacked the granite wall rock to which there was free access along the fissures, and they also found passage by means of cracks, joint planes, etc., outward for considerable distances from the main chan-

nels into the granite. While the alteration was in progress the oldest vein filling, largely quartz and pyrite, was being deposited along the fractures. In these early processes of granite alteration the attacking solution carried sulphur in some form. It is believed that at great depths it was, in part at least, in the form of hydrogen sulphide. Analyses of normal and altered pyritized granite show that, as a result of vein-forming action, the normal granite has lost sodium and calcium, but that in regions open to observation in the Butte mines, iron, although readily attacked, has remained fairly constant in quantity. The formation of pyrite in the process, accompanied by the extraction of sodium and calcium, would seem to indicate the presence of hydrogen sulphide. The action of the attacking vein-forming waters finally resulted in the formation of pyrite, largely *in situ*, from the iron of the original granite and the soluble alkaline sulphide, $Na_2S$, which was not again deposited in the fissures, but migrated far beyond the zone of ore deposition, probably reaching the surface through hot springs. The presence of soluble alkaline sulphides in the vein-forming waters is believed to have exerted a marked influence in the deposition of the Butte ores, as will be later seen.

The above line of reasoning suggests the possibility that a large part of the pyrite, and possibly the quartz of the Butte veins, were not primary constituents of the vein-forming waters as they began their ascent from great depths. The presence of veinlets of pyrite in the altered granite in connection with the Anaconda veins indicates that some of the pyrite formed through the action of deep-seated waters on the iron of the granite, migrated for short distances at least, to be redeposited as vein pyrite. It is doubtful, however, whether appreciable amounts of the pyrite of the veins originated in this manner from nearby wall rock. Since, however, it is evident that the vein-forming solutions did not attack the iron of the granite, it is not unreasonable to suppose that at greater depths conditions were such that the iron thus attacked did not immediately combine with sulphur to form pyrite *in situ*, but that the iron was taken into solution and remained so until more favorable conditions for deposition as pyrite were found. Such favorable conditions may have been encountered in the fissures at higher regions, now occupied by the ore deposits. The same method of reasoning may also be employed to account for the quartz of the Butte veins. The possible origin of the pyrite in the above manner has much significance when considered in connection with the variable mineralogical nature of the ores, as will be later set forth.

Further consideration of the possible chemical action between the uprising waters and the granite wall rock is of interest when viewed

in connection with the variations in mineralogical character of the ores deposited during different periods in the same locality, and also variations in ore types zonally arranged around the central copper zone previously described.

As already stated, it is believed that the earliest ascending waters vigorously attacked the wall rocks on their upward journey through the newly formed Anaconda fractures. Sodium and calcium were extracted from the normal granite, and since they were not again deposited, the attacking solution necessarily became more and more alkaline toward the surface. The chemical activity in the early stages was more intense because (1) the solutions were probably at a higher temperature than during later periods, and (2) when the fissures were first formed the solutions had direct access not only to the unaltered wall rocks of the fissures, but to much sheared and easily attackable, crushed, unaltered granite within and along the fissures. Under these conditions, if it be assumed that the original solutions contained certain proportions of hydrogen sulphide and alkaline sulphides, or alkaline carbonates, it is evident that the solutions would become more alkaline with a less preponderance of $H_2S$ as they moved upward toward the surface.

From the state of maximum alkalinity, which is believed to have occurred when the early high-temperature waters had only unaltered granite to work upon, there was probably a gradual return to conditions wherein a higher ratio of hydrogen sulphide was established. Observed facts show that the alteration of the granite took place outward from the fissures, cracks, and joint planes through which the solutions passed. It is reasonable to infer that after a certain length of time the solutions no longer reacted chemically with the already altered granite adjacent to the avenues of travel; in other words, a partial chemical equilibrium was established between solution and the fringe of altered granite contiguous to the fissure, as far as the process concerned the extraction of alkalies or the attack of the iron of the granite. Metasomatic replacement of altered rock adjacent to the fissures by vein minerals, principally quartz and pyrite, was probably taking place during this time as a part of the general process. In the later stages, therefore, the ascending waters were unable to take into solution such large proportions of sodium, calcium, iron, and silica per unit volume of solution passing, owing to the presence of the protecting layer of already altered granite bordering the fissures, upon which the solutions had but a slight effect. Assuming then a fairly constant $H_2S$ content of the original deep-seated solutions, there must have been at first an increase in the proportion of the alkalies and a

corresponding decrease in $H_2S$ as the solutions moved upward, until a certain maximum degree of alkalinity was reached, after which time there was a gradual decrease in alkalinity, due to the protective effect upon the normal granite exerted by the altered granite barrier from which the available sodium and calcium had been extracted. A further decrease in the alkalinity of the solutions would be effected at later periods because of the development of solid veins, for subsequent fracturing of the vein filling (a common phenomenon) would permit a ready passage for the vein-forming waters along and in contact with vein minerals. The $H_2S$ solution would thus be afforded a double protection against the influence of normal granite (1) by the vein filling, and (2) by the altered granite belt forming the vein wall rock.

If, as has been already suggested, the vein pyrite was derived from the granite through the action of uprising alkaline sulphide or $H_2S$ solutions on the iron of the normal granite, the decrease in alkalinity would be accompanied by a corresponding decrease in the pyrite content of the later vein-forming waters, for the same reasons as above outlined. The latest vein filling in a given vein should contain less pyrite than the earliest concentrations of vein minerals.

Regarding the above discussion of the probable effect of the chemical action between the uprising vein-forming waters and the granite, as affecting the deposition of certain ore types, the writer does not assume that such reactions have been the sole influence in the determination of the vertical and lateral distribution of the minerals and ore types in the Butte deposits. In fact, this discussion has been undertakeu largely for the purpose of bringing out more forcibly the possible genetic relations between certain ore types and the more important areas of altered granite. The elements Na, Ca, and Fe were chosen for discussion because comparative chemical analyses of normal and altered granite show clearly that these elements have entered prominently into the reactions between the vein-forming waters and the granite, and therefore, if such reactions were to any degree influential in the formation of ores, the above-named elements were, no doubt, the most directly concerned. To the variations in temperature and pressure, to changes in the composition of the original solutions from time to time, and to the mingling of unlike solutions must also be assigned important rôles in the distribution of the primary ores throughout the veins of the district.

Concerning the possible influence of these various factors as affecting the distribution of ore types, many facts have been observed that are of particular interest. For example, primary chalcocite and enargite occur only in association with the intensely altered granite of

the district. In the central copper zone, or in veins where alteration has reached an advanced stage, these two minerals are commonly found in intimate association with other vein minerals forming the ores, and, in addition, small veins or stringers, largely chalcocite and enargite, may be found extending outward into and replacing the wall rock, the granite being in actual contact with the copper minerals. In areas of unaltered granite, chalcocite and enargite are confined to highly altered zones within or along the veins, and never as stringers or disseminations inclosed by unaltered granite. It is improbable that solutions bearing these minerals did not find passage outward into the normal granite during the active vein-forming period, when chalcocite, bornite, and enargite were being deposited in abundance in the veins. The frequent occurrence of stringers of pyrite, quartz and sphalerite extending into wall rock of normal granite in the intermediate zone, with addition of manganese minerals in the peripheral zone, indicates that the mineral-bearing waters did to some extent find their way into the wall rock, but only where alteration was marked were chalcocite and enargite deposited contemporaneously in the veins proper and in the altered granite wall rock.

Applying a similar method of reasoning to the remaining important minerals of the veins, it is observed that quartz and pyrite are found abundantly in association with all stages of rock alteration and with all periods of primary ore deposition, although there is a noticeable decrease in the proportionate amount of pyrite in the intermediate and peripheral zones as compared to the central zone. It is true also that there is proportionately less pyrite in the Blue vein ores than in the Anaconda veins, and markedly less in the Steward ore shoots than in the Blue veins, an indication that the later vein-forming waters either contained less pyrite or that conditions were less favorable for deposition. These later ores are quite certainly less rich in pyrite for reasons previously stated. Quartz and pyrite are thus found to have been deposited under widely varying conditions of the solutions, such as composition, temperature, and pressure. Sphalerite is rarely found in zones of most intense alteration. It is sparingly developed in the slightly less altered areas of the Mountain Con and Diamond mines, and is very abundant in veins occurring in regions of relatively unaltered rock. It is believed to have been deposited under conditions of alkalinity similar to those under which enargite was formed, but generally at lower temperatures. Being more soluble, it would tend to migrate in solution before deposition through greater distances than would enargite.

Galena is a relatively uncommon mineral, associated usually with sphalerite. It is not found in the central copper zone. The manganese mineral, rhodochrosite, is an abundant constituent of the veins of the peripheral zone, and is not uncommon toward the outward limiting boundaries of the intermediate zone. It is entirely unknown in the central zone. Chalcopyrite is not an uncommon mineral associated with sphalerite, quartz, and galena in the manganese-silver veins.

In an endeavor to explain these general relationships between the various vein minerals and altered granite areas, certain assumptions must be made in order to form a working hypothesis. It is believed by the writer that the ores of Butte in their entirety have been derived from one general centralized source at great depths. There is no evidence in support of, and much evidence contradicting, the hypothesis that the so-called manganese-silver or zinc veins belong to a vein-forming epoch separate and distinct from that in which the copper veins were formed. Going outward from the central area (see Fig. 7) of typical zinc-free copper ores, there are no instances of early copper veins being cut by later veins of different mineralogical character, which would naturally be the case if, for example, the Blue fissures were of a distinctly later period than the copper veins of the east-west system. It has been suggested also that the manganese-silver ores represent an invasion of an older copper area, or *vice versa;* this is also untenable, for the reason that it is an unlikely assumption that a later mineralization period could almost completely surround a central core, depositing new mineral in fissures identically the same age as those of the central area, without some of the later ore types appearing within the inclosed zone. It has been shown that the mineral composition of the different vein systems is not a result of separate periods of mineralization, but rather of geographic position. The structural relations point strongly to the conclusion that the east-west fractures of the manganese-silver area are of the same age as the Anaconda fractures of the copper area, and that they were well mineralized by quartz, pyrite, rhodochrosite, and sphalerite at the time they were cut by fissures of the Blue series. The Blue veins, furthermore, with their typical silver mineralization of the peripheral zone, continue southward into the copper area, cutting the oldest copper veins, and there contain valuable copper ores and are characteristic copper veins. Fissures of the Steward system are also found in both the copper and the silver areas, cutting copper ores in the former case and zinc ores in the latter. Therefore, while some copper ore bodies are found in these fissures, and zinc mineral-

ization is also found in them, the bulk of the vein filling in both areas is prior to this system of fissures. These structural relations between mineral deposition and the faulting periods point to the conclusion that the primary ores of the whole district are products of one great vein-forming period beginning immediately after the appearance of the Anaconda fractures, and ending with the completion of the ore bodies of the Steward vein system. It is not improbable that during this period there were some interruptions and possibly at times ore deposition practically ceased, or that vein-forming action was more vigorous in some portions of the district than in others. The fracturing of early-formed ore with subsequent filling of such fractures with new minerals is not believed to mark distinct periods of vein-forming action, but rather to indicate faulting movement during the active deposition processes. It is reasonable also to infer that the older fissures became partly or wholly plugged or sealed by minerals, at certain places. The result of subsequent fracturing by faulting might merely divert ore-bearing waters from localities where they were yet active into new fractures in the formerly plugged portion of the veins.

Proceeding on the assumption that the primary vein minerals were derived from a common source at relatively great depths below where they are now found, it is believed that the broad, general, orderly arrangement of the ore types both vertically and laterally is due in a large measure to the variable temperature and pressure conditions encountered by the vein-forming waters along the lines of travel toward the surface, and to changes effected in the chemical composition of these transporting waters through their action on the granite wall rock. Moreover, there were, no doubt, wide variations in the metal content of the vein-forming waters from time to time. It is believed that the minerals of the veins were more soluble in strongly alkaline sulphide solutions than in hydrogen sulphide solutions under conditions of constant temperature and pressure, but in either alkaline sulphide or hydrogen sulphide solutions they were more soluble in hot than in cold waters.

The areal distribution of the mineral types found in the oldest vein system (the Anaconda) is believed to be due, in part, to the relative solubility of these various minerals in the original uprising waters from which they were deposited. It has been previously pointed out that these early solutions were carriers of alkaline sulphides due in part to the extraction of sodium and calcium from the granite. Under conditions of high alkalinity and elevated temperature, quartz and pyrite were deposited in abundance, forming the massive quartz-

pyrite veins of the central and intermediate zones. Under high temperature conditions, the minerals, sphalerite, rhodochrosite, galena, and chalcopyrite were more readily soluble and migrated outward to the intermediate and peripheral zones, where they were deposited along with quartz and pyrite, forming the primary vein filling of the so-called manganese-silver veins. During the period of migration of these solutions from the hotter zones the solutions lost none of their alkalinity, but the temperature was greatly reduced. These conclusions are drawn from the fact that there is no evidence to indicate that calcium and sodium were precipitated; in fact, slight alteration of the normal granite along the veins indicates a further addition of these elements to the vein-forming waters; and furthermore, the probable occurrence of alkaline carbonates is shown by the abundance of rhodochrosite in the border zones. The lower temperature conditions and comparative inactivity of the solutions toward the granite in the outer regions are well shown by the slight alteration of the wall rock adjacent to the fissures. In great veins a hundred feet or more in width in the manganese-silver-district, alteration of the granite extends for only a few feet outward from the general vein boundaries.

These early solutions contained some copper, as shown by its almost universal presence in the oldest known quartz-pyrite vein material and in the disseminated pyrite of the altered granite. Apparently no copper minerals were deposited in quantity in the earliest stages of vein formation, except possibly the chalcopyrite of the manganese-silver veins, but enargite is known to have formed in considerable abundance in the Anaconda veins prior to the appearance of the Blue fracture system, not, however, until the old veins were well formed. Enargite is therefore regarded as a relatively high-temperature mineral, but it probably formed under less alkaline conditions than existed in the early stages of the process, when quartz and pyrite were first deposited. The absence of notable quantities of other minerals containing copper or arsenic associated with this early enargite vein filling implies that these two elements were present approximately in the necessary proportions to form enargite. Copper was probably in excess, otherwise arsenopyrite would have been formed in cooler regions. Chalcocite did not form from the copper excess owing to the alkalinity of the solutions. It is probable that some bornite was formed at this early period, as was chalcopyrite in the outer zones of lower temperatures.

During the later stages when the Blue and Steward fault vein ores were formed, enargite, bornite, and chalcocite were deposited in large quantities in veins of all ages within the central and intermedi-

ate zones. In all of these veins the same order of deposition in noted; that is, pyrite and quartz were the oldest minerals, followed by enargite, sphalerite, bornite, and chalcocite, in the order named, chalcocite being associated with the older vein minerals only where the granite is intensely altered.

The general relationships above noted between the various ore types and conditions of granite alteration tend to show that under conditions of high alkalinity and high temperature and pressure, the vein minerals first deposited were principally pyrite and quartz. That this condition prevailed in the early stages of mineralization in fractures of all ages, except where the later fractures passed through areas of granite previously altered, is shown by the universal priority of a portion of the quartz and pyrite. As previously stated, the solutions became less alkaline, with a corresponding increase in the proportion of hydrogen sulphide present. It is known that the metals common to Butte, such as iron, copper, zinc, and manganese, are soluble in sodic sulphide solutions, but less so in hydrogen sulphide solutions; in fact, when present in sufficient proportions, hydrogen sulphide acts as a precipitant for these metals. The change of solution from a highly alkaline condition to one in which hydrogen sulphide predominates is brought about through conditions approaching chemical equilibrium between the uprising solutions and the granite wall rock. That chalcocite is a late mineral in the ores is well known. The explanation may be found in the suggestion here offered that it was deposited in the Butte veins only when the amount of hydrogen sulphide present in the vein-forming waters was large in proportion to the alkaline sulphides present. Observed conditions in the veins tend to support this view. Regardless of the geologic age of the fissure, quartz and pyrite are found to be the first minerals to form, chalcocite was one of the latest, and enargite, sphalerite, and bornite were found in intermediate stages. Quartz and pyrite, however, are formed under all conditions and, therefore, are intimately associated with minerals of the later stages, including chalcocite.

The deposition of chalcocite, the cuprous sulphide of copper ($Cu_2S$), directly from ascending waters, seems well within the range of possibility, considering that it is well known, or at least it has been frequently stated,[11] that the minerals enargite[12] and bornite, admittedly primary[13], contain the cuprous sulphide molecule $Cu_2S$.

[11] Dana's *System of Mineralogy*, pp. 76 and 147 (6th ed., 1892).

[12] Kirk, C. T., Conditions of Mineralization in the Copper Veins at Butte, Mont., *Economic Geology*, vol. vii., No. 1, p. 82 (Jan., 1912).

[13] Graton, L. C., The Sulphide Ores of Copper, *Bulletin* No. 77, May, 1913, p. 760.

With an excess of copper present in the cuprous form, over and above that required to satisfy the enargite molecule, it would seem reasonable to look for the deposition of chalcocite or bornite under favorable conditions.

Many minerals have undergone alteration, particularly the copper minerals, enargite and chalcocite. These alterations have been effected by primary processes. Enargite frequently alters to or is replaced by chalcopyrite and bornite, and chalcocite commonly exhibits slight alterations to bornite and rarely to chalcopyrite. Such alterations are believed to be due to the instability of the minerals under changing conditions in the solutions.

*Summary of Ore Genesis.*

The original source of the ores at Butte was the granite magma. Quartz-porphyry dikes formed a local closing phase of the igneous activity connected with the intrusion of the parent rock, and these dikes structurally and areally are in such close association with the ore deposits that they appear to be a direct factor in the localization of the ores. Heated waters and gases escaping from the cooling magma were the carriers of the metals to their place of deposition. The elements thus transported and deposited in the veins were silicon and oxygen as $SiO_2$, sulphur, iron, copper, zinc, manganese, arsenic, lead, calcium, tungsten, antimony, silver, gold, tellurium, bismuth, and potassium. Small quantities of potassium are believed to be added to the granite in the sericitization process.[14] Other elements, as sodium, calcium, and manganese, were undoubtedly carried by these solutions, but, as shown by analyses, they were extracted from the granite in the alteration process instead of being added as in the case of the first-named elements.

The chemical composition of these ascending waters varied in significant particulars as the process progressed. The granite wall rock was decomposed, furnishing much sodium, calcium, and possibly magnesium to the solution. Iron was also freed from the iron minerals of the granite to form pyrite with the sulphur of the invading waters. These interchanges affected the solvent capacity and character of the ore-bearing waters by the subtraction of the acid radical sulphur and the addition of alkaline radicals. While hydrogen sulphide and acidic conditions may have prevailed at the initial stages of ascent, the waters would tend to become alkaline through interaction with the wall rock. Along circulation channels, however, this

[14] Kirk, C. T., Conditions of Mineralization in the Copper Veins at Butte, Mont., *Economic Geology*, vol. vii., No. 1, p. 67 (Jan., 1912).

action would gradually become less pronounced after a barrier built of sericitized granite had been formed bordering the fissures, thus protecting the solutions from further reaction with the fresh granite, and permitting the acidic conditions to ascend to higher horizons. Also, the earliest vein minerals, chiefly quartz and pyrite, would tend to insulate the solution from the granite. And finally, increasing alkalinity of the solutions and lower temperature would lessen action on the granite at points further removed from the central source.

Applying the above reasoning to the facts of ore occurrence, it is found that chalcocite as a primary mineral is the latest important copper sulphide of the ores; it is, moreover, found only in association with the highly altered phases of the granite. From these facts the conclusion may be drawn that under the geologic conditions existing in Butte, the more acidic conditions were necessary for the deposition of this mineral. Similarly, enargite is associated with highly sericitized granite, and is therefore believed to have been deposited only under certain conditions pertaining to the temperature and relative alkalinity of the solution.

Sphalerite, rhodochrosite, and galena are increasingly abundant toward the intermediate and peripheral zones, suggesting their formation under lower temperature conditions with relative high alkalinity. Quartz and pyrite are everywhere present, and evidently are formed under all conditions. Pyrite is more abundant in the central and intermediate zones than in the peripheral zone. Quartz is more prominent as a gangue mineral in the peripheral zone than elsewhere.

Structurally there is no good evidence for distinct periods of mineralization in the Butte veins. It is here held that there was but one period of mineralization, varying in intensity, possibly, from time to time, with important changes in chemical character of solutions. But the mineralogical difference in vein material of the central, intermediate, and peripheral zones can be adequately explained, it is believed, by the reasoning herein set forth, which assumes that the copper mineralization indicates high temperature and acidic conditions versus lower temperature and alkaline conditions as the solutions migrated toward the peripheral fractures now represented by the manganese-silver veins.

Concerning the formation of chalcocite there is much geologic evidence, mainly structural, to support the theory above outlined, which assigns to this mineral a primary origin from deep-seated waters. The subject of chalcocite formation is of exceptional interest and well deserves special treatment in connection with the geology of the Butte

copper deposits. The evidence which tends to support the primary chalcocite theory held by the writer is briefly outlined in the chapter which follows.

*Origin of the Butte Chalcocite.*

Owing to the persistence to great depths of the mineral chalcocite in the Butte copper veins much interest has been aroused among geologists concerning the manner in which it was formed. In recent years the opinion has been quite generally held that chalcocite is largely, if not wholly, a product of descending sulphide enrichment. This view arose naturally through the discovery of the so-called "black sulphurets" (later proved to be sooty chalcocite) of Ducktown, Bisbee, and similar pyritic ore bodies. These belts of black amorphous chalcocite were found separating the oxidized zone from the lean pyritic ore below and they were early believed to have resulted from the reaction between the descending copper sulphate waters and the unchanged primary ores below. That this view was the correct one for the sooty chalcocite of this class of deposits has been abundantly proved by recent investigations.

The discovery of similar chalcocite ores in the early mining operations at Butte led many observers to the opinion that these remarkably rich ores were likewise of secondary origin and of limited vertical extent. When the zone of sooty chalcocite was penetrated, however, the predicted lean cupriferous pyrite ore was not found, but chalcocite-bornite-enargite ores were encountered, which have persisted to great depths. The chalcocite of the deeper levels does not occur in the sooty form, but instead, it is the gray massive mineral more or lese intimately mixed or intergrown with bornite, enargite or other ore minerals replacing directly altered granite. It is not necessarily a replacement of pyrite or any other sulphide mineral, being deposited directly from solution as chalcocite in veins along with bornite and other copper sulphides.

The problem of the formation of the chalcocite in the Butte veins was studied recently by C. T. Kirk [15], who endeavored to work out a definite relation between the chalcocite deposition and certain stages of granite alteration. He concludes that such a relation exists, and that the chalcocite formation is, in the main, associated with a certain phase of granite alteration which has developed through the action of descending meteoric waters. Weed,[16] in his recent report on Butte,

[15] Kirk, C. T., Conditions of Mineralization in the Copper Veins at Butte, Mont., *Economic Geology*, vol. vii., No. 1, pp. 35 to 82 (January, 1912).

[16] Weed, W. H., Geology and Ore Deposits of the Butte District, Montana, *Professional Paper No.* 74, *U. S. Geological Survey*, p. 76 (1912).

the district. In the central copper zone, or in veins where alteration has reached an advanced stage, these two minerals are commonly found in intimate association with other vein minerals forming the ores, and, in addition, small veins or stringers, largely chalcocite and enargite, may be found extending outward into and replacing the wall rock, the granite being in actual contact with the copper minerals. In areas of unaltered granite, chalcocite and enargite are confined to highly altered zones within or along the veins, and never as stringers or disseminations inclosed by unaltered granite. It is improbable that solutions bearing these minerals did not find passage outward into the normal granite during the active vein-forming period, when chalcocite, bornite, and enargite were being deposited in abundance in the veins. The frequent occurrence of stringers of pyrite, quartz and sphalerite extending into wall rock of normal granite in the intermediate zone, with addition of manganese minerals in the peripheral zone, indicates that the mineral-bearing waters did to some extent find their way into the wall rock, but only where alteration was marked were chalcocite and enargite deposited contemporaneously in the veins proper and in the altered granite wall rock.

Applying a similar method of reasoning to the remaining important minerals of the veins, it is observed that quartz and pyrite are found abundantly in association with all stages of rock alteration and with all periods of primary ore deposition, although there is a noticeable decrease in the proportionate amount of pyrite in the intermediate and peripheral zones as compared to the central zone. It is true also that there is proportionately less pyrite in the Blue vein ores than in the Anaconda veins, and markedly less in the Steward ore shoots than in the Blue veins, an indication that the later vein-forming waters either contained less pyrite or that conditions were less favorable for deposition. These later ores are quite certainly less rich in pyrite for reasons previously stated. Quartz and pyrite are thus found to have been deposited under widely varying conditions of the solutions, such as composition, temperature, and pressure. Sphalerite is rarely found in zones of most intense alteration. It is sparingly developed in the slightly less altered areas of the Mountain Con and Diamond mines, and is very abundant in veins occurring in regions of relatively unaltered rock. It is believed to have been deposited under conditions of alkalinity similar to those under which enargite was formed, but generally at lower temperatures. Being more soluble, it would tend to migrate in solution before deposition through greater distances than would enargite.

Galena is a relatively uncommon mineral, associated usually with sphalerite. It is not found in the central copper zone. The manganese mineral, rhodochrosite, is an abundant constituent of the veins of the peripheral zone, and is not uncommon toward the outward limiting boundaries of the intermediate zone. It is entirely unknown in the central zone. Chalcopyrite is not an uncommon mineral associated with sphalerite, quartz, and galena in the manganese-silver veins.

In an endeavor to explain these general relationships between the various vein minerals and altered granite areas, certain assumptions must be made in order to form a working hypothesis. It is believed by the writer that the ores of Butte in their entirety have been derived from one general centralized source at great depths. There is no evidence in support of, and much evidence contradicting, the hypothesis that the so-called manganese-silver or zinc veins belong to a vein-forming epoch separate and distinct from that in which the copper veins were formed. Going outward from the central area (see Fig. 7) of typical zinc-free copper ores, there are no instances of early copper veins being cut by later veins of different mineralogical character, which would naturally be the case if, for example, the Blue fissures were of a distinctly later period than the copper veins of the east-west system. It has been suggested also that the manganese-silver ores represent an invasion of an older copper area, or *vice versa;* this is also untenable, for the reason that it is an unlikely assumption that a later mineralization period could almost completely surround a central core, depositing new mineral in fissures identically the same age as those of the central area, without some of the later ore types appearing within the inclosed zone. It has been shown that the mineral composition of the different vein systems is not a result of separate periods of mineralization, but rather of geographic position. The structural relations point strongly to the conclusion that the east-west fractures of the manganese-silver area are of the same age as the Anaconda fractures of the copper area, and that they were well mineralized by quartz, pyrite, rhodochrosite, and sphalerite at the time they were cut by fissures of the Blue series. The Blue veins, furthermore, with their typical silver mineralization of the peripheral zone, continue southward into the copper area, cutting the oldest copper veins, and there contain valuable copper ores and are characteristic copper veins. Fissures of the Steward system are also found in both the copper and the silver areas, cutting copper ores in the former case and zinc ores in the latter. Therefore, while some copper ore bodies are found in these fissures, and zinc mineral-

time elapsing between the beginning of Blue vein movements and the beginning of Steward faulting), it is fair to assume that the ground surface was much higher than at present, necessitating, therefore, a former extremely deep meteoric ground-water circulation to reach chalcocite ore bodies of the Blue veins now found more than 3,000 ft. from the surface. When one considers the rate of the downward invasion of the oxidized zone, it is almost inconceivable that down-seeping sulphate waters could have formed the extensive chalcocite ore bodies found at these depths. The time required would be enormous, and, furthermore, the fact must not be lost sight of that under conditions favorable for sooty chalcocite formations, as we know them, a very large part indeed, if not all, of the copper of the descending sulphate waters is deposited as secondary chalcocite before a maximum depth of 1,200 ft. below the zone of oxidation is reached.

There is another important point inviting attention, relative to the probable condition of the underground circulation existing during the time of formation of the Blue vein ores and during subsequent periods extending to the present time. It is a self-evident fact that meteoric waters could not have descended to great depths along veins, faults, or fissures at a time when appreciable quantities of waters, presumably deep-seated, were ascending through such channels. It is a reasonable assumption, then, that no important downward movement of meteoric waters took place in the Butte fissures until after the cessation of movement of the uprising solutions from which were deposited the primary ores. It is not unreasonable to believe that some surface waters did reach these channels of uprising waters at comparatively shallow depths, not, however, by direct descent along fissures through which deep-seated waters were ascending, but by a downward-lateral movement through neighboring fissures adjacent to the main trunk channels. As has been formerly pointed out, however, the movement of cold surface waters through normal granite is scarcely appreciable, and it is therefore extremely improbable that such waters could have influenced chemically, physically or in any way whatsoever the action of the deep-seated waters as they moved upward through the fissures depositing minerals undoubtedly derived from deep-seated sources. The occurrence of undoubted primary ores, or quartz, pyrite, sphalerite, galena, and rhodocrosite, together with enargite, bornite, and chalcocite, in faults of the Steward system which are known to cut and displace chalcocite ore bodies in the Blue and older vein systems, is conclusive proof that ascending solutions depositing primary ore continued in action long after the formation of the Blue vein chalcocite. It is probable that ascending waters

continued to traverse the Steward and older fissures for a considerable length of time after the primary Steward ores had formed; in fact, it is not at all improbable that the alteration of the granite and deposition of pyrite in the Rarus fault resulted from ascending waters.

If an early circulatory system existed similar to that above assumed, it is difficult to understand how meteoric waters could have been active enough to transport large quantities of mineral from the oxidized zone to great depths at any period prior to the complete cessation of the upward movement of the primary ore solution. The time of cessation must have been as late as the end of the ore-forming period of the Steward fault veins, and possibly as late as the Rarus fault, both of which are known to be later than much of the chalcocite of the Blue and Anaconda veins.

The chalcocite of the deeper levels, or in a general way the massive chalcocite of all the veins, bears no definite relation to the present surface topography. This is in marked contrast to the occurrence of sooty chalcocite known to be of secondary origin. The tops or apices of many rich, massive, chalcocite ore bodies or shoots are found at depths ranging from 100 to 1,500 ft. from the surface. The size and richness of the ore body are in no way indicated by the depth of the zone of oxidation; in fact, many of the largest ore shoots of the fault veins are capped by from 500 to 800 ft. of barren crushed granite and fault clay having an oxidized zone of less than 25 ft. in vertical extent. In the quartz-pyrite veins of the Anaconda system where the development of secondary glance is greatest, there is an apparent close relation between the depth of oxidation and the quantity of sooty chalcocite found below. A deep zone of secondary chalcocite is certain to be found below a deep zone of oxidation, while a shallow zone of oxidation is accompanied by an unimportant development of sooty chalcocite below. If a secondary origin is assumed for the chalcocite ore bodies whose tops or apices are separated from an extremely shallow oxidized zone by hundreds of feet of barren crushed granite, the question as to the source of the copper to form the chalcocite becomes of vital interest.

Where the ore shoots do not extend upward to the oxidized zone it does not appear possible that the source of the chalcocite could have been at a point higher than the top of the ore shoot, for there is no evidence, direct or otherwise, that copper-bearing mineral of any character ever existed in the eroded and oxidized portions of the vein. The marked absence in the upper portions of many veins of an adequate source of supply for the copper found at great depths in the form of chalcocite is a common feature of these ore deposits. This

is true not only of the fault veins, but in many of the veins of the Anaconda system. In the Tramway and Leonard mines, for example, immense chalcocite-enargite ore bodies from 50 to 200 ft. in thickness, belonging to the Anaconda vein system, have been developed between the 1,200 ft. and 2,000-ft. levels. From the 1,200-ft. level to the surface these ore bodies are represented only by small veins from 2 to 6 ft. in thickness, carrying but small amounts of copper. Indeed, in many instances the identity of the vein is entirely lost as the higher levels are approached. In nearly every case the oxidized zone capping the big ore bodies of this section is shallow, and even if it be assumed that hundreds of feet of vein have been eroded, such eroded portions represent a source entirely inadequate to account for the chalcocite found below.

In the Shannon vein (belonging to the Anaconda system) of the West Colusa mine, the ore bodies of the upper levels were of tremendous size, particularly in the region immediately underlying the oxidized zone, where there occurred a big development of sooty chalcocite. It is significant that although the vein continued big and strong in depth, with every condition favorable for secondary enrichment, the vein became poor rapidly in depth, portions of it at the 900-ft. level being too low grade to mine. Many other examples might be mentioned where old quartz-pyrite veins have been broken by later faults, and all conditions seem ideal for the formation of chalcocite ores in depth, but, other than the sooty glance enrichment, no notable addition of chalcocite has taken place.

An interesting occurrence of chalcocite is found in the Mountain Chief ore shoot of the Jessie vein, belonging to the Blue fault system. Plate V., a longitudinal projection of the vein, has been prepared to show the forms and positions of the various ore shoots. As will be noted, the oxidized zone is extremely shallow, being not more than 25 ft. deep at any point in the vein. The ore shoots have been opened by continuous workings from the surface to the 2,200-ft. level.

There is a marked difference in mineralogical composition of these ore shoots between their upper and lower portions. The change is found to take place at a depth of from 500 to 800 ft. from the surface. Some of the shoots do not extend entirely to the oxidized zones. The line *A–B* is drawn to mark approximately the elevation at which the change takes place. Above *A–B* the ore is an intimate mixture of quartz, pyrite, and chalcopyrite, the latter mineral occurring in abundance. Sphalerite and rhodochrosite are also present in considerable amounts. In the Mountain Chief mine one shoot extends entirely to the surface, where it is oxidized to a rich ore com-

posed of cuprite and iron oxides. Immediately below these rich oxides there is but a slight development of secondary chalcocite, occurring as thin films coating the chalcopyrite and pyrite. At about the line *A–B*, within a vertical distance of from 50 to 75 ft., there is almost a complete transitional change from chalcopyrite ore to an ore consisting of chalcocite, bornite, enargite, pyrite, and quartz, with only small amounts of chalcopyrite. This character of mineralization has continued to the deepest levels yet opened, although there are some variations in the relative amounts of the minerals present. The development of chalcopyrite seems to take place only in the high levels or at the waning ends of the ore shoots, indicating possibly that under certain conditions it is a lower-temperature mineral than either enargite or chalcocite, assuming for the moment that all three are here of primary origin.

In this particular example it is impossible to conceive of a surface water origin for the chalcocite lying below the chalcopyrite capping. There certainly is no apparent adequate source for the copper. Where the chalcopyrite ore suffers oxidation the larger part of the copper is held in the oxidized zone as an oxide or carbonate, and, even assuming that a part of the copper was carried downward, it is quite impossible for the writer to believe that it could have remained in solution while passing downward over the chalcopyrite-pyrite ore, to be later deposited as chalcocite from 800 to 2,200 ft. below the surface.

As already stated, many of the Butte veins have but slight oxidized zones, accompanied by sooty chalcocite zones of small vertical extent separated by hundreds of feet of barren vein from the chalcocite ores below. In such instances it is impossible to trace any genetic relation between the meteoric water circulation and the chalcocite commonly occurring at depths greater than 1,000 ft. Where, however, as in the case of most of the quartz-pyrite veins of the Anaconda system, an important chalcocitization zone exists associated with massive chalcocite ores, and is underlain at greater depths by large quantities of chalcocite not associated with sooty chalcocite, it is, perhaps, reasonable upon first thought to suppose that all the chalcocite of both higher and lower levels has had a common origin. The early prominence of the copper veins of this class has been largely responsible for the former general belief in a secondary origin for the chalcocite in the Butte veins.

The changes which occur in the oxidized zone of the old copper-bearing quartz-pyrite veins in Butte are due to processes which act slowly. The invasion of the oxidized zone downward into the sulphides took place at an extremely slow rate, and in view of this fact it

must be admitted that the sulphate solutions originating at the sharp contact between the oxides and sulphides were extremely dilute. The chemicals added to these downward-seeping surface waters through the oxidation of the sulphides are chiefly copper and iron sulphates, and possibly small amounts of free sulphuric acid. These cold surface waters, after taking up their burden at the oxide-sulphide contact, pass immediately downward along the vein, or partly through country rock, moving more or less constantly in direct contact with the pyrite and other vein sulphides. As is well known, the reaction between the sulphides and sulphate waters results in the formation of sooty chalcocite as a direct replacement of the sulphide attacked. These sulphate waters also attack the altered granite, resulting in greater porosity and in the formation of abundant kaolin. There is also a chalcocitization of the disseminated pyrite so common in the altered granite. In meeting already existing ground-waters below, a large proportion of which, although of meteoric origin, did not take copper into solution on their downward journey, the descending waters along the veins must become more and more dilute and certainly less active chemically as greater depths are reached. As a matter of fact, actual comparisons of the veins and granite of the upper and lower mine levels show conclusively that the downward-seeping waters actually become weak and inactive at not great depths below the surface, and it was due to this fact, in part at least, that C. T. Kirk was able to differentiate so clearly between the chloritic, sericitic, and kaolinitic alteration phases in the Butte granite.

It is extremely important to understand clearly this feature, because, apparently much more vigorous chemical processes have been active in the formation of the massive chalcocite of the deeper levels than were necessary for the formation of the secondary chalcocite of the higher levels. In the sooty chalcocite zone only sulphides are attacked and replaced by the chalcocite, while at greater depths massive chalcocite alone, or intimate mixtures of chalcocite, pyrite, bornite, and enargite, directly replace altered granite in quantities within and along the fault veins and veins of the oldest system. The writer believes that such replacements could not have been effected by dilute meteoric waters, which, in the act of reaching great depths, not only became extremely dilute and of doubtful activity, but they have been deprived wholly or in part of their copper in the regions of sooty chalcocite formation.

The observed facts which have led primarily to the belief that the chalcocite of the Butte copper deposit is of secondary origin may be briefly stated as follows:

1. The chalcocite is often of a later age than the vein minerals with which it is associated.

2. In some instances the proportionate amounts of chalcocite in the veins have decreased rapidly with depth.

3. It has been abundantly proved at Ducktown, Morenci, Bingham, and in many other instances, that under certain conditions chalcocite is a product of descending sulphate waters.

4. The depth of the chalcocite enrichment in many of the Butte veins bears a definite relation to the depth of the zone of oxidation in the respective veins.

In an endeavor to solve the problem of the chalcocite formation, some investigations have been made. H. V. Winchell[18] succeeded in producing artificially, under normal conditions of temperature and pressure, chalcocite identical in chemical composition and physical character with the sooty chalcocite of the Butte veins. His experiments seem to prove conclusively that the formation of chalcocite, of the "sooty" variety, at least, is easily possible under the conditions of temperature and pressure found in the upper levels of the Butte mines. Similar conclusions have been reached by Stokes and others in the laboratories of the U. S. Geological Survey.

Perhaps the most elaborate investigation of this subject was undertaken by Charles T. Kirk[19] who made careful chemical and petrographic analyses of the altered granite occurring in and along the Butte copper veins. He found that during the vein-forming processes the granite suffered great changes in chemical and physical character. These changes took place more or less gradually. He separated them into three general alteration phases, namely: (1) the chloritic phase, which marks the earliest stage of alteration; (2) the sericitic phase, marked by the development of great quantities of sericite through the further action of heated waters in (1); and (3) the kaolinitic phase, a change from the sericitic phase brought about through the action of descending sulphate waters or sericitized granite.

Phases (1) and (2), therefore, result from the action of deep-seated ascending waters; (3) is effected by the action of descending cold meteoric waters on phases (1) and (2). With these three alteration phases Kirk links certain generalized groups of minerals. He believes the early quartz-pyrite ores began to form with the early chloritic phases; that the copper mineralization during this and the succeeding

[18] Winchell, H. V., The Synthesis of Chalcocite and Its Genesis at Butte, *Engineering and Mining Journal*, vol. lxxv., No. 21, pp. 783 to 784 (May 23, 1903).

[19] Kirk, C. T., Conditions of Mineralization in the Copper Veins at Butte, Montana, *Economic Geology*, vol. vii., No. 1, pp. 35 to 82 (Jan., 1912).

sericitic stage was principally enargite, bornite, and chalcopyrite; and lastly, that the chalcocite formation belongs entirely to the third or kaolinitic phase. He holds that kaolinite is wholly a product of cold meteoric water action and therefore the presence of it in deep levels indicates the presence of waters of meteoric origin. Many geologists, notably Gregory, dissent from this view and hold to the opinion that kaolinite may also be a product of ascending water alteration.

It is to be regretted that Kirk did not give a series of direct comparisons between samples of altered granite taken both from the sooty chalcocite zone and the deep levels. Certainly there is much yet to be learned concerning the relation between the altered granite and chalcocite formation. Even assuming for the moment that meteoric waters have sunk to great depth in the Butte veins, accompanied by the formation of kaolinite at all levels, it does not necessarily follow that the chalcocite was deposited from such descending waters. It cannot be doubted that the sulphate waters descended to depths greater than the lower limit of the sooty chalcocite zone, but it is evident that while the chemical effect of these waters upon the granite at greater depths may have been of the same general nature as in higher levels, that is, kaolinization, the chemical action toward copper mineralization was entirely different. The chalcocite of deeper levels is not necessarily a replacement of a sulphide mineral as in the upper levels. Since the replacement of the pyrite by chalcocite in the higher zones is accompanied by the formation of ferrous sulphate and sulphuric acid, the descending waters passing below the sooty chalcocite zone still retain an abundance of dissolved iron sulphates and probably sulphuric acid to act on the seriticized granite, forming kaolin, as in the higher levels, but it is more than probable that the descending waters were entirely robbed of copper in the secondary chalcocite zone. The small amount of kaolinite present in the deeper levels as compared with the great abundance in the oxidized and sooty chalcocite zones indicates less activity, due either to dilution or to change in chemical composition of the solution.

From these considerations it is readily seen that the association of chalcocite with minor amounts of kaolin below the chalcocitization zone does not necessarily imply that both have resulted from the same solutions. The descending sulphate waters may still continue to form kaolin in regions of primary chalcocite after having deposited all of the copper burden in the region immediately below the zone of oxidation. It may not be difficult to understand meteoric waters reaching to unusual depths in the Butte veins, but the writer seriously doubts that such waters could retain copper in appreciable

quantities after moving downwards for hundreds of feet in direct contact with newly formed chalcocite and an abundance of pyrite.

It is unfortunate that most of the granite samples used by Kirk in his investigations were collected from the Pittsmont vein, for the reason that this property is peculiarly situated with respect to the general topography of the district, and it is also in close proximity to the Continental fault, a fracture of comparatively recent occurrence. The effect of this fault upon the water level and oxidized zone may be readily understood by reference to Plate IV. It will be seen that the immediate effect has been to drop the former erosional surface and oxidized zone to a depth considerably below their former positions. The collar of the Pittsmont shaft at the present ground surface is about 400 ft. above the old erosional surface. In the mine workings the oxidized zone is from 250 to 300 ft. thick, measured downward from the former surface, and the sooty chalcocite belt is known to extend at least 500 ft. deeper. Summing up these figures, the result is reached that the deepest general working level of this mine (the 1,200 ft.) is not more than 600 ft. below the zone of oxidation, or, as a matter of fact, scarcely below the zone of sooty chalcocite. When it is remembered that in the Mountain View mine the sooty chalcocite zone is from 800 to 1,200 ft. thick, one is forced to the conclusion that Kirk's samples do not represent conditions far removed from the direct influence of copper-bearing surface waters. The results of his work are extremely interesting and of value, especially his investigations concerning the alteration of the granite, but in the opinion of the writer he has erred in attempting to apply his method of reasoning to the chalcocite of deep ore bodies of the Butte veins with which he is evidently unfamiliar. His results are valuable inasmuch as they further corroborate and establish, from a new point of attack, the conclusions already reached by others that the sooty chalcocite, and massive chalcocite to a limited extent, of the Butte deposits, have resulted from the work of downward-seeping sulphate waters whose copper was derived from the oxidized zone.

W. H. Weed[20] has set forth some facts which, in his opinion, tend to prove the secondary origin of the Butte chalcocite. He observes generally that the old quartz-pyrite veins were originally of very low grade and they became commercially valuable through the later addition of enargite, bornite, chalcocite, and other copper minerals. He believes that this copper mineralization followed various periods of faulting, the enargite and bornite being the first to appear, probably

[20] Weed, W. H., Geology and Ore Deposits of the Butte District, Montana, *Professional Paper No. 74, U. S. Geological Survey*, p. 152 (1912).

contemporaneous in a general way with the Blue and Steward fault system. Chalcocite, which forms the bonanza ores of the district, is thought by him to have been almost entirely a product of descending sulphide enrichment processes, acting at great depths, however, only where the older quartz-pyrite veins were crackled and broken by faults, thus permitting a ready passage for the downward-seeping waters. He cites many examples of such intersections of faults and older veins in support of this view, and maintains that the old quartz-pyrite veins are workable only where thus fractured.

The writer's own observations do not confirm Weed's conclusions as above outlined. Actual examination of a great many intersections of old quartz-pyrite veins by later faults have shown conclusively that as a general proposition the east-west veins are no richer at or near intersections with Blue vein faults than at other points along the vein except in cases where the fault vein ore shoots cross the older vein. It is extremely difficult to form even an approximate idea as to the extent of primary enrichment in the older veins due to the late faults of the Steward system. Mineralization processes were active in the early veins prior and subsequent to the Blue vein period, so that it is impossible to determine, in the absence of any characteristic minerals, what influence was exerted by the later faults upon the older veins. As might be expected, the fault vein intersections are usually accompanied by a breaking and shattering of both the older vein and the country rock in the immediate vicinity, thus developing favorable factors tending to greatly influence ore deposition at such points. In any case, where a chalcocite enrichment of a vein of the Anaconda system is shown to have resulted from the influence of an intersecting fissure of the Blue or Steward system there remains the strong probability that such enrichment is due to primary waters, if, as believed by the writer, the primary chalcocite was deposited in great quantities, after the appearance of these faults, not only within the faults themselves, but in the fractured older veins.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.
[SUBJECT TO REVISION.]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Butte meeting, Aug. 18 to 21, 1913, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1913, when Vol. XLVI. of the *Transactions* will go to press. Any discussion offered thereafter should preferably be in the form of a new paper for publication in Vol. XLVII. (with suitable cross references in both volumes).

# Mineral Associations at Butte, Mont.

BY D. C. BARD AND M. H. GIDEL, BUTTE, MONT.

(Butte Meeting, August, 1913.)

THESE notes are based on the megascopic study of a suite of 2,400 specimens of minerals and ores from the Butte mines, combined with field observations at intervals over a period of several years.

The estimate of average vein composition is based on field acquaintance with the veins. It is probably correct in its broader proportions, and should be of use for comparison.

*Chemical and Mineral Composition of Butte Sulphide Vein Matter.*

| | Per Cent. | |
|---|---|---|
| Oxygen, | 48.00 | —Quartz, rhodonite, rhodochrosite, barite, calcite, and hübnerite. |
| Silicon, | 36.00 | —Quartz, rhodonite, and willemite (very rare). |
| Sulphur, | 5.00 | —Sulphides of iron, zinc, copper, lead, and sulphates of barium and rarely calcium. |
| Iron, | 5.00 | —Ferric sulphides (pyrite, bornite, chalcopyrite), also in sphalerite, tetrahedrite, and rarely hematite and siderite. |
| Manganese, | 2.00 | —Rhodonite, rhodochrosite, and hübnerite. |
| Zinc, | 1.00 | —Spha erite, very rare wurtzite and willemite, and tetrahedrite and tennantite. |
| Carbon, | 0.50 | —Rhodochrosite and calcite. |
| Copper, | 0.75 | —Cuprous (+) and cupric (—) sulphides. |
| Aluminum, | 0.50 | —Sericite and kaolinite. |
| Hydrogen, | 0.50 | —Water, sericite, and kaolinite. |
| Potassium, | 0.20 | —Sericite. |
| Lead, | 0.20 | —Galena. |
| Arsenic, | 0.10 | —Tennantite and enargite. |
| Barium, | 0.10 | —Barite. |
| Calcium, | 0.10 | —Fluorite, calcite, and rare gypsum. |
| Antimony, | 0.02 | —Tetrahedrite, famatinite, and silver sulphantimonites. |
| Fluorine, | 0.05 | —Fluorite. |
| Silver, | | —Argentite, and sulphantimonites. |
| Tungsten, | | —Hübnerite. |
| Phosphorus, | | —Wavellite. |
| Bismuth, | 0.05 | —Sulphide (?) |
| Tellurium, | | —Telluride (?) |
| Selenium, | | —Selenide (?) |
| Cadmium, | | —Sulphide (?) |
| Gold, | | —Native and telluride (?) |

The area considered in these notes is limited by the Blue Bird mine on the west, Walkerville on the north, the Pittsmont mine on the east, and Silver Bow creek on the south.

Only such minerals are considered here as are thought to have some bearing on the genesis of the ore deposits. The oxidized zone minerals are in general omitted.

*Mineral Composition in Relation to Chemical Elements in the Vein Matter.*

*Iron* is deposited entirely as ferric sulphides.

*Manganese* is deposited as carbonate and silicate, rather than as a sulphide, possibly because of insufficient sulphur to satisfy all the bases.

*Zinc* is deposited solely as sphalerite except very rarely for some willemite. The absence of appreciable willemite suggests sufficient sulphur to satisfy the zinc. Wurtzite has been found in the upper levels of the Gagnon mine.

*Copper* is usually deposited as a cuprous compound, rarely as the cupric covellite.

*Arsenic* is deposited solely as the sulpharsenate, enargite, and the sulpharsenite, tennantite. Arsenopyrite is absent, suggesting a stronger affinity of the arsenic for copper than for iron. Also simple arsenic sulphides are absent, suggesting that in the presence of excess of copper only the sulpharsenic compounds are formed. Arsenates are lacking in the oxidized zone.

*Aluminum* occurs in sericite and kaolinite, which formed in place from the feldspars of the granite walls. The alteration of the feldspars to sericite freed an excess of silica, which occurs as quartz disseminated through the sericite in addition to that originally indigenous to the granite. Rarely, aluminum has been deposited along fractures in the earlier formed vein matter as wavellite.

*Water* in the vein minerals is in very little if any greater quantity than that in the original granite.

*Lead* is found only as galena, the sulpho-salts of lead being absent.

*Calcium* occurs as fluorite, apparently being one of the earlier, high-temperature minerals. Gypsum rarely occurs in delicate crystals in vugs. Calcite is not uncommon.

*Antimony* occcurs in the sulphantimonites of copper and silver—never as stibnite.

*Tungsten* is found solely in hübnerite, usually in vugs.

*Bismuth, tellurium, and selenium* have been noted in the smelting operations, but have not been recognized in minerals. One specimen

of argentiferous chalcocite from the Gagnon mine assayed 0.5 per cent. of bismuth.

*Source of the Vein Materials.*

Most of the vein minerals contain elements foreign to the granite country rock, at least in the concentrated form found in the veins. Exceptions to these statements are the sericite and kaolinite which could result from the decay of the granite in place; and the wavellite which is locally concentrated, but no greater in total amount than could be supplied by the phosphorus in the apatite of the granite. The original granite contained more than sufficient calcium and barium in feldspars to satisfy the fluorite, calcite, gypsum, and barite of the veins.

Likewise, the granite could and probably did furnish 50 per cent. of the quartz of the veins. Also there is sufficient iron in the original granite to satisfy the average iron content of the veins, although the iron was subjected to concentration and alteration to the ferric condition. Furthermore, it may be that our estimate of average iron content is too low.

The elements which were foreign to the granite wall rock in any such amounts as are found in the veins are: Silicon, sulphur, manganese, zinc, carbon, copper, arsenic, lead, antimony, silver, tungsten, bismuth, selenium, tellurium, cadmium, and gold. To this list should be added oxygen, although there is nearly the same amount of oxygen in the granite as in the veins.

The elements which are found in the veins in no greater quantity than in the original granite are: Iron, aluminum, hydrogen, potassium, barium, calcium, fluorine, and phosphorus.

The elements which are found in greater quantity in the granite than in the veins, and which therefore have been in part or entirely removed by the vein-forming processes, are, in order of amount: Aluminum, calcium, sodium, magnesium, potassium, phosphorus, and titanium.

*Observed Paragenesis of the Vein Minerals.*

Quartz is found in all relations to the other minerals, being deposited throughout the mineralizing period. The only exceptions are that no quartz has been observed later than calcite or wavellite.

Fluorite was among the first minerals formed. It is earlier than rhodochrosite and sphalerite. But that it was forming over some length of time is shown by its occurrence in the east-west veins and in the later northwest-southwest veins.

Another early vein mineral is hübnerite. It occurs usually in vugs or in porous parts of the veins, associated with quartz. It is sometimes replaced by later pyrite, chalcopyrite, and enargite.

Molybdenite is sometimes given as a vein mineral from Butte, but has been observed by us only in the earlier pegmatite and aplike dikes which are not structurally connected with the vein systems. One occurrence of wulfenite has been noted in the oxidized zone associated with wavellite.

Rhodonite and rhodochrosite are found intimately intergrown and associated with quartz, sphalerite, and rarely with willemite. They have been noted later than some sphalerite, pyrite, galena, and chalcopyrite; also later than some sphalerite. Rhodochrosite earlier and later than bornite has been observed. The only evidence that some of the rhodochrosite may be later than the rhodonite is that the rhodochrosite is more often found along the fractures and walls of the veins. The evidence is not clear that rhodochrosite is an alteration product of rhodonite. We have no evidence that rhodonite becomes, with depth, more plentiful than rhodochrosite. They both appear to be primary minerals contemporaneously deposited. It seems to be a fact that the manganese minerals are less plentiful in the deeper parts of the veins. The manganese minerals are rare in the copper veins.

Barite is a primary mineral, widely distributed but in small quantity. It is sometimes syngenetic with the sulphide minerals, more especially the sulpho-salts, and sometimes later than them. One specimen shows barite with later quartz and tennantite.

Gypsum is very rare. It has been observed in long, delicate crystals in vugs in the deeper levels, and is evidently primary. It is too rare to have any paragenetic significance. Gypsum, secondary from fluorite, occurs in the oxidized zone.

Calcite is widespread, but not common. It is apparently among the last minerals to form.

Wavellite is a rare mineral found in crusts in older vein filling. No mineral has been observed later than it.

Vivianite occurs rarely in sheaf-like crystals in vugs. It is apparently a late mineral.

Pyrite is of all ages.

Marcasite is not common. Its distribution does not seem to depend on depth, as it has been found on the 1800-ft. level of the Badger vein. Here it seems to be later than the underlying pyrite-bornite-chalcocite ore.

Covellite in Butte is widespread in the copper veins, but is not plentiful. Certain veins carry larger amounts of it. It is generally crystalline and primary, being earlier or at least syngenetic with enargite and pyrite. One specimen shows covellite veinlets in enargite. It often alters to chalcocite, enargite, and chalcopyrite.

Enargite is one of the earlier vein minerals. In some cases it alters to tennantite. It frequently alters to chalcopyrite, and is also replaced by bornite. It is often syngenetic with bornite and chalcocite.

Chalcopyrite is not a common primary mineral. It is found syngenetic with bornite and pyrite. In the copper veins it is an occasional alteration product from enargite, chalcocite, covellite, bornite, and sphalerite. Chalcopyrite has been found deposited on pyrite. Tennantite deposited on chalcopyrite has also been observed.

Tennantite occurs as a late-formed mineral, particularly on the edges of the copper zone. It alters to chalcopyrite, and replaces sphalerite and galena.

Chalcocite is generally a primary mineral syngenetic with bornite, enargite, and pyrite. It also occurs later than these and replacing them. It also replaces sphalerite at times.

Tetrahedrite is not common. It appears to be an early, primary mineral, sometimes syngenetic with sphalerite. It alters to chalcopyrite.

Hematite has been found in one good specimen from the 2,200-ft. level of the Anaconda mine. It is associated with chalcocite, pyrite, and quartz, and appears to be syngenetic with them. It is rare.

Galena is common in the silver zone. It is syngenetic with sphalerite, chalcopyrite, and pyrite.

Sphalerite is common in the silver zone. It and galena have not been observed later than copper minerals except in one specimen of sphalerite in a vug from the 2,200-ft. level of the Speculator mine, which is inclosed by quartz, bornite, and enargite.

The mineralogical relations in the Butte veins do not suggest distinct and separate periods of mineralization, but rather one continuous period with varying degrees of intensity. But a small portion of the copper in the Butte veins seems to result from enrichment by descending, meteoric waters.

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**In order to have the papers to be presented at the Butte meeting in the hands of the members well in advance of the meeting, it was necessary to print this number of the *Bulletin* in two sections. The absence of pages 1633 to 1800 does not indicate the omission of any part of the contents of this *Bulletin*.**

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# The Great Falls System of Concentration.

## Installed in Section No. 1 of the Washoe Concentrator at Anaconda, Mont.

BY ALBERT E. WIGGIN, ANACONDA, MONT.

(Butte Meeting, August, 1913.)

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  - 3. Fine Middling Division, . . . 1821
  - 4. Table Division, . . . 1822
  - 5. Slime Division, . . . 1822
- *b.* Special Features of Great Falls Flow Sheet, . . . 1823
  - 1. Development of Hancock Jig Flow Sheet, . . . 1823
  - 2. Development of Finishing Table Flow Sheet, . . . 1824
  - 3. Treatment of Coarse Middling Table Products, . . . 1824
  - 4. Results of Test Made on 500-Ton Section at Great Falls Mill, . . . 1826

VI. Comparison of Great Falls and Evans Systems at Anaconda, . . . 1828

THE copper-bearing sulphide ores from the mines in Butte, Mont. which are for the most part concentrated at the Boston & Montana Reduction Works in Great Falls and at the Washoe Reduction Works in Anaconda, contain the following minerals of economic value: Chalcocite, $Cu_2S$; enargite, $Cu_3AsS_4$; cupriferous pyrite; bornite, $Cu_3FeS_3$; covellite, $CuS$; tetrahedrite, 4 $CuS_2Sb_2S_3$; and tennantite, $Cu_3AsS_7$. All of the ores carry a high percentage of pyrite. Many of the above sulphides are argentiferous and also carry from 10 to 20 cts. per ton in gold. The gangue is quartz and highly altered granite, with sphalerite and galena and more rarely barite and hubnerite as accessory minerals.

An average chemical analysis of the second-class ore concentrated at Great Falls and Anaconda would be as follows:

| | |
|---|---|
| Copper | 3.2 to 3.4 per cent. |
| Silver | 2.6 to 3.0 oz. per ton. |
| Gold | 0.07 to 0.15 oz. per ton. |
| $SiO_2$ | 55 to 60 per cent. |
| FeO | 12 to 15 per cent. |
| Sulphur | 12 to 15 per cent. |
| $Al_2O_3$ | 8 to 10 per cent. |
| CaO | 0.5 to 0.8 per cent. |
| $As_2O_3$ | 0.5 to 0.6 per cent. |
| Lead | 0.2 to 0.3 per cent. |

Before describing the Great Falls flow sheet as installed at Anaconda a brief history of the Great Falls concentrator will be given, including a description of the various concentrating machines and systems which have been tested in the search for a flow sheet which would give a higher recovery of copper.

## I. EARLY HISTORY OF THE BOSTON & MONTANA MILL AT GREAT FALLS

*a. Original mill.*—This mill was erected during 1891 and began to treat the ore from the Boston & Montana Co.'s mines in February, 1892. As first built, the mill consisted of three sections, rated at 800 tons each. The third section was not put into operation until about a year after the first two began to operate. At this time there were no blast furnaces at the Great Falls plant, all of the concentrate being roasted and then smelted in tilting furnaces. On this account the concentration did not begin until all the ore had been crushed through 0.5 in. Blake crushers and rolls were used for the coarse crushing, and the first jigging material was the undersize of 0.5 in. and the oversize of 8 mm. This product was treated on two single-sieve slide-arm jigs per section, manufactured by the Fraser & Chalmers Co. These jigs made a concentrate and a middling, the latter product being crushed through rolls to 8 mm. The total 8 mm. undersize was sent to V classifiers, the spigots of these classifiers going

three rows of Collom jigs and the overflow to V tanks feeding the round tables. The jigs made a concentrate, a middling, and a tailing. The middling was crushed in rolls to about 2.5 mm. and fed to a second classifier, the spigot of this classifier going to a fourth row of Collom jigs and the overflow to the round-table feed tanks. These jigs made only concentrate and tailing. The round tables were wooden-deck convex buddles about 18 ft. in diameter. These tables made a concentrate and a middling but no tailing. The middling was treated on 4-ft. Frue vanners, making a concentrate and a tailing.

*b. Coarser concentration.*—About 1894, blast furnaces were erected at the Great Falls plant. In order to obtain coarser concentrate for these furnaces, the one-sieve slide-arm jigs were replaced by two-compartment Harz jigs and the limiting size of crushing was increased from 0.5 in. to about 0.75 in. Shortly after this, in order to obtain more coarse concentrate, the limiting size was increased to 1.25 in. and two one-compartment Harz jigs were added to each section to treat the undersize of 1.25 in. and oversize of 0.75 in., the material through 0.75 in. and over 8 mm. going to the two-compartment Harz jigs.

*c. Screen sizing of fine jig feed.*—The next improvement was the substitution of screen sizing for the feed to the three rows of Collom jigs in place of the very imperfect classification. Two sets of trommels were added,—5 mm. round hole and 2.5 mm. round hole. The material between 5 and 8 mm. was sent to one set of jigs and that between 5 and 2.5 mm. to a second set. The undersize of the 2.5 mm. trommels was classified in an Evans launder classifier and sent to the third set of jigs. This classifier made four spigot products and an overflow. Each spigot was treated on a separate jig and the overflow was sent to the round-table feed tanks. About this time the Collom jigs were replaced by Harz jigs of the Evans type. The middling from these three sets of jigs was crushed through rolls as before, but a 2 mm. round-hole trommel was added to the roll system, thus forcing the rolls to crush through 2 mm.

*d. Installation of finer grinding.*—It was found that the tailing from the roll middling jigs was running too high in copper, and to overcome this a middling was made from these jigs, which was sent to 5-ft. Huntington mills for finer grinding. These mills were equipped with slotted screens, the openings being 1.5 by 10 mm. Soon after this a middling section was built, equipped with 5-ft. Huntington mills and Harz jigs to treat the middling from the 2.5 mm. and the roll middling jigs. The Harz jigs were fed from Evans launder classifiers, the overflow of the classifiers being returned to the table divisions of the main sections.

*e. Stamp tried in place of Blake crushers and rolls.*—About 1893 a steam stamp was installed in Section No. 3 to replace the Blake crushers and rolls in the coarse-crushing section. This stamp was equipped with

screens having openings of $\frac{3}{16}$ by $\frac{3}{8}$ in. Later the openings were increased to as large as $\frac{7}{8}$-in. round hole, but it was found that the stamp made altogether too much slime, and it was soon discarded.

*f. Mill increased from three to six sections.*—During the year 1900 three more sections were added to the mill, the equipment in the new sections being the same as that of the first three sections. A Huntington mill middling section was added to treat the fine middling from the three new sections.

*g. Tonnage increased from 300 to 450 per section.*—The tonnage was gradually increased from the start and by 1901 the mill was treating 2,700 tons a day or 450 tons per section.

*h. Round tables replaced by Wilfley tables.*—During 1901 the round tables in Section 6 were replaced by Wilfley and Overstrom tables. These tables gave very satisfactory results, producing a much cleaner concentrate than the round table and making a considerable amount of tailing, which reduced the load on the vanners following. A sufficiently clean tailing to be discarded could not be made from the round tables. The following figures taken from tests made in September, 1903, give the analyses of the concentrates from the two different tables:

| | Cu. Per Cent. | $SiO_2+Al_2O_3$ Per Cent. | Fe. Per Cent. |
|---|---|---|---|
| Concentrate from round tables | 11.55 | 48.1 | 14.1 |
| Concentrate from Wilfley tables | 16.57 | 25.2 | 21.5 |

The round tables in all but one of the remaining five sections were replaced by Wilfley tables in 1905.

There is, of course, no question to-day concerning the relative merits of a round table and a reciprocating table for the treatment of sands. There is, however, a splendid field for the round table in the concentration of slime.

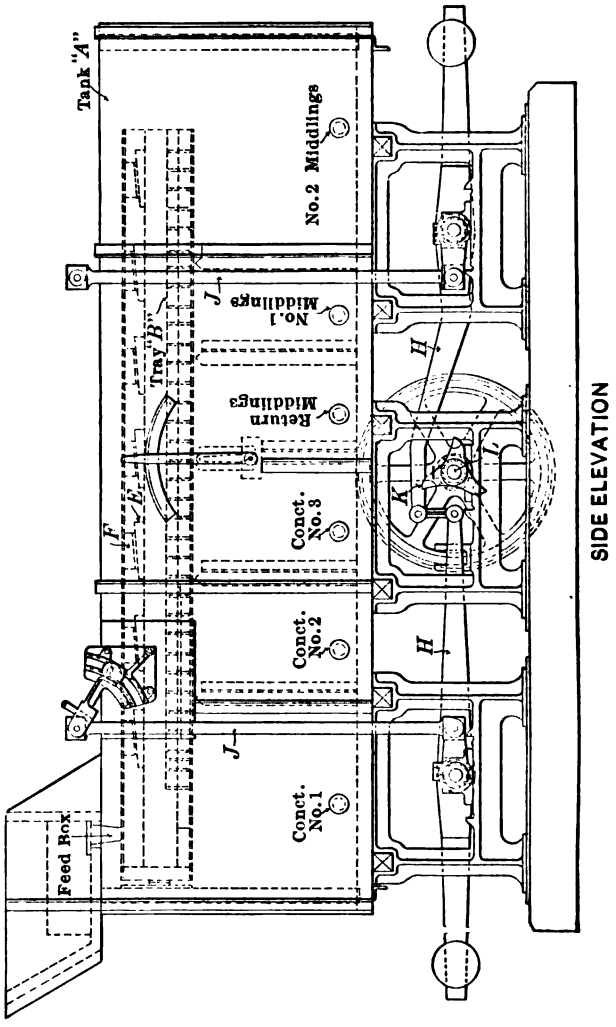
## II. The Hancock Jig Replaces the Evans Jig.

During 1905 a 25-ft. Hancock jig was installed in one section of the mill to be tested against the Evans jig. The Hancock jig has proved to be quite superior to those of the Harz type, to which class of jigs the Evans belongs, and has been incorporated in the flow sheet as installed in the Washoe concentrator at Anaconda. For this reason the construction and operation of this jig will be described briefly and an account of some of the experimental work that was done at Great Falls in perfecting this jig will be given.

*a. Description of Hancock jig.*—The Hancock jig belongs to the movable-sieve type of jigs. It is of Australian origin and was invented by H. R. Hancock. The larger size jig consists of a tank, *A*, Fig. 1, 25 ft. long by 4 ft. wide and 5 ft. deep, in which is suspended a frame, *B*, 20 ft. long by 2 ft. 8 in. wide. This frame carries a series of screens, the sizes of the apertures depending upon the material to be concentrated. These screens rest on hard-wood slats spaced 5 in. between centers. Above the screen are brass castings, *C*, forming pockets 5 by 10 in. by 3 in. deep. The transverse ribs of these castings fall immediately above the ribs supporting the screens. The brass castings are held firmly against the screen by side boards, *D*, which are in turn held securely in place by hard-wood wedges, *E*, driven in between them and steel lugs, *F*, on the side boards of the tray. The tray is carried on two cast-steel cross-bars, *G*, which are in turn connected to the levers, *H*, by the connecting arms, *J*, on either side of the jig hutch. These levers engage a three-way cam, *K*, on the main drive shaft, *L*, of the jig. This shaft revolves at 60 to 65 rev. per min. and imparts an up-and-down motion to the tray through the rocking arms and connecting rods. The upward movement of the tray is uniform, while the downward movement is accelerated and accompanied by a "bump." The up-and-down stroke ranges from ⅜ to ¾ in. in length. The connecting arms at the head end of the jig are connected to the jig hutch by adjustable quadrant arms which impart a forward-and-backward motion to the tray. Both the up-and-down and the forward-and-backward motions can be adjusted to suit the various requirements of the different pulps to be treated. Bedding material, consisting of coarser mineral particles, iron slugs, or punchings, etc., is retained in the pockets above the screen. The feed is introduced on the tray at the head end and travels the length of the tray. The concentrate is drawn down through the bedding and screen into the hutch from which it discharges, while the gangue, or lighter material, being unable to penetrate the bed, is carried along the tray and discharged over the end. The ore bed on the jig is usually about 4 to 6 in. deep above the bedding and the water level in the tank or hutch is maintained about 4 in. above the top of the ore bed.

The first jig installed at Great Falls was defective mechanically and did not stand up well. Since this time a great many improvements in the mechanical design of the jig have been made at Great Falls, so that to-day the jigs in use there give very little trouble from breakdowns and all the wearing parts are accessible and easily replaced with but little lost time.

*b. Tests made on Hancock jig at Great Falls.*—In this paper the term "natural feed" will be applied to a pulp which has neither been classified nor sized. For example, the undersize of a screen which is subjected to



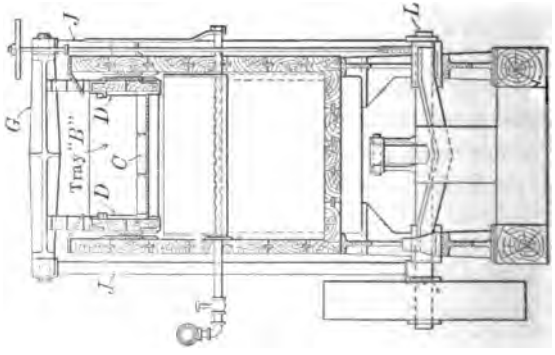




FIG. 1.—HANCOCK JIG.

no further sizing or classifying but contains particles ranging in size from the given screen size down through the finest slime, would constitute a "natural feed" for a machine. A pulp in which the range of size of the particles is limited by some sizing device will be termed a "sized feed." The product of a classifier will be termed a "classified feed." However, it should be noted that all products from a classifier or classifier system are not truly classified products, as the coarsest product from a classifying system or device will contain an increment of coarse mineral grains while the finest product will contain an increment of fine gangue material.

*1. Treating natural feed, undersize of 8 mm. round-hole trommel.*—The first tests, made in June, 1905, were upon the treatment of the undersize of the 8 mm. round-hole trommels. In the table on page 1809 the results obtained from two typical tests upon this class of feed are given, one in which the jig was treating a low tonnage and one in which the jig was treating a high tonnage. The material was fed directly to the jig, with no dewatering or desliming. Referring to the tabulated results of these two tests, it will be noticed that the low-tonnage test shows a very good recovery of copper, whereas the feed treated in the high-tonnage test was evidently too heavy for the jig to handle efficiently. The consumption of water is high on this class of feed, due to the necessity of adding a considerable quantity of water beneath the tray of the jig to keep the slime and fine sand out of the concentrate. The Evans equipment replaced by the one 25-ft. Hancock jig in the low-tonnage test consisted of eight double-compartment jigs treating 8 to 5 mm. material, eight double-compartment jigs treating 5 to 2.5 mm. material, two four-spigot classifiers treating the undersize of the 2.5 mm. trommels, eight double-compartment jigs treating the spigot products of the two classifiers, two 5 mm. trommels and two 2.5 mm. trommels, the preceding being the equipment for that division of a 500-ton section. In the high-tonnage test the above equipment in two 500-ton sections was replaced by the one Hancock jig.

*2. Treating sized material, 8 to 2.5 mm. round hole.*—A test was made in March, 1909, upon a sized feed, 8 to 2.5 mm., which has been chosen as being typical of the Hancock-jig work on this class of feed. During this test the undersize of 8 mm. and oversize of 2.5 mm. from two sections were treated on one Hancock jig. One 25-ft. jig replaced 16 two-compartment Evans jigs treating 8 to 5 mm. material, 16 two-compartment Evans jigs treating 5 to 2.5 mm. material, and four 5 mm. trommels. The jig produced a better grade of concentrate from this sized feed than from the natural feed, used considerably less water, and made lower-grade middlings, although the percentage of recovery is less on account of the lower grade of feed.

*3. Treating classified material, 11 to 0.25 mm. quartz.*—A test on this

class of material made during November, 1910, has been used to demonstrate the work of the jig. During this test the jig was fed from an Anaconda classifier designed at the Boston & Montana mill. This classifier received as feed the total undersize of an 11 mm. square-hole trommel from one 500-ton section. The overflow of this classifier was sent to the Wilfley-table feed tanks and the plug discharge was treated on the jig. The preceding tests were all made on a 25-ft. jig with a 20-ft. tray, whereas in this test a 19-ft. jig with a 15-ft. tray was used. Following are screen sizing tests made on the spigot product and overflow of the Anaconda classifier preceding the Hancock jig:

*Sizing Tests Made on the Spigot Product and Overflow of the Anaconda Classifier.*

*Hancock Jig Treating Spigot Product. Wilfley Tables Treating Overflow.*

| Screen Sizes | Spigot Product | | | Overflow | | |
|---|---|---|---|---|---|---|
| | Cumulative Per Cent. Solids | Assay Per Cent. Cu. | Cumulative Per Cent. Cu. | Cumulative Per Cent. Solids | Assay Per Cent. Cu. | Cumulative Per Cent. Cu. |
| On 11.3 mm..... | 0.5 | 3.05 | 0.5 | ..... | ..... | ..... |
| On 8.0 mm..... | 12.3 | 2.38 | 10.1 | ..... | ..... | ..... |
| On 5.7 mm..... | 28.8 | 2.18 | 22.4 | ..... | ..... | ..... |
| On 4.0 mm..... | 42.7 | 2.65 | 35.0 | ..... | ..... | ..... |
| On 2.83 mm..... | 52.4 | 2.66 | 43.8 | ..... | ..... | ..... |
| On 2.00 mm..... | 58.5 | 2.83 | 49.7 | ..... | ..... | ..... |
| On 1.41 mm..... | 68.9 | 2.88 | 60.2 | ..... | ..... | ..... |
| On 1.00 mm..... | 74.5 | 2.98 | 65.9 | ..... | ..... | ..... |
| On 0.71 mm..... | 82.7 | 3.07 | 74.5 | ..... | ..... | ..... |
| On 0.50 mm..... | 87.4 | 3.12 | 79.5 | ..... | ..... | ..... |
| On 0.25 mm..... | 93.3 | 3.53 | 86.6 | 2.4 | 0.90 | 0.6 |
| On 0.110 mm..... | 97.4 | 5.86 | 94.8 | 16.1 | 1.36 | 6.4 |
| On 0.075 mm..... | 98.4 | 6.74 | 97.1 | 31.6 | 3.47 | 23.2 |
| Through 0.075 mm... | 1.6 | 5.31 | 2.9 | 68.4 | 3.59 | 76.8 |
| Total..... | ..... | 2.93 | ..... | ..... | 3.20 | ..... |

The jig did remarkably good work upon this classified feed, producing a middling assaying 1.10 per cent. of copper from a feed assaying 2.95 per cent. of copper.

*c. Tests made on Evans jigs.*—During June, 1905, a test was made on the Evans jig system and the results of this test are included in the tabulation on page 1809, as being typical of the work of these jigs. The system comprised eight double-compartment jigs treating 8 to 5 mm. material, eight double-compartment jigs treating 5 to 2.5 mm. material, and eight double-compartment jigs treating the classified 2.5 mm. undersize, slime removed. This system produced very clean concentrates, but made a lower recovery of copper and used a great deal more water than

the equivalent Hancock jig system. The system produced a small amount of tailings amounting to 4.7 per cent. of the original feed and assaying 0.97 per cent. of copper.

The percentage recoveries of copper given in the tabulated results on page 1829, as well as in some of the test data to follow, are based upon the total copper in the feed to machine or system. Our more recent practice is to base the recovery, not upon the total copper in the feed, but upon that copper which is in such a form in the feed that it may be recovered in a suitable grade of concentrate by direct mechanical concentration. This latter method gives, of course, the true efficiency of the machine or system under test.

*Tests Made on Hancock Jig at Great Falls, Mont.*

| Data | Natural Feed Through 8-mm. round-hole trommel (Slime contained) | | Sized Feed Through 8-mm. on 2.5-mm. round-hole trommel | Classified Feed 11-mm. to 0.25-mm. quartz | Test Made on Evans Jig System at Great Falls, June, 1905 |
|---|---|---|---|---|---|
| | Low Tonnage | High Tonnage | | | |
| Feed per 24 hours: | | | | | |
| Average | 420 | 865 | 545 | 480 | 430 |
| Maximum | 450 | 980 | ..... | ..... | 480 |
| Minimum | 400 | 750 | ..... | ..... | 420 |
| Feed assay, per cent. Cu | 3.31 | 3.43 | 2.71 | 2.95 | 3.38 |
| Concentrate assay, per cent. Cu. | 9.00 | 9.15 | 13.90 | 9.58 | 10.5 |
| Concentrate assay, Insol. ($SiO_2+Al_2O_3$) | 30.7 | 25.8 | 18.7 | 23.8 | 12.3 |
| Middling assay, per cent. Cu | 1.56 | 2.04 | 1.46 | 1.10 | 1.70 |
| Per cent. copper recovered | 59.2 | 49.6 | 51.6 | 58.72 | 48.7 |
| Machines displaced by Hancock Jig: | | | | | |
| Evans jig compartments | 32 | 64 | 64 | 72 | ..... |
| Evans classifiers | 2 | 4 | None | 3 | ..... |
| Trommels, 3 by 6 ft | 4 | 8 | 4 | 4 | ..... |
| Screen area of Hancock jig, sq. in | 6,690 | 6,690 | 6,690 | 5,500 | ..... |
| Screen area of Evans jig retired, sq. in | 36,500 | 73,000 | 48,670 | 54,720 | ..... |
| Water used per ton per 24 hours, gals | 1,530 | ..... | 350 | 515 | 3,500 |

*General notes on Hancock jigs.*—The chief advantages of the Hancock jig over jigs of the fixed-sieve type are:

Higher recovery of copper;

Great saving in floor space;

Great saving in water consumption;

Less operating and repair labor;

Less power required.

We have not found it advantageous to produce a tailing from the Hancock jig, the usual practice being to make concentrates from the first

three hutches, a middling from the fourth hutch, which is returned to the jig, and a middling for grinding from the fifth and sixth hutches.

The size of the openings in the screens with which the tray is clothed is dependent on the size and quality of the feed to be handled and can only be determined by trial. "Ton-cap" screen has been used with very satisfactory results in the Hancock jigs. It outwears other forms of screen and is much easier to keep open.

## III. Richards Pulsator Classifier System.

*a. Experimental work.*—During the summer of 1907, a six-pocket Richards pulsator classifier of the inverted type was installed in Section No. 1.[1] The classifier was preceded by a pyramid-shaped desliming tank,

*Richards Pulsator Classifier System Treating Middling from Hancock Jig Crushed through 2.5 mm. Round Hole.*

| Product | Rate Per 24 Hours | | Assay | | Per Cent. of Total | |
|---|---|---|---|---|---|---|
| | Pulp | Solids | Cumulative | Insoluble | Solids | Copper |
| | Gallons | Pounds | Per Cent. | Per Cent. | | |
| Feed | 107,900 | 256,000 | 1.35 | ..... | 100.0 | 100.0 |
| Concentrate | 7,100 | 30,000 | 6.57 | 29.0 | 11.7 | 56.9 |
| Tailing | 54,500 | 71,400 | 0.34 | ..... | 27.9 | 6.9 |
| Middling | 91,600 | 75,400 | 0.60 | ..... | 29.5 | 13.0 |
| Vanner feed and slime | 219,800 | 79,200 | 1.02 | ..... | 30.9 | 23.2 |
| Total product | 373,000 | 256,000 | 1.35 | ..... | 100.0 | 100.0 |

| | |
|---|---|
| Tons treated per 24 hours by system | 128 |
| Tons treated per 24 hours by classifier | 117 |
| Gallons water used per ton treated by system | 2,070 |
| Gallons water used per ton treated by classifier | 1,410 |

which removed part of the slime from the pulp. The spigot discharge of this tank was fed directly to the classifier. This tank received the middling from the Hancock jig crushed through 2.5 mm. round hole, but not any of the undersize of 2.5 mm. from the mine fines or coarse-crushing section. The feed was therefore comparatively low grade, assaying 1.35 per cent. of copper. A test was made on this system during September 1907; as the flow sheet and complete details of this test are given in Richards's *Ore Dressing*, Vol. III., p. 1392, only a summary of the test will be given here.

The classifier made six spigot products, each of which was treated on

[1] For a description of this classifier see paper Development of Hindered-Settling Apparatus, by Dr. Robert H. Richards, *Trans.*, XLI, 396 (1910).

a Wilfley table. The spigot products are numbered from 1 to 6, No. 1 being the finest and No. 6 the coarsest product. The No. 1 and No. 2 spigots each contained slime. The tables treating the No. 1 and No. 2 spigots produced clean concentrate, middling and head-water slime to vanners and tailing to a 50-mesh single-belt Callow screen, the oversize being tailing to waste and the undersize going to vanners. The No. 3 and No. 4 spigot tables produced clean concentrate, middling to vanners and tailing to waste. The No. 5 and No. 6 spigot tables produced clean concentrate and middling for grinding. The results of the test are summarized in the table on p. 1810.

Shortly after the foregoing test was completed a second test was made on the pulsator-classifier system, using as feed the product of 5-ft. Huntington mills crushing through 1.25 by 10 mm. slot screens. The flow sheet of this test was the same as for the preceding test except that the pulsator-classifier spigots, 4, 5 and 6 (coarse) were combined after leaving the classifier and treated on one Wilfley table instead of being treated separately.

*Richards Pulsator Classifier System Treating Middling from Evans Jigs Crushed through 1.25 by 12 mm. in Huntington Mills.*

| Product | Rate Per 24 Hours | | Assay | | Per Cent. of Total | |
|---|---|---|---|---|---|---|
| | Pulp | Solids | Cumulative | Insoluble | Solids | Copper |
| | Gallons | Pounds | Per Cent. | Per Cent. | | |
| Feed | 207,300 | 374,200 | 1.32 | ..... | 100.0 | 100.0 |
| Concentrate | 5,900 | 26,070 | 6.98 | 26.5 | 7.0 | 36.9 |
| Tailing | 61,900 | 128,530 | 0.44 | ..... | 34.4 | 11.5 |
| Middling | 44,500 | 146,120 | 0.92 | ..... | 39.1 | 27.4 |
| Vanner feed and slime | 307,700 | 73,480 | 1.64 | ..... | 19.5 | 24.2 |
| Total product | 420,000 | 374,200 | 1.32 | ..... | 100.0 | 100.0 |

| | |
|---|---|
| Tons treated per 24 hours by system | 187 |
| Tons treated per 24 hours by classifier | 171 |
| Gallons water used per ton treated by system | 1,140 |
| Gallons water used per ton treated by classifier | 760 |

Further experiments were made on the treatment of the 2.5 mm. undersize resulting from the crushing of the Hancock-jig middling by the pulsator-classifier system in which the last three spigots of the classifier (coarse) were combined and treated on one Wilfley table. This amounted to making four instead of six classifications, a procedure to which Dr. Richards was opposed from the first.

The classifier system as installed in Section No. 1 had shown so much better results than the Evans jig and unclassified table-feed system which

was in use at that time in the mill that it was decided to remodel one section (No. 4) of the mill and install the pulsator-classifier system.

*b. Richards pulsator classifier system installed in Section No. 4.*—The flow sheet of this remodeled section will be outlined briefly. The crude ore after being crushed through 1.5-in. round hole by Blake crushers was passed over a series of three screens: trommels 3 by 6 ft.—⅞-in. round hole, and 8 mm. (⅓-in.) round hole and 2.5 mm. ($\frac{1}{10}$-in.) round hole, respectively. The oversize of the ⅞-in. and 8 mm. trommels was combined and treated on a 42 by 30-in. Woodbury jig. This jig produced coarse concentrate, a hutch product which was sent to the Hancock jig, and a middling which was divided into coarse and fine by screening and crushed through two sets of rolls (15 by 28 in.), coarse and fine respectively, the products of the rolls being returned to the head of the system. The oversize of the 2.5 mm. trommels was treated on one Hancock jig (20-ft. tray). This jig made a coarse concentrate from the first three hutches, a return middling to the jig from the fourth hutch, and a middling to be crushed from the fifth and sixth hutches. This Hancock-jig middling was crushed in two sets of rolls, through 2.5 mm. round hole, and fed to two six-compartment Richards pulsators in which only four compartments were used. The four classified products were fed to four sets of Wilfley tables. The four tables treating the first two spigots separately—fine material—produced a concentrate, a middling to vanners which in turn produced concentrate and tailing, a tailing which was treated on a 50-mesh Callow screen, the oversize being tailing and the undersize going to vanners which produced a concentrate and a tailing, and a head-water slime which went to the slime division for treatment. The two tables treating the third spigot made a concentrate, a tailing to waste, head water to the slime division, and a middling which was ground in the Huntington mills. The two tables treating the fourth or coarse spigot made a concentrate, a middling and tailing to be ground in the Huntington mill, and a head water (carrying practically no solid matter) to waste. The undersize of the 2.5 mm. trommels containing the original mine fines and the fines from the coarse-crushing section was treated in two six-compartment pulsator classifiers, but four compartments being used. The flow sheet of this classifier system is identical with the one just described except that the fourth spigot, instead of going directly to tables, was treated on a Richards pulsator jig. This jig made a concentrate and a middling, the middling being sent directly to two Wilfley tables, which made a concentrate, a middling and tailing to be ground in the Huntington mills, and a head water to waste. The middling was ground in 5-ft. Huntington mills through 1.25 by 12 mm. slots and treated in one six-compartment pulsator classifier, only four compartments being used. The two tables treating the first two spigots (fine) separately made

a concentrate, a middling to vanners which produced a concentrate and a tailing, a tailing to 50-mesh Callow screens, the oversize being tailing and the undersize going to vanners which made concentrate and tailing a head-water slime to the slime division. The table treating the third spigot made a tailing and head water to waste and a middling which was returned to the Huntington mills. The table treating the fourth spigot made a concentrate, a middling which was returned to the Huntington mills, and a tailing and head water to waste. The total slime, consisting of the overflow of the desliming tanks preceding the pulsator classifiers and the head water slime from the Wilfley tables as noted above, was treated in the slime division. This division consisted of 30 8-ft. Callow cones for thickening the slime, the spigot of the cones going to 16 round tables, 18 ft. in diameter, and one Deister No. 3 slime table, and the overflow going to waste. These tables made concentrate and tailing. This section had a capacity of 500 tons per day.

The new section did not do as good work as had been anticipated, the tailing being little, if any, better than that produced in the old sections and the concentrate more siliceous. The failure of the section is attributed largely to the presence of the original mine fines in the pulsator-classifier system, this material being rich in fine free mineral. It should be noted that this product was not included in the pulps that were treated experimentally by the pulsator-classifier system in Section No. 1. Another condition which had a decided effect upon the results was the constant variation in the quantity and quality of the mine fines, a condition with which the pulsator classifiers could not successfully cope on account of their extreme sensitiveness to changes in the feed. It must be borne in mind that the Great Falls mill was built before the day of independent crushing plants, and although the value of separate crushing has long been recognized at this mill, one thing and another has interfered with the carrying out of the plans for an independent crushing unit. The mine-run ore is fed direct to each section which does its own crushing, and consequently the quantity and quality of the undersize from the original 2.5 mm. trommels varies immediately and considerably with the variations in the character of the ore.

Several alterations were made in the section, such as the substitution of six classifications in place of four in the pulsator classifiers, the elimination of the pulsator jig, the mixing of the original 2.5 mm. undersize with the 2.5 mm. middling product and Huntington mill product, and the substitution of Harz jigs for Wilfley tables in the treatment of the coarser spigots of the pulsator classifiers. None of these changes, however increased the efficiency of the section to any extent.

## IV. Woodbury Classifier and Jig System.

Experiments made in 1905 on Woodbury classifiers and jigs had shown such favorable results that it was decided to install this system in Section 4, to be run in parallel with the Hancock jig—Richards pulsator-classifier system. A Woodbury classifier and two jig compartments connected in series, known as an 8 mm. unit, were set up in parallel with the Hancock jig. This unit received the total 8 mm. undersize as feed. The classifier removed the slime and fine sands (undersize of about 0.4 mm.), which were sent to V tanks, the spigot product being treated on Wilfley tables and the overflow of the tank sent to the slime division. In addition, this classifier made a cup and a hutch concentrate and a classified middling, which was fed to the first jig. This jig made a cup and a hutch concentrate and a middling, which passed over the second jig. The second jig produced a cup concentrate, a hutch product which was treated on a Wilfley table, and a middling which was crushed through a 2.5 mm. round hole by two sets of rolls. The total 2.5 mm. was split into halves, one part going to a Richards pulsator system similar to that described and the other part being treated by the Woodbury 2.5 mm. system. The Woodbury 2.5 mm. unit consisted of a Woodbury classifier followed by four Woodbury jig compartments (48 by 30 in.) arranged in series. The classifier removed the slime and fine sand, which were sent to a V tank, the spigots of the tank being treated on Wilfley tables and the overflow going to the slime division. The classifier also produced a hutch concentrate and a classified middling, which went to the first jig. This jig produced a hutch product, which was treated on a Wilfley table, and a middling which passed to jig No. 2. Jig No. 2 made a hutch product for table work, a middling to be ground in Huntington mills and a middling which went to jig No. 3. This jig made the same products as No. 2, passing a middling to the No. 4 or last jig. The No. 4 jig made a hutch product for table work and a tailing. The Wilfley tables treating the classifier overflows made a concentrate, a middling for vanner treatment, the vanners making a concentrate and tailing, a tailing and a head-water slime to the slime division. The tailing was screened over a 50-mesh Callow screen, the oversize going to waste and the undersize to vanners which made concentrate and tailing. The Wilfley tables treating the hutch products of the jigs made a concentrate, a middling to Huntington mills, and a tailing to waste. After considerable experimental work and change in the flow sheet this system was also abandoned as not being applicable for the most efficient treatment of the Butte ores.

## V. The Great Falls System of Concentration.

Two classifying systems, the Richards pulsator and the Woodbury, had now been tried out and had not proved satisfactory. During the summer

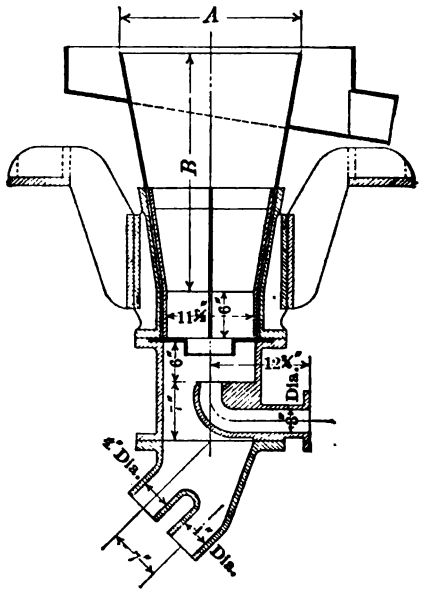
of 1911 the Richards-Janney classifying system had been tried in the Washoe concentrator at Anaconda, which mill receives about the same ore as the Great Falls mill, and had not given satisfactory results. The experimental work that had been carried on had proved quite conclusively that the Hancock jig is more efficient for the concentration of the Butte ores than the Evans jig, that the Harz type of jig will not produce a satisfactory tailing on account of the fine free mineral which the jigging action throws into the tailing, that a reciprocating table is the best form of concentrator for the production of tailing, and that the feed to any concentrating machine should be so prepared, either by classifying or sizing, that it will contain only those grains of mineral and gangue material which can be treated efficiently by the machine in question. Classification was preferred to screening in preparing for treatment material finer than 4 mm. on account of its simplicity and lower cost. The problem then became one of either finding or developing a suitable form of classifier. As some of the best classifiers on the market had been tested and found wanting, it seemed that a classifier would have to be developed for the particular work to be done. This classifier had to be a flexible one which could handle efficiently a feed which varied considerably in quantity and quality.

During the summer of 1910 experiments had been carried on to determine the adaptability of the Richards pulsator classifier of the direct type. Some difficulty was experienced in keeping the slime contained in the feed out of the spigot products, and to overcome this difficulty Dr. Richards advised that we use a conical tank to deslime partly the feed to the classifier. He suggested the use of a short sorting column at the base of the cone in which a pulsating rising current of water was to be introduced. It was found later that this pulsation was of no advantage and a plain rising current was used. This deslimer gave very satisfactory results, removing all of the slime from the feed to the Richards classifier.

At this time the Hancock jigs were receiving as feed the total undersize of about ⅜ in., the slime being partly removed by a plain V settling tank before the material was fed to the jigs. The presence of the slime and fine sand in the feed was detrimental to the work of the jigs and it was decided to try one of these desliming cones ahead of the jig, not only to remove the slime from the jig feed but also the fine sand. This installation gave very satisfactory results, the work of the jig being improved considerably; a large amount of table feed was sent directly to finishing tables, where it belonged.

Having obtained such satisfactory results from these desliming cones, which were virtually classifiers, it was decided to develop this machine with the hope of producing an efficient hindered-settling classifier. The experimental work, which extended over a period of a year or more, re-

sulted in the development of an extremely simple classifier, which gave practically perfect hindered-settling products.[1]




No. 1 and No. 3, *A*—7 ft. 0 in.; *B*—4 ft. 3 in.
No. 2, *A*—2 ft. 4 in.; *B*—3 ft. 6 in.
No. 4, *A*—2 ft. 1 in.; *B*—3 ft. 6 in.
No. 5, *A*—1 ft. 11 in.; *B*—2 ft. 6 in.
No. 6, *A*—5 ft. 0 in.; *B*—3 ft. 6 in.

FIG. 2.—ANACONDA CLASSIFIER.

Having developed what seemed to be an efficient hindered-settling classifier, we were now ready to design a flow sheet embodying the principles which our years of experimental work had brought out. In designing this flow sheet, the following factors were kept constantly in mind:

1. Maximum recovery of valuable minerals;
2. Simplicity of flow sheet and machines;
3. Minimum consumption of water;
4. Minimum cost of operation;
5. Maximum tonnage per unit of floor space.

*a. Description of Great Falls flow sheet as installed at Anaconda.*—Reference should be made to the pictorial flow sheet of the Great Falls system, Fig. 3, and to the pictorial flow sheet of the Anaconda concentrator, Sections 2–8, Fig. 4. The Anaconda flow sheet, sometimes referred to as the "Evans system," was modeled after the flow sheet used in the Great

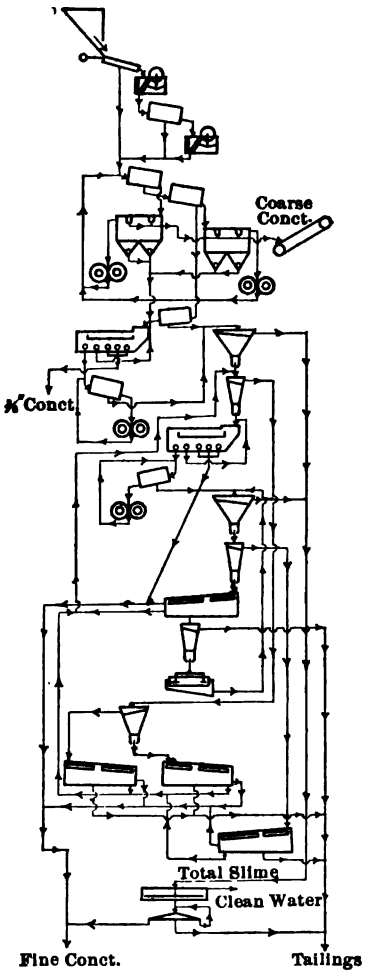
[1] See paper, The Anaconda Classifier, presented by Robert Ammon at August, 1913, meeting.

Falls mill and is practically the same, the most important change being the use of Wilfley tables in place of Frue vanners for the treatment of the Wilfley table middlings. The Anaconda flow sheet is the one which was replaced in Section No. 1 of the Anaconda mill by the Great Falls system.

*1. Coarse crushing and jigging division.*—In installing the Great Falls system of concentration in Section 1 of the Anaconda mill, the coarse crushing and jigging section remained practically as it was. Although the installation of an independent coarse-crushing plant has been under consideration for a number of years the opportunity for its installation has not presented itself as yet. The mine-run ore is therefore fed directly to the concentrating sections. There is one 1,000-ton ore-supply bin for each section of the mill. The ore is fed from the bin through two air-operated gates to two shaking screens with 2-in. round holes. Two men are employed as feeders, one on each shaking screen, and the usual practice is for one man to feed while the other cleans his screen. In this way a fairly uniform feed is maintained. The oversize of the shaking screens goes to one 12 by 24-in. Blake crusher set to crush through 2.5 in. The product of the crusher goes to two trommels, 3 by 6 ft., clothed with 2-in. round-hole, cast manganese-steel screens. The oversize of these screens goes to two 5 by 15-in. Blake crushers set to crush through 2-in. The undersize of the shaking screens, the undersize of the 2-in. trommels, and the product of the two smaller Blake crushers are elevated to the head of the mill, where they pass through four 1-in. round-hole trommels, 3 by 6 ft., clothed with punched sheet steel. The oversize of these screens is treated on two double-compartment coarse Harz jigs, which make a cup concentrate, a hutch product, and a middling. The middling is partly dewatered by inclined screens and passes to one set of coarse rolls, 24 by 54 in., crushing through 1 in., and from there returns to the head of the system. The water from the middling, together with the hutch product, joins the feed to the coarse Hancock jigs. The concentrate is dewatered by a shaking screen and passes up an inclined conveyor to the coarse-concentrate bins. The water from the concentrate is used to sluice the undersize of the 2-in. shaking screens. The undersize of the 1-in. trommels goes to four 10 mm. ($\frac{2}{5}$-in.) round-hole trommels, 3 by 6 ft., clothed with punched-steel screens. The oversize of these screens passes to four double-compartment Harz jigs, which make a cup concentrate, a hutch product, and a middling. The concentrate joins the concentrate from the coarse Harz jigs. The middling is partly dewatered and passes to one set of fine rolls, 24 by 54 in., crushing through 10 mm. ($\frac{2}{5}$ in.), and from there returns to the head of the system. The hutch product, together with the water from the middling, unites with the feed to the coarse Hancock jigs.

The Anaconda flow sheet in the remaining sections of the mill differs from the above in the use of but two 1-in. trommels and two 10 mm. trommels.

*2. Hancock jig division.*—The total undersize of the 10 mm. trommels




1 Ore Bin.

2 Shaking Screens, 2-in. Round Hole.

1 Blake Crusher, 12 by 24 in.

2 Trommels, 2-in. Round Hole.

2 Blake Crushers, 5 by 15 in.

4 Trommels, 1-in. Round Hole.

4 Trommels, 10 mm. Round Hole.

2 2-Comp. Harz Jigs (Coarse). 2 2-Comp. Harz Jigs (Fine).

1 Set of Rolls, 24 by 54 in. (Coarse). 1 Set of Rolls, 24 by 54 in. (Fine).

8 Trommels, 4 mm. Round Hole.

2 Hancock Jigs (Coarse). 6 No. 1 Anaconda Classifiers.

6 Trommels, 4 mm. Round Hole. 6 No. 2 Anaconda Classifiers.

1 Set of Rolls, 16 by 42 in. 3 Hancock Jigs (Fine)

8 Trommels, 1.25 by 12-mm. Slots.

2 Sets of Rolls, 16 by 42 in. 8 No. 3 Anaconda Classifiers.

6 No. 4 Anaconda Classifiers.

9 Wilfley Tables (Coarse Middlings).

2 No. 5 Anaconda Classifiers.

4 Huntington Mills, 6 ft.

3 No. 6 Anaconda Classifiers.

19 Wilfley Tables (Fine Primary). 12 Wilfley Tables (Coarse Primary).

23 Wilfley Tables (Secondary).

14 Dorr Thickeners, 28 by 3 ft.
35 Round Table Decks (Concrete).

FIG. 3.—GREAT FALLS SYSTEM OF CONCENTRATION. REMODELED SECTION NO. 1, ANACONDA CONCENTRATOR.

goes to eight 4 mm. ($\frac{1}{16}$-in.) round-hole trommels, 3 by 6 ft., clothed with punched-steel screens. The oversize of these screens is fed to two coarse 19-ft. Hancock jigs (trays 15 ft. long), which make a concentrate from the first three hutches, a middling from the fourth hutch, which is returned to the jig, and a middling from the fifth and sixth hutches, which is ground

through 4 mm. by one set of intermediate rolls, 16 by 42 in., the product of the rolls going to six 4 mm. round-hole trommels, 3 by 6 ft., the oversize of the trommels returning to the rolls. The concentrate passes to a set of dewatering tanks in the tank house and is known as ⅜-in. concen-

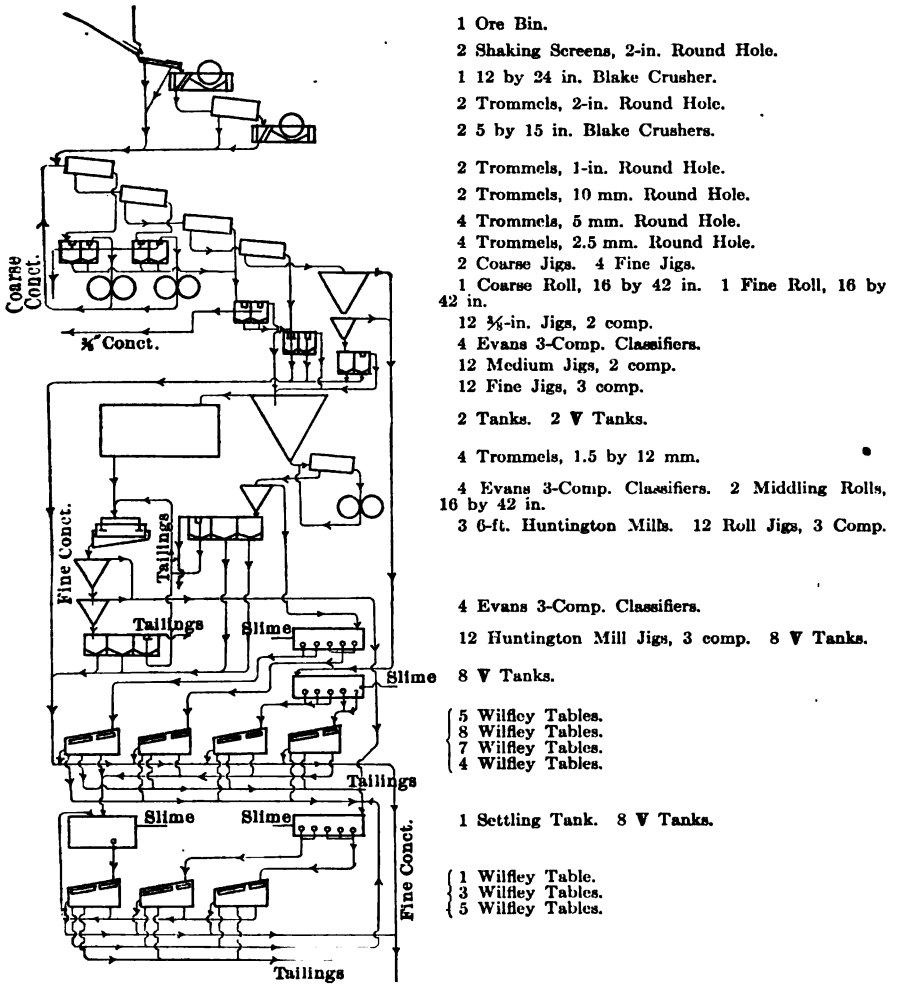



FIG. 4.—EVANS SYSTEM OF CONCENTRATION. SECTIONS NOS. 2 TO 8, ANACONDA CONCENTRATOR.

trate. These coarse Hancock jigs receive also the water from the middlings and the hutch products from the Harz jigs in the coarse crushing and jigging division, as noted above. This water replaces part of the water that would otherwise have to be added in the hutch of the jigs and as a result only about 100,000 gal. of water are added in the hutch of each jig.

The overflow of water from the jigs, together with the overflow water from the jig middling dewatering tanks, furnishes about three-fourths of the water required on the fine Hancock jigs. The total undersize from the 14 4-mm. trommels passes to six Anaconda classifiers No. 1. (See Fig. 2, p. 1816.) These No. 1 classifiers overflow slime (0.08 mm. to 0 quartz) which is sent directly to the slime division. The spigot discharge of these classifiers goes to the six No. 2 classifiers, which overflow 0.75 to 0.08 mm. material and discharge the 4.0 to 0.75 mm. material through the spigot. All sizes given for the Anaconda classifier products refer to the diameter of the quartz grain, the mineral grain being about one-fourth the diameter of the quartz grain. For example, a classifier product of 0.75 to 0.08 mm. material would contain quartz grains ranging in size from 0.75 to 0.08 mm. and free mineral grains ranging in size from 0.20 to 0.02 mm. This spigot discharge is treated on three fine Hancock 19-ft. jigs with 15-ft. trays. These jigs make fine concentrate from the first three hutches, middling from the fourth hutch, which is returned to the jig, and middlings from the fifth and sixth hutches, which unite and go to eight trommels, 3 by 8 ft., clothed with punched-steel screen having openings 1.25 by 12 mm. This middling product is sent to the screens before going to the rolls to remove any fine free mineral which may be present. The water overflowing from the fine jigs, together with the overflow from the middling dewatering tanks, is used in the fine middling division as wash water on the tables. The final products from this division are then: ⅜-in. concentrate from the coarse Hancock jigs, fine concentrate from fine Hancock jigs, table feed from No. 2 classifiers going directly to table division, slime from No. 1 classifiers going directly to slime division, and fine middling from fine Hancock jigs.

The equipment in the Anaconda flow sheet that was replaced by this division consists of four 5 mm. round-hole trommels receiving undersize from 10 mm. trommels in coarse-crushing division, four 2.5 mm. round-hole trommels treating the undersize of the preceding trommels, four 3-spigot Evans classifiers treating the undersize of the 2.5 mm. trommels, 12 double-compartment Evans jigs treating oversize of 5 mm. trommels, 12 three-compartment Evans jigs treating oversize of 2.5 mm. trommels, and 12 three-compartment Evans jigs treating the Evans classifier products. The overflow of the Evans classifier, consisting of slime and sand, goes to the table section. The jigs treating the oversize of 5 mm. receive also the hutch product of the Harz jigs in the coarse-jigging division. These 5 mm. oversize jigs make a cup concentrate which goes to separate tanks in the concentrate tank house and is called ⅜-in. concentrate. The hutch product from these jigs goes to the next set of jigs. The 2.5 mm. oversize jigs make a cup and a hutch fine concentrate and the 2.5 mm. undersize jigs make a hutch fine concentrate.

All three sets of jigs make a middling which passes to the roll middling division. This middling after being dewatered passes over four 3 by 8-ft. trommels clothed with slotted steel punched screens, 1.5 by 12 mm. The oversize goes to two sets of 16 by 42-in. finishing rolls, the product of the rolls being returned to the trommels. The undersize of the trommels goes to four Evans three-compartment classifiers which feed 12 Evans three-compartment jigs. The overflow from the classifier, containing slime and sand, goes to the table section. These jigs make a hutch fine concentrate, a middling to the Huntington mill division, and a tailing to waste.

*3. Fine middling division.*—The fine middling from the fine Hancock jigs goes to eight trommels as noted previously, the oversize of these trommels (1.25 by 12 mm.) going to two sets of 16 by 42-in. rolls, and the product from the rolls being returned to the trommels. The undersize of these trommels goes directly to eight No. 3 Anaconda classifiers, which remove the slime and send it directly to the slime division. The deslimed spigot discharge of the No. 3 classifiers passes to six No. 4 classifiers of the same type, where it is classified into finishing table feed, 0.75 to 0.08 mm., and coarse middling table feed, 1.25 to 0.75 mm. The coarse middling tables, nine in number, make a fine concentrate, a rough concentrate, and a middling tailing. The rough concentrate, containing the coarser mineral grains, is returned to the No. 2 classifiers preceding the fine Hancock jigs, where the coarser free mineral is discharged through the spigot and recovered at once by the jigs. The middling-tailing, being a mixture of the coarser middling and tailing grains, goes to two No. 5 Anaconda classifiers, which overflow about 30 per cent. of the material as tailing and discharge the remainder through the spigot, this product going to four 6-ft. Huntington mills of the Anaconda type, clothed with punched-steel screens 0.75 by 12 mm. The excess water contained in the tailing classifier overflow is recovered and used in the No. 4 classifiers. The Huntington mill product is delivered to the No. 3 classifiers, where the slime is at once removed. The coarse middling tables are standard 16-ft. Wilfleys of the No. 6 type, slightly modified mechanically. The riffles are spaced 1.5 in. between centers and are 0.75 in. high at the head end, tapering to $\frac{1}{8}$ in. at the concentrate end, and all but the upper eight are carried to the end of the table. The final products from this division are: Fine concentrate from the coarse middling tables, rough concentrate returned to the No. 2 classifiers, slime from the No. 3 classifiers sent directly to the slime division, and table feed overflowed from the No. 4 classifiers going to the table division.

The portion of the Anaconda flow sheet displaced by this division in Section 1 consists of three 6-ft. Huntington mills grinding the middling from the Evans roll middling jigs through 1 by 12 mm., 4 three-spigot

Evans classifiers treating the Huntington mill product, and 12 three-compartment Evans jigs treating the classifier spigots. The overflow from the Evans classifiers goes to the table section. The jigs make a fine concentrate from the hutches, a tailing to waste, and a middling which is returned to the Huntington mills.

*4. Table division.*—Standard 16-ft. Wilfley tables are used in this division both in the remodeled section and in the original Anaconda sections. This division receives the deslimed overflow from classifiers No. 2 and No. 4. The overflow from classifiers No. 2 containing the original mine fine material is further divided into two classes by three No. 6 classifiers, which overflow 0.35 to 0.08 mm. material and discharge through the spigot 0.75 to 0.35 mm. material. Each class goes to a set of Wilfley tables which make a concentrate, a tailing, and a middling, the latter product being returned to the No. 2 classifiers. The overflow from the No. 4 classifiers, ranging in size from 0.75 to 0.08 mm., is treated directly on a set of tables which make a concentrate, a tailing and a middling which is returned to the No. 2 classifiers. The feed to the tables is dewatered, the water being used as wash water in the table division and the slime division.

The Anaconda table division is divided into two parts. The upper table floor receives the overflow from the 2.5 mm. Evans classifiers and from the roll middling Evans classifiers. This overflow is partly deslimed in V tanks before being fed to the tables. These tables make a fine concentrate, a tailing, a middling which goes to the tables on the lower floor, and a head-water slime which goes to a square tank on the lower table floor, the spigot of the tank being treated on one table and the overflow uniting with the overflow from the upper table feed tanks and leaving the mill as slime. The lower table floor receives as feed, in addition to the upper table floor middlings, the overflow from the Huntington mill Evans classifiers. This total feed is partly deslimed in V tanks and is then treated on Wilfley tables making a concentrate, a tailing, a middling, which is returned to the table-feed tanks, and a head-water slime which goes to the square tank mentioned above. The overflow from these lower table-floor V tanks unites with the slime leaving the mill.

*5. Slime division.*—The slime division is an experimental plant and has a capacity of about 50 tons of slime per day, which is about 20 per cent. of the total slime produced in the remodeled section. This plant has now been in operation for about one year; after considerable experimental work, in which most of the leading slime concentrators on the market were tested, it was finally decided to adopt the concrete-and-steel round table. These tables are 18 ft. in diameter and have a slope of deck surface of 1.25 in. per foot.[1] The slime pulp, having a density of 2 per cent.

[1] See paper, Concentration of Slime at Anaconda, presented by Ralph Hayden at August, 1913, meeting, this *Bulletin*.

of solids, was originally thickened in 8-ft. Callow cones, operated to settle 95 per cent. of the solids in a product having a density of about 9 per cent. These cones did very good work, but have now been discarded in favor of 28-ft. diameter Dorr thickening tanks 3 ft. deep. These latter tanks, being larger units and flat bottomed, have the advantage over the Callow cones of a considerably lower cost of installation. These 3-ft. deep thickeners will probably be installed in stands of four tanks, one above the other.

A slime plant, consisting of Dorr thickeners and multiple-deck concrete-and-steel round tables, to treat the entire slime pulp from the eight sections of the mill, will undoubtedly be built this year. At the present time this slime pulp is settled in large ponds, the recovered solids being briquetted with fine first-class ore screenings and fine concentrates and treated in the blast furnaces.

*b. Special features of the Great Falls flow sheet.*—A great deal of experimental work was carried out during the development of this flow sheet and some of the more interesting experiments will be described.

*1. Development of Hancock jig flow sheet.*—One of the principles which was adhered to in designing the flow sheet was to treat on a given machine only that class and size of material which could be most efficiently treated by that machine. Thus it became necessary early in the work to determine the minimum size of free-mineral grain which can be recovered by the Hancock jig. This was investigated by screen sizing samples of the Hancock jig concentrate and middling taken during regular operation, and then sorting each screen-sized product into a true middling and true concentrate of given grades. This work showed conclusively that all mineral grains finer than 90 mesh (0.17 mm.) should be eliminated from the feed to the jig. Above this size the recovery of the copper which was present in the feed in the form of free mineral ranged between 95 and 100 per cent., but just at and below this limiting size the recovery dropped off abruptly, being only about 50 per cent. for mineral grains ranging in size between 90 and 120 mesh (0.17 to 0.10 mm.). As the Anaconda classifiers produce practically perfect hindered-settling products, the smallest quartz grain contained in the jig feed would be 0.17 by 4 = 0.68 mm., if we limit the smallest mineral grain to 0.17 mm. All of the quartz grains finer than 0.68 mm. would then be thrown off by the classifier and pass into the overflow together with the mineral grains finer than 0.18 mm. It so happened that it had previously been determined that the largest tailing grain should be about 0.75 mm., which size practically coincided with the maximum size grain in the overflow of the classifiers preceding the Hancock jigs. Thus it followed that this classifier overflow should be put on machines producing a tailing. It was found advisable to divide the Hancock jig feed, ranging in size as it did from 0.10 to 0.75 mm., into two parts by screening it over 4 mm. round-hole

trommels. The jigs did better work on the more closely sized feed and it also enabled us to introduce a more graded crushing by crushing the middling from the coarse jigs through 4 mm. in one set of rolls and the middling from the fine jig through 1.25 mm. in two other sets of rolls. An investigation was also carried on to determine whether the oversize of 4 mm. resulting from the first passage through the rolls should be returned to the coarse jigs or further crushed at once through 4 mm. Sorting tests made on this product showed the presence of such a small amount of free mineral that the jigging would not pay, therefore it was decided to use a closed circuit of rolls and 4 mm. trommels, and to crush all of the middling through 4 mm. before subjecting the product to concentration.

*2. Development of finishing table flow sheet.*—It having been determined that a reciprocating table is the best form of concentrator on which to produce a tailing and that the maximum tailing grain was to be about 0.75 mm., the next point to be investigated was the minimum size mineral grain which can be recovered by the reciprocating table, this table being in this particular case a standard 16-ft. Wilfley. This investigation was carried on for the most part in the testing laboratory on a small size Wilfley table. The limiting grain was found to be approximately 0.025 mm. in diameter, which would carry with it in the hindered-settling classifier product a grain of quartz approximately 0.08 mm. in diameter (0.08 mm. is equivalent to a 200-mesh screen opening). This meant that all slime must be eliminated from the table feed,—the term "slime" being rather loosely used to apply to all material, both granular and colloidal, finer than 0.08 mm. quartz and 0.025 mm. mineral. As a matter of fact this slime contains about 65 per cent. of granular material and 35 per cent. of colloidal material.

It will be noted that while the overflow of classifiers No. 2 is further classified into two products, 0.75 to 0.35 mm. and 0.35 to 0.08 mm., before being subjected to table treatment, the overflow of the No. 4 classifiers, consisting entirely of crushed low-grade middlings and ranging in size from 0.75 to 0.08 mm., is treated directly on the tables without further classification. This procedure is also the result of careful investigation, the experimental work having proved conclusively that while the further classification of the lower-grade material was of no advantage, the classification of the richer material increased the recovery on the tables by about 7 per cent.

*3. Treatment of coarse middling table products.*—As has been noted, it was originally intended to make no tailings coarser than 0.75 mm. However, when tests were made on the 500-ton unit, which was installed at Great Falls, it was found that the middling from the coarse middling table contained a considerable amount of tailing material, although

ranging in size from 0.75 to 1.25 mm. During the first test this entire product had been returned to the Huntington mills for crushing, but this so congested the middling section that the tonnage to the unit had to be materially reduced. During the second test a tailing was cut out from the product as it left the tables. This resulted in the elimination at this point in the system of 22.2 per cent. of the solids in the feed to the entire 500-ton unit assaying 0.43 per cent. of copper. It was then suggested that possibly more tailing material could be produced by the classification of this middling product, the overflow of the classifier being tailing. A laboratory test was made in which some of this middling product ranging in size from 2.0 to 0.8 mm. was treated in a hindered-settling classifier. Six products were drawn off through the spigot, the one settling against the strongest rising current being called No. 1 and the one settling out against the weakest rising current being designated as No. 6.

*Classification of Middling from Coarse Middling Table.*

| Spigot No. | Total Solids | | Assay Cu | | | Total Copper | |
|---|---|---|---|---|---|---|---|
| | Individual | Cumulative | Individual | Cumulative | | Individual | Cumulative |
| | | | | Down | Up | | |
| | Per Cent. | Per Cent. | Per Cent. | Per Cent. | Per Cent. | Per Cent. | Per Cent. |
| 1 | 8.8 | 8.8 | 0.98 | 0.98 | 0.40 | 21.0 | 21.0 |
| 2 | 15.9 | 24.7 | 0.57 | 0.70 | 0.35 | 22.4 | 43.4 |
| 3 | 18.8 | 43.5 | 0.36 | 0.56 | 0.30 | 17.1 | 60.5 |
| 4 | 37.4 | 80.9 | 0.30 | 0.44 | 0.28 | 27.6 | 88.1 |
| 5 | 12.2 | 93.1 | 0.25 | 0.41 | 0.25 | 7.9 | 96.0 |
| 6 | 6.9 | 100.0 | 0.22 | 0.40 | 0.22 | 4.0 | 100.0 |
| Total | 100.0 | ..... | 0.40 | ..... | ..... | 100.0 | ..... |

The first spigot contained the coarser middling grains and a few coarse quartz grains; the second spigot contained finer middling and tailing grains, while the last spigot contained for the most part only tailing grains. There were no free-mineral grains in the original middling fed to the classifier. The preceding tabulation shows that it is possible to discard in the tailing classifier 56.5 per cent. of the solids (Spigots 4+5+6), assaying 0.28 per cent. of Cu, as tailing, and to send to the Huntington mills only 43.5 per cent. of the solids, assaying 0.56 per cent. of Cu. In practice we have never been able to make a tailing as low as 0.28 per cent. of Cu at this point. In the third test at Great Falls on a 500-ton section this classifier tailing contained 25 per cent. of the total solids fed to the section and assayed 0.42 per cent. of Cu.

The rough concentrate from the coarse middling tables contains the coarser grains of free mineral and the finer middling grains. Experiments

were made in which this product was treated on a second set of Wilfley tables. The results, however, of this treatment were not at all satisfactory, the secondary tables making a low-grade concentrate and a tailing altogether too rich in copper. The next scheme tested was to treat this product in a classifier, with the idea of drawing off the coarser mineral grains through the spigot as a concentrate and overflowing the finer material and tailing grains for table treatment. This worked out very well on a small scale, but it was found to be impractical to produce a concentrate from a classifier in large-scale operations. The next set of experiments was made in the laboratory and consisted of a classifying and jigging flow sheet, the spigot of the classifier treating the rough concentrate containing the coarser mineral and middling grains being jigged and the overflow sent to tables. This flow sheet gave very good results and was finally adopted, the rough concentrate being returned to the No. 2 classifiers preceding the fine Hancock jigs, where the coarser mineral and middling grains would pass through the spigot and on to the jig, the mineral grains being saved at once as concentrate and the middling grains going to the roll system for further grinding. The fine mineral grains and tailing grains being overflowed from the No. 2 classifiers, pass at once to the finishing table system.

*4. Results of test made on 500-ton section at Great Falls.*—A test made during October, 1911, on a 500-ton unit at the Great Falls mill gave the following results:

*Test Made at Great Falls Plant on 500-Ton Section.*

| Product | Solids per 24 Hours | Assay | | Copper per 24 Hours | Per Cent. of Original | |
|---|---|---|---|---|---|---|
| | | Cumu. | $SiO_2+Al_2O_3$ | | Solids | Copper |
| | Pounds | Per Cent. | Per Cent. | Pounds | | |
| Ore to section..... | 986,075 | 3.51 | 73.3 | 34,651 | 100.0 | 100.0 |
| Coarse concentrate .. | 67,000 | 13.83 | 24.5 | 9,266 | 6.8 | 26.7 |
| Fine concentrate..... | 166,595 | 10.00 | 13.7 | 16,652 | 16.9 | 48.1 |
| Slime concentrate.... | 42,610 | 9.49 | 53.8 | 4,044 | 4.3 | 11.7 |
| Total concentrate.. | 276,205 | 10.85 | 22.5 | 29,962 | 28.0 | 86.5 |
| Mill tailing..... | 469,120 | 0.40 | 96.3 | 1,900 | 47.6 | 5.5 |
| Slime tailing..... | 211,745 | 1.01 | 87.0 | 2,149 | 21.5 | 6.2 |
| Total tailing..... | 680,865 | 0.59 | 93.4 | 4,049 | 69.1 | 11.7 |
| Overflow loss..... | 29,005 | 2.21 | 82.5 | 640 | 2.9 | 1.8 |
| Total products.... | 986,075 | 3.51 | 73.3 | 34,651 | 100.0 | 100.0 |

Following are some screen analyses of the tailings made during the tests of the system at Great Falls:

*Screen Sizing Analyses of Tailings.*

| Screen Size Millimeters | Secondary Wilfley Tables | | | Coarse and Fine Primary Tables | | | No. 5 Classifier Overflow | | | Slime Plant | | | Total Mill Tailing | | | Total Mill and Slime Tailing | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Solids Per Cent. | Assay Cu. Per Cent. | Copper Per Cent. | Solids Per Cent. | Assay Cu. Per Cent. | Copper Per Cent. | Solids Per Cent. | Assay Cu. Per Cent. | Copper Per Cent. | Solids Per Cent. | Assay Cu. Per Cent. | Copper Per Cent. | Solids Per Cent. | Assay Cu. Per Cent. | Copper Per Cent. | Solids Per Cent. | Assay Cu. Per Cent. | Copper Per Cent. |
| On 2.38 | ..... | ..... | ..... | ..... | ..... | ..... | 0.10 | 0.66 | 0.13 | ..... | ..... | ..... | 0.06 | 0.66 | 0.08 | 0.04 | 0.66 | 0.03 |
| On 1.41 | ..... | ..... | ..... | ..... | ..... | ..... | 6.84 | 0.52 | 8.35 | ..... | ..... | ..... | 3.84 | 0.52 | 5.44 | 2.65 | 0.52 | 2.16 |
| On 0.84 | 2.31 | 0.23 | 2.39 | 1.65 | 0.43 | 1.74 | 45.42 | 0.43 | 46.49 | ..... | ..... | ..... | 26.41 | 0.42 | 30.99 | 18.11 | 0.42 | 12.31 |
| On 0.50 | 15.58 | 0.20 | 13.66 | 8.00 | 0.34 | 6.88 | 40.03 | 0.39 | 36.82 | ..... | ..... | ..... | 28.08 | 0.36 | 27.54 | 19.25 | 0.36 | 10.93 |
| On 0.35 | 35.39 | 0.21 | 33.42 | 10.23 | 0.44 | 11.29 | 7.61 | 0.45 | 8.21 | ..... | ..... | ..... | 15.70 | 0.30 | 13.09 | 10.77 | 0.30 | 5.20 |
| On 0.17 | 28.64 | 0.19 | 24.32 | 57.42 | 0.30 | 44.38 | ..... | ..... | ..... | ..... | ..... | ..... | 17.23 | 0.25 | 12.03 | 11.82 | 0.25 | 4.78 |
| On 0.07 | 14.98 | 0.14 | 9.08 | 19.54 | 0.37 | 18.18 | ..... | ..... | ..... | 16.62 | 0.32 | 4.40 | 7.31 | 0.24 | 4.78 | 10.23 | 0.28 | 4.55 |
| Through 0.07 | 3.10 | 1.25 | 17.13 | 3.16 | 2.19 | 17.53 | ..... | ..... | ..... | 83.38 | 1.38 | 95.60 | 1.37 | 1.60 | 6.05 | 27.15 | 1.38 | 60.04 |
| Total | 100.00 | 0.23 | 100.00 | 100.00 | 0.39 | 100.00 | 100.00 | 0.42 | 100.00 | 100.00 | 1.20 | 100.00 | 100.00 | 0.36 | 100.00 | 100.00 | 0.62 | 100.00 |

## VI. Comparative Data of Operation of Great Falls and Evans Systems at Anaconda.

The Great Falls flow sheet was installed in Section No. 1 of the Anaconda mill during the summer of 1912. The remodeled section was first put into operation during October, 1912, and by November 1 was running normally, treating its rated capacity of 1,500 tons per day.

The data contained in the accompanying table for the Great Falls system, remodeled Section No. 1, were obtained from a test made during March, 1913, and those for the Evans system, Sections 2 through 8, were taken from the metallurgical sheets covering the first six months of 1912. The tailing obtained during the test made on Section 1 assayed 0.48 per cent. of copper and the tailing for the first six months of operation of this section also averaged 0.48 per cent. of copper, so that we may say that the test conditions, which as a matter of fact were the regular operating conditions for the section, did not favor the section at all. Also by referring to tabulation on p. 1826 entitled "Test Made at Great Falls Plant on 500-Ton Section," we find a mill tailing assaying but 0.40 per cent. of copper. It is thought that the mill tailing for the remodeled section at Anaconda will be lowered in the near future to from 0.40 to 0.45 per cent. of copper as the operators become more familiar with the flow sheet. It should be noted that the ore treated during the test on the remodeled section happened to be of a lower grade than the usual run of concentrating ore at Anaconda,—2.98 per cent. of copper as against an average of 3.27 per cent. for Sections 2 through 8. Other conditions being constant, this lower copper content of the ore would of course lower the recovery.

The slime produced in the mill is not being concentrated at present, but a plant consisting of Dorr tanks and concrete multiple-deck round tables is being planned and will soon be erected for the treatment of all of the mill slime. With the complete treatment the remodeled section shows a net increase in recovery of copper of 3.0 per cent. over the original mill. The recoveries of silver and gold follow very closely that of the copper. The recoveries of copper in the slime from the two systems are based upon the results of treating these slimes in the 50-ton experimental slime division.

The Great Falls system as originally installed at Anaconda includes a "fine sand" division equipped with 20 8-ft. Callow tanks followed by 16 Wilfley tables. This division receives the total classifier overflow from the remodeled section. This pulp is thickened in Callow tanks and the thickened pulp fed to the Wilfley tables. These tables make concentrate, middling, tailing, and head-water slime. The middling is returned to the tables and the head-water slime unites with the overflow from the

*Comparative Data: Great Falls System and Evans System at Anaconda.*

| Products | Great Falls System, Remodeled Section No. 1 | | | | | Evans System, Sections 2 Through 8 | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | Per Cent. of Total | | Assay | | | Per Cent. of Total | | Assay | | |
| | Solids | Copper | Cu | Insol.[a] | FeO | Solids | Copper | Cu | Insol.[a] | FeO |
| | | | Per Ct. | Per Ct. | Per Ct. | | | Per Ct. | Per Ct. | Per Ct. |
| Original ore | 100.0 | 100.0 | 2.98 | 68.8 | 14.2 | 100.0 | 100.0 | 3.27 | 66.4 | 13.9 |
| Total concentrate | 32.9 | 77.0 | 7.00 | 25.8 | 35.3 | 32.3 | 77.0 | 7.78 | 25.3 | 34.2 |
| Slime | 19.6 | 14.1 | 2.14 | ..... | ..... | 17.2 | 11.2 | 2.13 | ..... | ..... |
| Overflow loss | 2.4 | 1.6 | 2.03 | ..... | ..... | 3.4 | 2.2 | 2.13 | ..... | ..... |
| Tailing loss | 45.1 | 7.3 | 0.48 | ..... | ..... | 47.1 | 9.6 | 0.67 | ..... | ..... |
| Total loss | 47.5 | 8.9 | 0.56 | ..... | ..... | 50.5 | 11.8 | 0.76 | ..... | ..... |
| Total products | 100.0 | 100.0 | 2.98 | ..... | ..... | 100.0 | 100.0 | 3.27 | ..... | ..... |

*Final Summary: Results When Slime is Concentrated.*

| | Great Falls System, Remodeled Section No. 1 | | Evans System, Sections 2 Through 8 | |
|---|---|---|---|---|
| | *Per Cent. of Total* | | *Per Cent. of Total* | |
| | *Solids* | *Copper* | *Solids* | *Copper* |
| Original ore | 100.0 | 100.0 | 100.0 | 100.0 |
| Mill concentrate | 32.9 | 77.0 | 32.3 | 77.0 |
| Slime concentrate | 3.1 | 8.5 | 2.8 | 5.6 |
| Total concentrate | 36.0 | 85.5 | 35.1 | 82.6 |
| Mill loss | 47.5 | 8.9 | 50.5 | 11.8 |
| Slime tailing | 16.5 | 5.6 | 14.4 | 5.6 |
| Total loss | 64.0 | 14.5 | 64.9 | 17.4 |
| Total products | 100.0 | 100.0 | 100.0 | 100.0 |

[a] "Insoluble" is silica plus alumina.

Callow tanks and leaves the plant as slime. This sand division has been recovering about 3.0 per cent. of the copper in the original ore, so that if we include the work of this fine-sand division the net mill recovery in the table on p. 1829 would be 80 per cent. instead of 77 per cent. However, we have found that a reciprocating table is not well adapted to the treatment of this fine sand (approximately 200 mesh and finer), and as soon as the slime plant is completed this fine-sand division will be discarded. I have therefore eliminated this division from this paper, and all of the classifier overflow from the remodeled section is included in the product termed "slime."

The increase in the recovery of copper in the remodeled section over that of the other sections using the Evans flow sheet is due for the most part to the saving of the fine free mineral which is lost in the tailings from the Evans jigs and the fine free mineral which is lost in the Wilfley table tailings of the Evans system. Another decided advantage of the Great Falls flow sheet is its capability of producing a very high grade of fine concentrates without any sacrifice in the recovery of the valuable minerals. This advantage is due entirely to the excellent classification resulting from the hindered-settling classifiers preceding the Wilfley tables. The system is operated at Anaconda to produce a fine concentrate containing about 25 per cent. of insoluble, as this grade of concentrate meets the present smelter requirements. However, during tests made at the Great Falls plant fine concentrates running as low as 10 to 15 per cent. of insoluble were produced in a 500-ton section. The insoluble in the coarse concentrate cannot be reduced below 18 to 22 per cent. on account of the fact that this amount of insoluble is combined with the mineral.

I desire to thank the following gentlemen for valuable data supplied: C. W. Goodale, E. P. Mathewson, A. E. Wheeler, W. A. Estabrook and Arthur Crowfoot.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.
[SUBJECT TO REVISION].

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Butte meeting, Aug. 18 to 21, 1913, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1913, when Vol. XLVI. of the *Transactions* will go to press. Any discussion offered thereafter should preferably be in the form of a new paper for publication in Vol. XLVII. (with suitable cross references in both volumes).

# Great Falls Converter Practice.

BY ARCHER E. WHEELER AND MILO W. KREJCI, GREAT FALLS, MONT.

(Butte Meeting, August, 1913.)

The Boston & Montana Reduction Works at Great Falls, Mont., was formerly the reduction works of the Boston & Montana Consolidated Copper & Silver Mining Co., and continued as the reduction plant for the independent company until June 1, 1910, when the consolidation of a large number of the operating companies of Butte was effected. It then became merged in the Anaconda Copper Mining Co. as the Boston & Montana Reduction Department of the Anaconda Copper Mining Co.

The history of copper converting at this plant, while not covering the entire period of the commercial use of the process in this country, does cover by far the larger part of it, and as the operations and appliances of the process have undergone many changes during the time, it may, perhaps, be of some interest to put the record of these changes into permanent form.

The plan followed in the preparation of this paper has been to give first a brief chronological record of the principal events and changes and then, even at the risk of some repetitions, to give more detailed accounts of some of these changes under separate headings.

The headings for the complete paper are as follows:

1. General Chronological Review.
2. Mechanical Manipulation.
3. Casting of Finished Copper.
4. Tamping of Linings.
5. Early Acid Linings.
6. Acid Linings With the Use of Ore.
7. Introduction of Flux Through the Tuyeres.
8. Bottom Blast.
9. Disposal of Converter Slag.
10. Development of Different Classes of Converters After Class I.
11. Basic Lining.
12. Number and Size of Tuyeres.
13. Cast Copper Tuyeres.
14. Efficiency of Air.
15. Size of Converter Mouth.
16. Height of Converter.
17. Summary.

## General Chronological Review.

Ground was broken for the general concentrating and smelting plant in the spring of 1891. The first departments to start operations were the concentrator in March, 1892, and the Brückner roasting and reverberatory smelting departments in April, 1892. The converter department

started operations in August, 1892. There were no blast furnaces until April, 1893. Therefore the first work of the converters was on reverberatory matte.

During the interval from April, 1892, to August, 1892, while the reverberatory furnaces were producing matte, but before the converters were

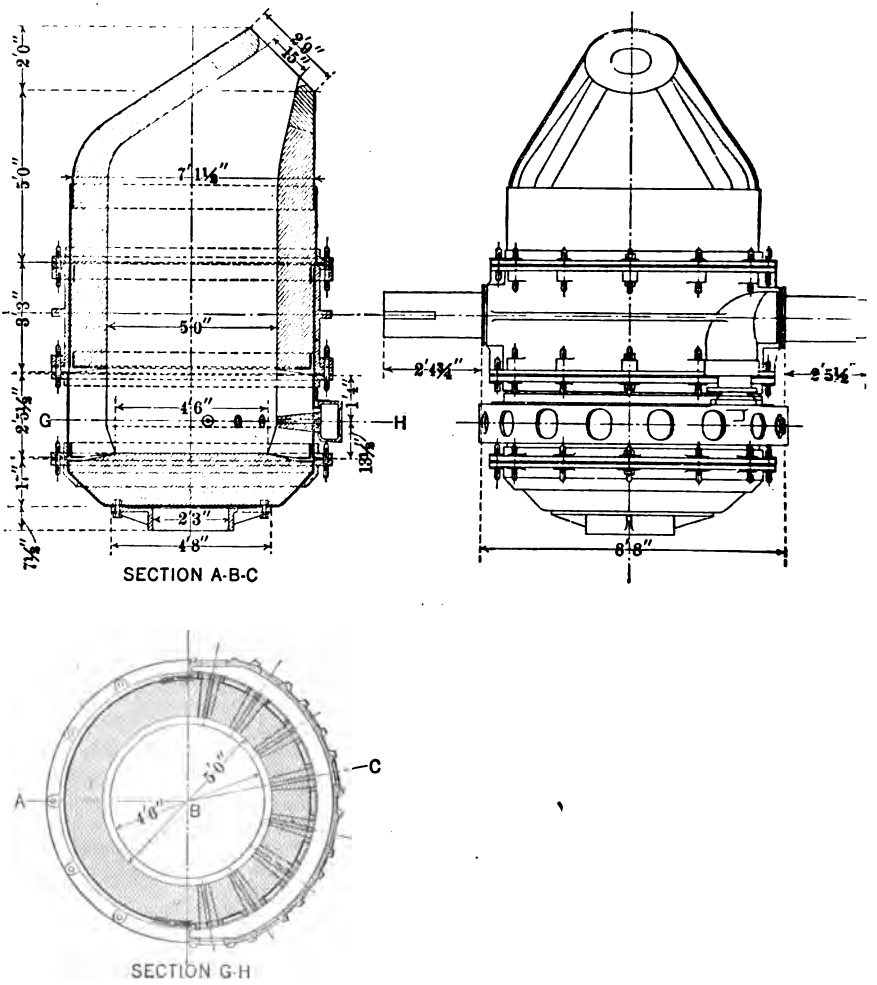



FIG. 1.—FIVE-TON CONVERTER.

installed, the matte was cast into molds, broken up when cold, sacked, and shipped to Lewisohn Bros., the predecessors of the United Metals Selling Co., at New York.

The first converters followed very closely in their design those used in steel practice, and were called five-ton converters. The design of this

converter is shown in Fig. 1. These converters were of the upright type, 14 ft. 9.5 in. high over all, circular in cross-section, 7 ft. 1.5 in. in diameter with eight side-blast tuyeres, and arranged for acid lining. They were tilted by hydraulic power, both the tilting and the manipulation of the air blast for the whole plant being controlled from a central "pulpit"; this method of control continued until 1899.

The plant started with two stalls. The converters were lined by hand, in place in the stalls, and it was soon found that the two stalls were not

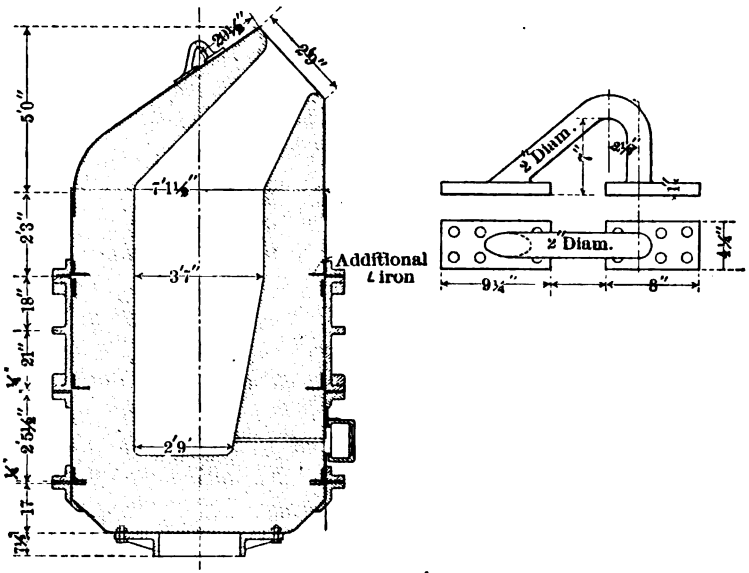



FIG. 2.—ADDITIONAL WORK ON CONVERTERS.

sufficient to allow proper time for drying and two more stalls were added of the same type as the first.

Whether or not the particular shape of lining shown in Fig. 1 was used is not recorded, but that it was not used long, if used at all, is evident from Fig. 2, which is from a drawing dated Mar. 18, 1893, seven months after the starting of the plant, which shows the shape of lining then adopted, and which in a general way, but with minor changes as to dimensions, was used as long as acid-lined converters were used. The chief characteristic is the thickening of the lining at the tuyere line. It was evidently early found that the greatest wear was at this point, and the lining was put in accordingly.

This same date, Mar. 18, 1893, is the first record of a plan for increasing the number of tuyeres, another drawing showing 17 instead of eight.

The first record of a design or attempt to put flux into the converter with the blast is a drawing dated Apr. 21, 1894, showing a sand box delivering sand into the wind box (see "Introduction of Flux Through the Tuyeres"). A trial of putting flux in with the blast was made in 1896.

In 1895 four more stalls were put in, making eight altogether, and all of the original type.

The earliest record which we find of a brick lining is a drawing dated Feb. 27, 1895, showing a Class I converter with a brick lining backed with ordinary lining clay. There is nothing to show whether the brick were acid or basic, nor is there any record as to whether or not it was ever tried. All we can say is that the matter had been thought of seriously enough to have a drawing made.

In 1894 plans were discussed for a machine for putting in the linings but it was not until about Apr. 1, 1896, that the machine had been put in and started.

In April, 1897, or perhaps in March, a basic (magnesite brick) lining was used, of which more detailed mention will be made under "Brick and Basic Linings".

In 1897 a converter was also operated into which ore was introduced through the mouth to furnish flux and save the lining

Another event of 1897 was the use of two bottom-blast converters which will be mentioned under "Bottom Blast".

On Dec. 4, 1899, a converter was lined with ore from the company mines and following this trials were made with linings two-thirds ore and one-third clay, and one-half ore and one-half clay. These linings were used until Apr. 20, 1900, when it was stopped until Aug. 27, 1900, after which linings of ore and clay were used until basic lining was adopted.

In 1900 the first change was made from central hydraulic and blast control for the plant, to individual control immediately at the converter.

In 1901 designs were begun for a converter slag casting machine, and it was completed and put into use on May 23, 1902.

In 1901 rights for the use of Walker copper-casting machines were purchased and on Mar. 31, 1902, a refining furnace for handling molten converter copper was started, this furnace delivering its copper to a Walker machine.

The furnace was abandoned on Jan. 6, 1903, and the machine used alone until a larger casting machine could be designed and built. This new machine went into service on Oct. 8, 1903.

In 1901 further trials of bottom blast were made.

In 1902 we began to discuss larger converters but it was not until Nov. 29, 1904, that the first Class II (larger than Class I) converter was actually put into service.

On Apr. 5, 1906, instructions were issued for the installation of the

second Class II stall, and extra bowls were built so that when this second stall went into service we had four of the Class II bowls or bottoms.

We felt that further improvement was possible and a larger converter called Class III was begun and the first one went into service on July 18, 1907.

On Jan. 21, 1907, a trial of smelting dried fine concentrates by dumping them into the converter was tried. Following this, in the same year, the blowing of these concentrates through the tuyeres was discussed and on Feb. 20, 1908, the drawings for an apparatus to do this were completed.

On Nov. 29, 1908, and following, this was tried, but never adopted as regular practice.

After this had been in operation some time we began the design of a larger converter called Class IV. The designs were completed on Mar. 24, 1909, but it was not until Feb. 4, 1910, that the first Class IV was put into service.

This became the standard and all Class I and Class II converters were abandoned. The Class III converters were retained as they gave good service and fitted into the same stalls as the Class IV.

On Mar. 9, 1911, the first run of what finally developed into the regular use of basic linings was started. It proved so successful that, as rapidly as possible, the department was changed over to basic lining.

Discussion was begun for plans for a still larger converter to be called Class V. This discussion resulted in the building of a converter 20 ft. in diameter, basic lined, and of the upright type. The first converter of this class was started on Aug. 3, 1912, and has proved entirely successful. While the Class IV converters are still in use the Class V is now considered the standard for the plant, and will be the only one used in the remodeled plant.

Experiments have been made on the use of different sizes of tuyeres, and on the efficiency of the air, and these subjects will be treated in a little more detail.

With this brief outline we will take up individually the subjects listed.

## Mechanical Manipulation.

The original installation provided for the manipulation and tilting of the converters by hydraulic power. The converter revolved on trunnions, through one of which the air blast entered and was then carried by a passage cast in the trunnion section to the wind box. The other trunnion was fitted with a pinion which meshed with a rack attached to the piston rod of a hydraulic cylinder. The I-beams of the converter stand carrying the trunnion bearings formed the guide in which the rack moved back

and forth when driven by the piston, and thus rotated the converter in the usual manner.

The hydraulic pressure in the cylinders was normally 500 lb. per square inch. No pump was used at first, this pressure being obtained by supplying water under a static head of 125 lb. per square inch from tanks on a hill back of the smelter, to intensifiers which raised the pressure to 500 lb. per square inch.

The entire control of the air blast and of the tilting was from a central pulpit on the opposite side of the building from the converters and in full view of them. The converter crew directed the "tilter," as he was called, as to what was to be done by signals, much as train crews handle a train by signals.

Each converter was equipped with a four-way hydraulic valve, which, by the way, was of very complicated design, for rotating the converter in its stall, and also with a globe valve for the control of the air.

This system was used without change until the second set of two converters was installed, when the intensifiers were discarded and a steam-driven hydraulic pump was installed to pump directly into the main at 500 lb. pressure per square inch.

In order to provide ample water for the sudden demands of tilting several converters at once, an accumulator was installed which was simply a large piston of ample displacement, loaded on top with sufficient weight of cast iron, etc., to develop the pressure of the main over the area of the piston. This accumulator was connected to the main, and when the converters were not being tilted the pump would deliver into this accumulator, raising the piston to its full height before the pump would be stopped by the governor. Thus in case of a sudden demand for water at a rate greater than the capacity of the pump the accumulator would descend and force water into the main.

Mechanical difficulties were encountered and a great deal of work was required to keep this system in good working order. A complicated system of both water and air pipes was necessary, but except for improvements in details such as the substitution of a different type of water valve and many smaller details the general system was retained until 1900.

On July 25, 1900, a drawing was completed of an arrangement for the control of both the hydraulic tilting and the air blast immediately at the converter by one of the converter crew. With a few minor changes this proved to be very successful and convenient, and as rapidly as possible all converters were changed over.

This system continued in use without change until the installation of the first Class II converter. Electrical control of the tilting was installed on this converter and first put into use on Nov. 29, 1904, when the converter was first used, although the first conception of it was several months

previous. This is, we believe, the first application of electric control to a converter.

Very briefly, the converter was provided with a gear on one of its trunnions, this gear meshing with a pinion driven by a shaft on which was a worm gear meshing with a worm driven from the motor. Direct current at 500 volts was used.

No serious electrical troubles were encountered in the use of this tilting and as rapidly as new converters were installed, the electrical control was used, not only on the Class II converters but on all the larger classes following it.

## Casting of Finished Copper.

The original plan for casting the finished copper was to pour it from the converter into a ladle with a bottom discharge hole, this discharge hole to be plugged with a silica block on a rod which was protected by clay blocks strung on it as in the steel practice of casting ingots. The method was tried as planned when the plant started in August, 1892, but the plug very promptly froze in the hole and could not be pulled

After a few further attempts this method was abandoned and the method adopted of pouring the copper from the converter into a ladle, which was carried by the crane over molds set up on beams on the floor, and the copper tapped from a hole through the side of the ladle into the molds. This was not continued very long.

Until some other means could be provided the molds were set up on beams lengthwise of the converter floor; the copper was poured from the converter into a ladle hanging on the crane hook, and the crane then poured it into the molds which were turned over by hand; the copper was dumped onto the floor and taken away and trimmed by manual labor.

The next arrangement, immediately following this, was a spout curved vertically to fit the arc of travel of the mouth of the converter, this spout being mounted on wheels and designed to serve two converters standing side by side, by being set down by the crane in front of whichever one was to cast copper. The spout delivered the molten copper into molds which were mounted on a truck moved back and forth by hand on a track lengthwise of the converter floor. Each truck carried seven pig molds, or, when anodes were cast, four anode molds.

Considerable difficulty was encountered in keeping the track, which was entirely in the converter floor, free of obstruction, and the method also required considerable crane work. A change was made by using a somewhat similar spout to the one described above, but providing one for each converter stall, and hinging it on the stall, so that it could be swung into place by the converting crew when required. Drawings for this

change were completed and dated September, 1892. In Fig. 3 one of these spouts can be seen.

The trucks carrying the molds were run on tracks under the converter and at right angles to the converter floor. Pouring into the molds in both these cases was done directly from the converter by gradually tilting it.

FIG. 3.—SHOWING CURVED SPOUT HINGED TO CONVERTER STALL.

When the molds were filled, the truck carrying them was run out on the floor of the "copper shed" back of the converters. The copper was allowed to cool, then dumped on the floor, trimmed by hand and trucked into railroad cars.

As the plant included an electrolytic refinery, the necessary anodes to supply it were cast, and the rest of the copper was cast as pig and shipped to Pawtucket, R. I., for refining.

This method of casting was continued until 1902. During this time it became evident that it was a crude, laborious, and uneconomical method of casting. It was hard and laborious, because the men were moving heavy trucks with heavy loads of hot copper over uneven floors on which obstructions of dirt, scraps of copper, etc., were constantly falling and blocking the wheels, and the work of removing the copper, particularly the anodes, from the molds was very heavy and hot work.

There was also a great deal of spilling and slopping of copper on the

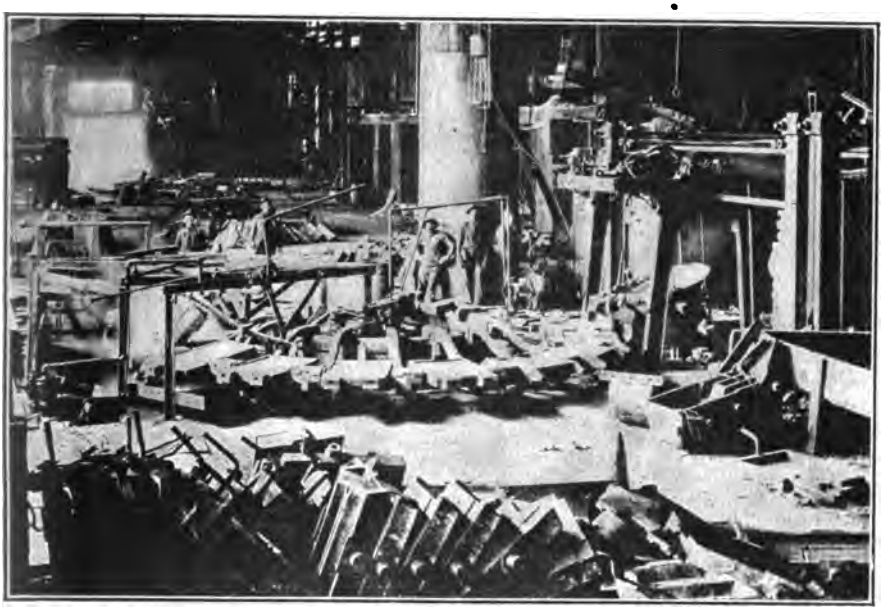


FIG. 4.—EARLY ARRANGEMENT OF CASTING MACHINE.

floor and over the edges of the molds, due to not being able to control the converter quickly enough.

A great disadvantage was the lost time of the converter. During all the time of pouring, which sometimes occupied as much as 1.5 hr. or more, the converter was not working. This amounted to a large percentage of the time and the elimination of this delay would add very materially to the possible production from a given equipment of converters.

Plans for ladles on trucks, pouring into stationary and also movable molds, were made as early as 1895 and 1896, but were not adopted.

In 1901 the trial was made of transferring the finished copper from several converters to one from which the casting was done. The idea was to lose the time of one converter only instead of each of them. It was found that the copper chilled in the converter to a great extent, due prob-

ably to its being exposed to the air while pouring from the first converter into the ladle, again in pouring from the ladle into the converter, and again the chilling of the small stream running from the mouth into a rather long spout for casting. Further than this the method did not improve the conditions of unloading the molds, and trimming and loading the copper.

In 1901 Mr. Klepetko, then manager, purchased the right to use some Walker casting machines. Plans were immediately begun for their use

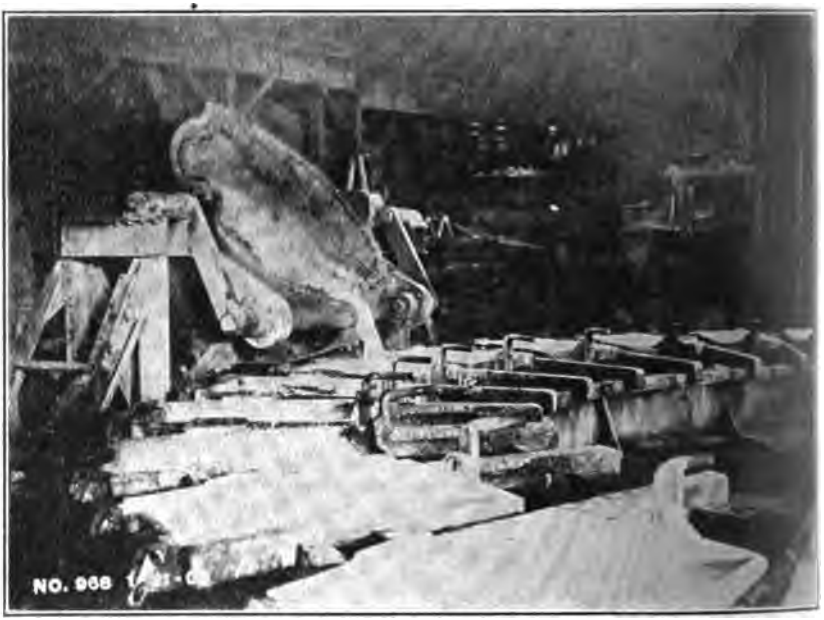



FIG. 5.—LADLE IN POURING POSITION ON CARRIAGE.

and the final plan adopted was to pour the copper from the converter into a ladle hanging on the crane hook. This was immediately carried by the crane to a coal-fired refining furnace and poured into it through a side door. As soon as sufficient copper had been accumulated, refining by rabbling and poling was begun.

As only one furnace was installed, it being the intention to await the results on this before erecting the second, the copper which was being produced by the converters during the refining of a charge was cast in the old way into pigs for shipment to the Eastern refinery. When the charge was refined it was tapped into the molds on the casting machine by allowing it to run over a bay which was cut down as required.

The first charge from this furnace was cast on Mar. 31, 1902. Considerable difficulty was encountered with both the furnace and the

machine. One of the principal difficulties with the machine was that it was not large enough to allow the copper to cool properly before dumping into the bosh.

Comprehensive experiments were being carried on at the electrolytic refinery at this time to determine whether or not there were sufficient advantages in the refined anode as compared with the rough converter anode to pay for the extra cost of refining. It began to develop that

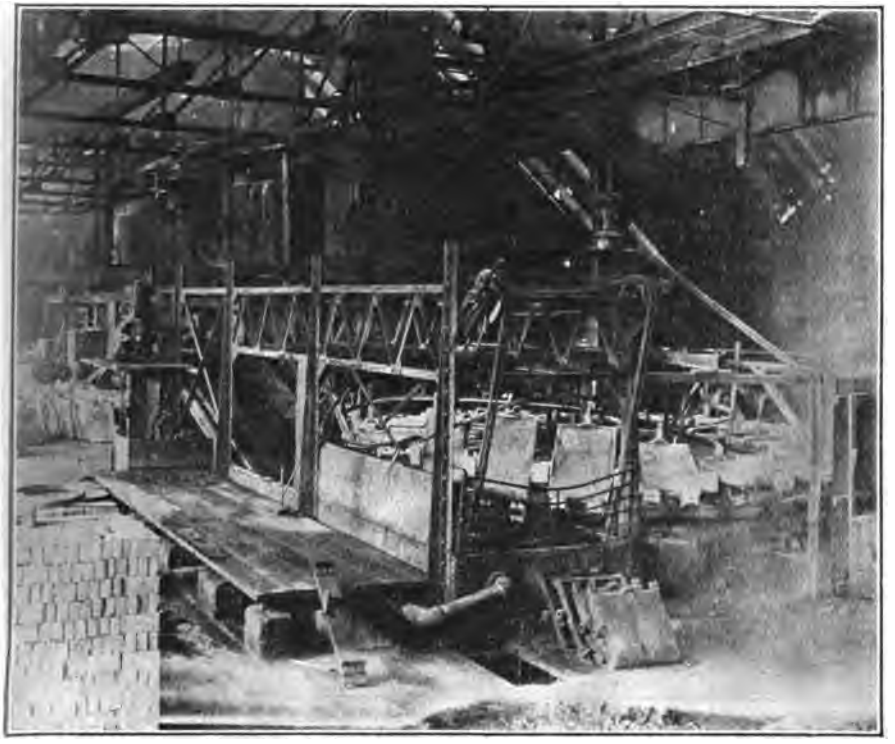


FIG. 6.—GENERAL VIEW OF CASTING MACHINE.

there were not sufficient advantages and therefore it was determined to abandon it. This was done on Jan. 6, 1903.

Arrangements were immediately made for the removal of the necessary parts of the furnace to allow a car carrying a ladle of molten converter copper to run from the converter floor to the casting machine. The copper was then cast by pouring it directly from the converter into a ladle which was placed on the transfer car by the crane, run to the casting machine and cast as rapidly as the size of the machine would allow. The first use of the machine in this way was on Mar. 30, 1903. A picture of the arrangement is shown in Fig. 4.

Designs were immediately begun of a machine of the Walker type, but very much enlarged and elaborated, which would allow the copper to be cast as fast as the stream could be poured from the ladle into the mold, and provide cooling time for this rate of casting. This meant primarily a machine of larger diameter. It was so placed that the crane could set a ladle of copper directly on the pouring carriage. This carriage was an entirely new arrangement hydraulically operated by a vertically moving piston, but the particularly novel feature was the

FIG. 7.—PNEUMATIC CYLINDER HOIST HANDLING ANODES.

double-hinging arrangement whereby the combination of all the motions produced the same effect as though the ladle were turning about trunnions located almost at the spout, thereby reducing the movement of the spout to such a small amount that the spatter and slop was very little. Fig. 5 shows a ladle in a pouring position on the carriage. Fig. 6 is a general view of the machine in action. The stream from the ladle can be seen at the right of the picture. The casting wheel is revolving clockwise and the molds with anodes which have just been cast and cooled can be seen in the center of the picture in different stages of dumping. On the left can be seen the anodes stacked on the floor. The copper is dumped automatically into a bosh containing water. It is carried up from this by

a drag conveyor and delivered to the copper-shed floor. Here it is picked up by means of a pneumatic-cylinder hoist which travels on the lower flange of an overhead I-beam 38 ft. long. One end of this beam is centered at the delivery end of the bosh and the other end travels through an arc of a little less than 90°, thus covering a large storage floor. Fig. 7 shows a view of this pneumatic cylinder in action and the anodes as they are stacked by it.

The machine is electrically operated, and is reversible in its motion, which was not true of the original machine. It was put into service on Oct. 8, 1903, and has been eminently satisfactory.

Three men per shift of eight hours, one manipulating the machine, one attending to the molds, etc., and one taking away copper, will easily cast, and stack on the storage floor, 200 tons or more of copper in 24 hr. On smaller quantities the machine is operated with less labor.

## Tamping of Linings.

The first linings were tamped by hand, with the converter standing in the stalls, the caps having first been removed. It early became evident that an improvement in the life of the lining could be made if more clay could be put in and also tamped in harder.

In 1894 plans were begun for a machine to do this work, and the machine was built, installed, and its use begun about Apr. 1, 1896. Fig. 8 shows a view of this machine with a converter bottom on it ready for tamping.

The cap was removed from the converter and the bottom was then taken out of the stall by the crane and set on a revolving table. Over this table was an air cylinder set in a frame allowing the cylinder to be moved by power up or down to any desired position and clamped. The frame and cylinder swung horizontally so as to cover the whole distance from the center of the converter to the circumference, so that the combination of all the motions covered the whole horizontal section of the converter, and the up-and-down movement of the cylinder allowed for the varying heights of the lining as it was being put in. The movement of the piston and rod which carried a tamping shoe on the bottom was controlled by the operator by means of a hand lever, an upward movement raising the piston, and a downward movement lowering it and striking a blow.

The machine was installed immediately in front of the clay-grinding machines so that the material delivered from them was shoveled directly into the converter bottom on the lining machine and tamped in. The lining was tamped solid, and the tuyeres punched afterward by driving a sharp-pointed bar through the lining for each hole. No attempt was

made to line the caps by machine. The machine was very successful from the beginning and was used as long as acid linings were used.

A comparison of the linings by months in 1895 and 1896 is given in Table I. Machine tamping began about Apr. 1, 1896.

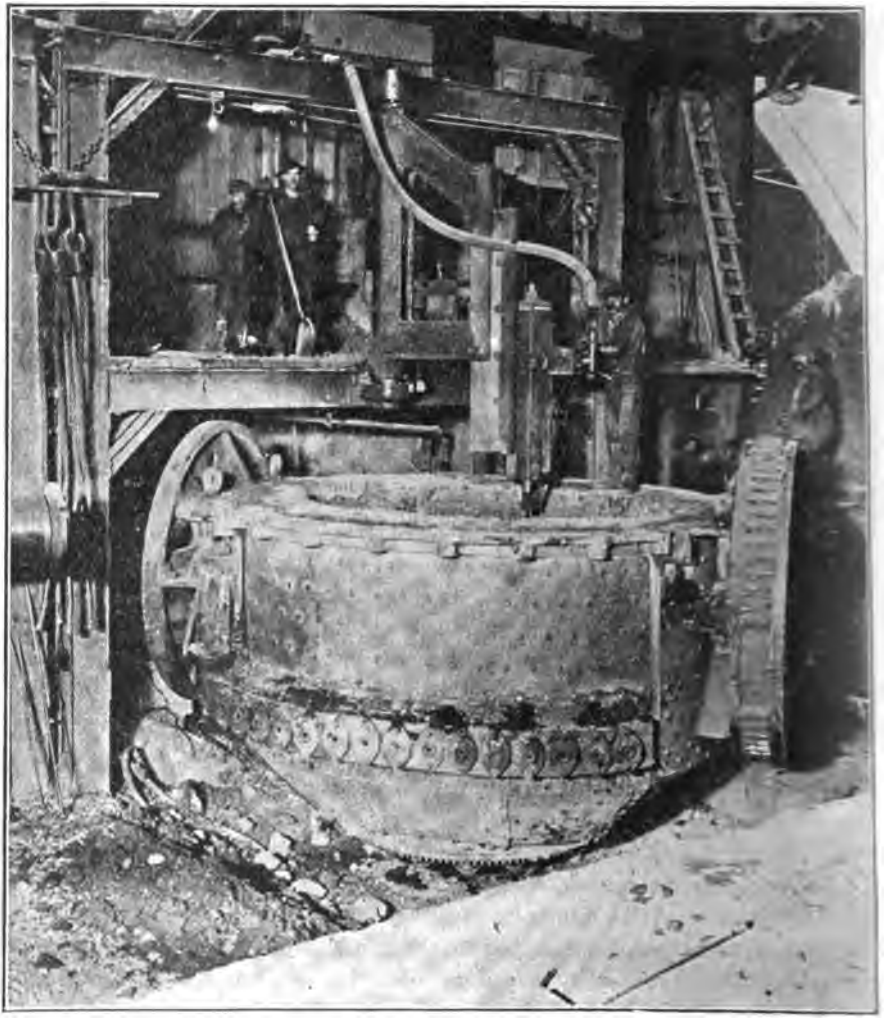


FIG. 8.—MACHINE FOR TAMPING LININGS.

Although the grade of the matte will average higher after Apr. 1, 1896, than before, it is not, we think, sufficiently different to account for the increased production per lining and the decreased consumption of lining material per ton of copper. The apparent decrease in the tons of lining

material in one lining may be due to a change in the shape of the core around which the lining was tamped.

It appears that the decreased consumption of lining material per ton of copper produced is due to the harder tamping of the material and the consequent decrease in caving in of the linings.

Just before the trials of basic lining were made, which led to the final adoption of them, a new tamping machine, striking a heavier blow, was

TABLE I.—*Comparison of Performance of Linings.*

| Month | Grade of Matte Per Cent. Copper | | Lb. Copper Produced Per Lining | | Tons Lining Per Ton Copper Produced | |
|---|---|---|---|---|---|---|
| | 1895 | 1896 | 1895 | 1896 | 1895 | 1896 |
| January | 52.5 | 50.3 | 30,416 | 26,840 | 0.385 | 0.505 |
| February | 48.9 | 51.6 | 26,046 | 26,740 | 0.485 | 0.536 |
| March | 45.2 | 54.7 | 23,443 | 26,800 | 0.602 | 0.369 |
| April | 47.3 | 54.6 | 24,688 | 32,631 | 0.680 | 0.389 |
| May | 47.4 | 56.2 | 23,720 | 31,566 | 0.577 | 0.444 |
| June | 46.6 | 57.1 | 25,650 | 32,644 | 0.663 | 0.435 |
| July | 47.8 | 49.7 | 24,186 | 31,999 | 0.624 | 0.498 |
| August | 46.8 | 50.0 | 23,165 | 29,081 | 0.712 | 0.637 |
| September | 48.2 | 54.1 | 26,800 | 33,799 | 0.596 | 0.457 |
| October | 48.4 | 57.2 | 28,902 | 38,561 | 0.453 | 0.407 |
| November | 48.2 | 58.4 | 28,271 | 36,530 | 0.644 | 0.346 |
| December | 47.9 | 57.4 | 25,848 | 34,101 | 0.526 | 0.512 |
| Year | 47.9 | 54.3 | 25,818 | 31,443 | 0.581 | 0.457 |

designed and had been partly built, but was never completed on account of the adoption of the basic lining as regular practice.

## EARLY ACID LININGS.

When the first converter was started in August, 1892, the lining was a mixture of a siliceous rock from near Boulder, Mont., and a sand rock from near the bank of the Missouri river on the smelter property. Analyses of these materials are not available, but the Boulder material was a clean-looking pure quartz, and the sand rock was a more aluminous material furnishing the binding properties.

The lining was tamped into the converter by hand, with the converter in place in the stall. Whether or not it was tamped in the form shown in Fig. 1 we do not know, but that such shape was not continued long is certain, because in November, 1892, the center around which the lining was tamped was 3 ft. in diameter and set so as to make the lining much thicker on the tuyere side than on the opposite side.

During all the time that the original type or Class I was in use, the general shape of this lining was maintained with, however, changes from time to time in the thickness of the lining on the tuyere side on account of the faster wearing of the lining at this point.

During this early period a fair average performance of these converters was as follows, the figures being for 40 linings in January, 1895:

| | |
|---|---|
| Total number of charges blown | 217 |
| Total copper cast, tons | 582.488 |
| Total time blowing on finish, min | 11420 |
| Average number of charges per lining | 5.425 |
| Copper per average charge, tons | 2.685 |
| Copper per lining, tons | 14.56 |
| Time blowing on finish per charge, min | 53 |
| Time blowing on finish per ton of copper, min | 19.6 |

The matte during this month of January averaged 53.2 per cent. Cu. The time quoted above is for the finish period only as this is comparable for all grades of matte and will be quoted in other parts of this paper for purposes of comparison.

The mixture of quartz and sand rock above referred to was used until October, 1895, when a sand rock obtained near the stack on the company property was substituted. This was known as B. & M. sand rock. An average analysis of this sand rock was: $SiO_2$ 70.4; $Al_2O_3$ 18.9; Fe 3.4; CaO 0.8; MgO 0.5 per cent. The use of this material was continued either for the entire lining or mixed with other materials as long as acid linings were used.

## Acid Linings with Use of Ores.

The first use of ore in the converter was in 1895. At this time the rich pieces of copper glance were picked out at the mines, sacked, and shipped to the smelter separately from the other ores. This ore was thrown into the converter during the blow, but was not used in the lining.

In 1897 an extended experimental run was made of putting first-class ore into the converter through the mouth while the converter was blowing. The record of the experiment is given in Table II.

It is to be remembered that the converter in these runs was Class I side blast, acid lined, and the usual production of copper per lining was probably not over about 14.5 tons, from which it can be seen that the life of the lining was materially increased. The records as to why the practice was not continued are not complete.

On Dec. 4, 1899, the first converter, an original Class I, was lined with ore, unmixed with other material. Second-class West Colusa, that is the lower grade West Colusa, was used, the analysis of which was as fol-

lows: Cu 8.79; $SiO_2$ 63.8; Fe 9.6 per cent. It was ground and handled the same as ordinary lining clay or sand rock, and tamped in with the machine in the same way as other linings. The converter was fired and dried in the usual way and put into service on Dec. 5, 1899. This ore

TABLE II.—*Experiment in Feeding Converter while Blowing.*

| Date 1897 | No. of Charges | Copper Produced Lb. | Ore Used Lb. |
|---|---|---|---|
| FIRST RUN | | | |
| Jan. 28 | | 31,200 | |
| Jan. 29 | | 32,470 | |
| Jan. 30 | | 4,200 | |
| Feb. 1 | | 3,400 | |
| Feb. 2 | | 8,400 | |
| Feb. 3 | | 10,400 | |
| Total for the run | | 90,070 | |
| SECOND RUN | | | |
| Feb. 16 | 4 | 24,560 | |
| Feb. 17 | 6 | 63,650 | Not known |
| Feb. 18 | 5 | 47,700 | |
| Feb. 19 | | 30,580 | |
| Total for the run | | 166,490 | |
| THIRD RUN | | | |
| Mar. 3 | 3 | 12,140 | |
| FOURTH RUN | | | |
| Mar. 15 | 3 | 15,150 | |
| Mar. 16 | 1 | 3,400 | |
| Mar. 17 | 5 | 21,200 | 8,100 |
| Mar. 18 | 2 | 14,000 | 6,000 |
| Mar. 19 | 3 | 14,600 | 18,790 |
| Mar. 20 | 4 | 22,800 | 30,570 |
| Mar. 21 | 3 | 17,000 | 24,940 |
| Total | 21 | 108,150 | 88,400 |

lining was continued until Dec. 9, 1899, and a great deal of trouble was reported in making the linings last.

On Dec. 10, 1899, a change was made to a mixture of two-thirds second-class ore and one-third B. & M sand rock. An analysis of this mixture was: Cu 7.0; $SiO_2$ 60.4; $Al_2O_3$ 14.66; Fe 6.3; S 9.6; CaO 0.3; MgO 0.5 per cent.

The linings continued to wear out more rapidly than those entirely of B. & M. sand rock and on Dec. 14, 1899, the mixture was changed to one-

half second-class ore and one-half B. & M. sand rock, and was continued so until Apr. 20, 1900, when the ore was discontinued for a time.

On Aug. 27, 1900, a mixture of second-class ore and B. & M. sand rock was again used, and from that time on, as long as acid linings were used, mixtures of sand rock with ore, refinery slag, electrolytic-plant refuse, copper precipitates from the mine waters at Butte, etc., were used in the linings. As much copper-bearing material as possible was used.

The practice of dumping ore as it came from the mine into the converter became the regular practice in February, 1905, and has always continued, both through acid- and basic-lining practice.

On Jan. 21, 1907, a trial of smelting dried fine concentrates by dumping them into the mouth on top of the charge when the converter was turned down was tried. Only two charges were tried. In the first one 1.5 tons of concentrates were used in a charge finally producing 6.05 tons of copper, and in the second one 2.5 tons of concentrates were used in a charge producing 6.79 tons of copper. There was, of course, some blowing out of concentrates when the converter was turned up, but no attempt was made to determine what this amounted to. The smelting was entirely successful and rapid.

In discussing the results of this test it was decided to try introducing the concentrates through the tuyeres and this will be mentioned under the heading "Introduction of Flux Through the Tuyeres."

## Introduction of Flux Through the Tuyeres.

Very early in the use of the converters it appealed to the management that the weak and expensive part of the process was the lining, and many suggestions and plans were made to improve this feature. One of the first suggestions was the introduction of siliceous flux through the tuyeres by means of the air blast.

A drawing was completed on Apr. 21, 1894, showing a sand box attached to the converter and discharging sand into the wind box from which it would be carried through the tuyeres and into the converter by the blast. This is shown as Fig. 9, and is the earliest record of any plan for this work.

We do not know whether or not this arrangement was ever put into service, but it is the recollection of men still connected with the plant that sand was blown in through the tuyeres as early as 1896. It was not continued more than a few days, as a great deal of trouble was experienced from the material blowing back into the punchers' eyes when the tuyeres were opened for punching.

No further attempts were made to introduce material through the tuyeres until after the experiment of smelting fine concentrates by dump-

ing them into the converter as mentioned under "Acid Linings with Use of Ores" was tried.

After the above experiment no actual steps were taken to continue the experimenting until the latter part of 1907, when plans were discussed and a drawing completed Feb. 20, 1908, of an apparatus for the introduction of these fine concentrates through the tuyeres. The proposed appa-

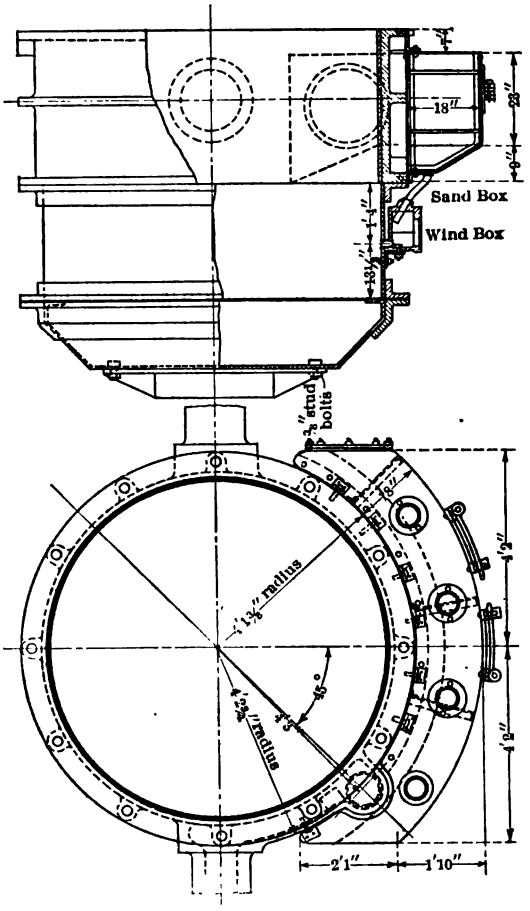



FIG. 9.—CONVERTER EQUIPPED WITH SAND BOX.

ratus was installed but it was not until Nov. 29, 1908, that it was tried, these trials being on regular acid-lined Class I converters.

It had been found by previous experiments that, on account of the tuyere opening being some distance above the bottom of the wind box, concentrates would accumulate in the box below the tuyeres, and when the converter was turned down these concentrates would run down into

each end of the wind box and plug the end tuyeres. To overcome this difficulty the bottom of the wind boxes of the two converters used in the experiments was filled with cement up to the tuyere openings.

During November and December, 1908, and January, 1909, 23 charges were run with the introduction of dried fine concentrates through the tuyeres. In the following tabulation only 22 charges are given, as no observation of the time of blowing was made for one of the charges.

As a matter of comparison 30 regular charges without concentrates are given for practically the same dates. The data for these latter charges were not especially obtained by observers as were those on the concentrate charges, but were taken from the regular daily reports.

The following is the analysis of the concentrates charged on Dec. 20, 1908, and is a fair analysis of those used during the entire run: Cu 8.45; $SiO_2$ 20.2; $Al_2O_3$ 4.7; Fe 28.2; S 34; CaO 0.2 per cent.

The tests were run more as qualitative tests to determine whether or not the scheme would work than as quantitative tests to determine the relations between matte, concentrates, etc., but nevertheless the results are quoted below:

| | Regular Charges | Charges With Concentrates |
|---|---|---|
| Number of charges | 30 | 22 |
| Total copper produced, tons | 84.7 | 49.45 |
| Total time matte to white metal, min. | 3,162 | 2,762 |
| Total concentrates used, tons | None | 47.594 |
| Copper per charge | 2.823 | 2.248 |
| Minutes blowing per ton of copper | ..... | ..... |
| Matte to white metal | 37.3 | 55.9 |
| Concentrates used per ton copper produced | None | 0.9614 |

A thermal calculation on matte and concentrates made previous to these experiments showed a deficiency in heat which would be produced by the oxidation and chemical reaction of the converter charge of matte and concentrates when the concentrates were used in the proper quantity to produce the right slag. Accordingly in these experiments quantities of coke varying from 1 to 2.4 per cent. of the weight of the concentrates were used on the different charges.

From the above it will be seen that when concentrates are used the elapsed time of blowing from charge to white metal is greater per ton of copper than when concentrates are not used. This is on the assumption that the grade of the matte is the same in both cases and also that the lost time is the same. The grade of the matte and the lost time for the concentrate charges were observed and recorded, but as no corresponding data for the regular charges were available, it is not necessary to tabulate this information here.

With the assumptions made above we may calculate the extra time consumed in smelting the concentrates as follows:

| | |
|---|---|
| Copper produced from charges smelting concentrates, tons | 49.45 |
| Time blowing per ton of copper in regular charges, min | 37.3 |
| Estimated time for 49.45 tons, min | 1,844 |
| Time blowing concentrate charges, min | 2,762 |
| Excess time for blowing concentrates, min | 918 |
| Excess time for blowing per ton of concentrates, min | 19.3 |

A fair average of the slag skimmed from the charges in which concentrates were smelted is: $SiO_2$ 23.7; $Al_2O_3$ 6.1; FeO 60.3; CaO 0.7 per cent. This is practically the same as the slag produced by a converter running on a regular charge.

The feeding of the concentrates was from a closed tank discharging into the air main. Several mechanical difficulties were encountered which pointed the way to a better mechanical arrangement had the experiments developed into regular practice.

It was observed that the concentrates produced a marked chilling effect on the charge, particularly when the grade of the matte was high. The addition of the small percentages of coke mentioned produced some effect, but the greatest factor in heat production was the grade of the matte. This chilling of the charge produced attendant troubles such as heavy working, fouling of the mouth, etc.

Another difficulty was the mechanical wear of the tuyere holes in the lining. As the tuyeres in one of the bottoms used were badly spaced, that is, two of them opened into the converter near together, the enlarging of the original holes soon brought these two holes together and resulted in a great deal of throwing out of material at the mouth, with consequent fouling of the mouth, and delays to clean away the accumulation. This lost time is included in the time per ton of copper for blowing and of course tends to make it abnormally high.

It was also found that the concentrates cut through the steel tuyere pipes carrying the air from the wind box into the converter, and even cut down into the shell of the converter. A new converter, which was being built at the time these experiments were going on, was made with large tuyere pipes and removable cast bushings to take care of this wear should the process be adopted.

The rate of feeding concentrates varied from 52 to 210 lb. per minute, based on the time of actual feeding, which was intermittent. The average rate based on the total elapsed time from charge to white metal was 34.4 lb., or 0.0172 tons. The total elapsed time of blowing from white metal to copper was 1,199 min. With 2,762 min. from charge to white metal, the elapsed time from charge to copper was 3,961 min. Based on this

time the rate of feeding concentrates was 24 lb., or 0.012 tons per minute.

It should be borne in mind that this was an experimental run with only a small amount of previous trial of the manipulation of the process, and was carried out in a small converter of a class which has since been abandoned. With the present basic-lined converters of a much larger size, there is no doubt but that much better results can be obtained, particularly with practice.

The further trial of this process was dropped, however, as other processes were under consideration for trial, and development.

## Bottom Blast.

The first drawings for the converter plant show both side-blast and bottom-blast converters, but the first installation was side blast. No trial of bottom blast was made in a commercial way until 1897, or 1898.

On July 17, 1897, a drawing was completed of a bottom-blast bottom section to fit the upper part of a regular Class I converter. The tuyeres were of course vertical. Two bottoms were made, one with a single row of 17.75-in. tuyeres spaced on the center line passing through the trunnions, and another with three parallel rows of the same size tuyeres spaced on this center line and a line on each side of and a few inches from it.

The converter with three rows of tuyeres was used several times blowing complete charges, demonstrating that bottom blast could be used commercially. The converter was not punched while blowing, but while it was turned down and with no metal over the tuyeres. The wear of the side lining was observed to be very uniform, much more so than on a side-blast converter. The operation of this converter did not appear at all impossible, but was not as convenient as the side blast and was abandoned.

In February, 1901, another attempt at operating bottom blast was made, which as it developed might be considered somewhat in the light of a trial of basic lining.

A regular Class I converter was fitted with a 9-in. magnesite-brick bottom close to the shell. On top of this 18 in. of regular clay was tamped and then the regular clay (acid) lining continued up the sides. The converter, having been previously dried and heated, was put into service on Feb. 3, 1901, being charged with matte carrying 55.1 per cent. of copper. The clay bottom floated up on the first charge, and this charge and the balance of this campaign were finished on the magnesite-brick bottom. The converter made 22 tons of copper before the side lining was worn out as against 15.9 tons of copper for other regular acid-lined side-blast converters during the same period.

The converter was then relined with clay as on the first campaign and put into service on Feb. 6, 1901. The clay bottom again came up on the first charge and the campaign was finished on the magnesite-brick bot-

tom. On this campaign the converter made 18.8 tons of copper from a matte carrying 48.5 per cent. of copper, as against 14.4 tons of copper per lining from a 48.1 per cent. matte by the other acid-lined side-blast converters.

This is the extent of the trials of bottom blast at this plant.

## Disposal of Converter Slag.

For a considerable time after the starting of the plant the converter slag was poured, while molten, into the reverberatory furnaces. When

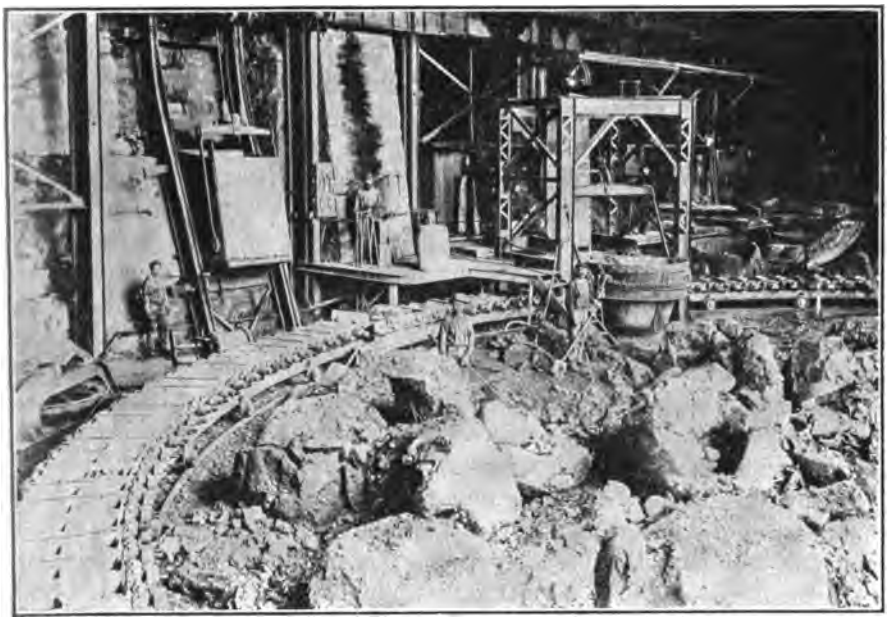


Fig. 10.—Machine for Casting Molten Slag Equipped with Special Pouring Ladle.

the blast furnaces were started it was found that this slag was required in them as a basic flux. Accordingly the slag was poured by the crane from the ladles into large sand beds on the floor. The resulting slabs were about 6 or 7 ft. in diameter by perhaps 18 in. thick at the center, and running out to a thin edge. When these had solidified in the beds they were picked up by the crane by means of hooks, which were driven under the edge, and piled up on the floor, where they were broken up by hand labor. This was, of course, slow, laborious and expensive, and the next change was the use of a drop weight of about 4,000 lb., which was hoisted to the highest point possible by the crane and released by a trigger when a rope was pulled, thus breaking the slab. Later this weight was increased in size, and did better work than the previous one, but a great deal of hard

labor was still required to prepare the slabs for use in the furnaces. Accordingly, in 1901 designs were begun for mechanical casting of the molten slag. The machine finally built was a circular casting machine, consisting essentially of the rim of a horizontal wheel without spokes or center frame work. This wheel was 70 ft. in diameter on the center line of the molds, and was driven like a pulley by a wire cable passing around the outside of the rim and operated by two motors diametrically opposite each other. The rim of the wheel was carried on trucks running on a circular track.

The molds were carried on trunnions turning in bearings and were automatically dumped when the revolution of the whole machine had brought them to the point designed for dumping. After dumping they were automatically turned back to the proper receiving position. The center of this machine was left clear as described above, to provide space for the breaking up of the ladle skulls.

At first no special pouring ladle was provided, it being the intention to have the crane pour from one of the regular ladles directly into the molds.

This machine was started on May 23, 1902. It was found after some operation, that the pouring of slag by the crane was not satisfactory, and a special pouring ladle was installed. Fig. 10 shows the machine thus equipped and also shows the skip which is loaded from the pocket into which the molds automatically dump. This skip hoists the cast slag and dumps it into bins on the blast-furnace floor, from which the blast-furnace charge cars draw it. This machine in its original form has been in constant use since its installation and has given good satisfaction.

Many designs have been considered for mechanically breaking and handling the ladle skulls which can be seen in Fig. 10 in the center of the machine, but none has been adopted. One of the great difficulties has been the large size of the pieces. This would require a large machine, and the tonnage to be handled is comparatively small. In the new plant now being erected, some mechanical means of doing this work will probably be adopted.

## Development of the Different Classes of Converters after Class I.

This subject may be opened with the general statement that all converters which have been used at the plant have been of the upright type, as distinguished from the horizontal or barrel type. We have used the word "class," Class I being the earliest, Class II the next, and so on, to distinguish the different converters, and still not signify any change in the general type.

The first, or Class I, converters were circular in cross-section, 7 ft. 1.5 in. in diameter, acid lined in all except experimental runs of basic, or

neutral (chrome) lined converters, and side blast except in the bottom-blast experiments.

When the question of improving the work of the converters came prominently to the front in 1903, not from any particular occurrence, but from general observation and study of the department, the fundamental thing aimed at was to do more work, that is produce more copper, in a given

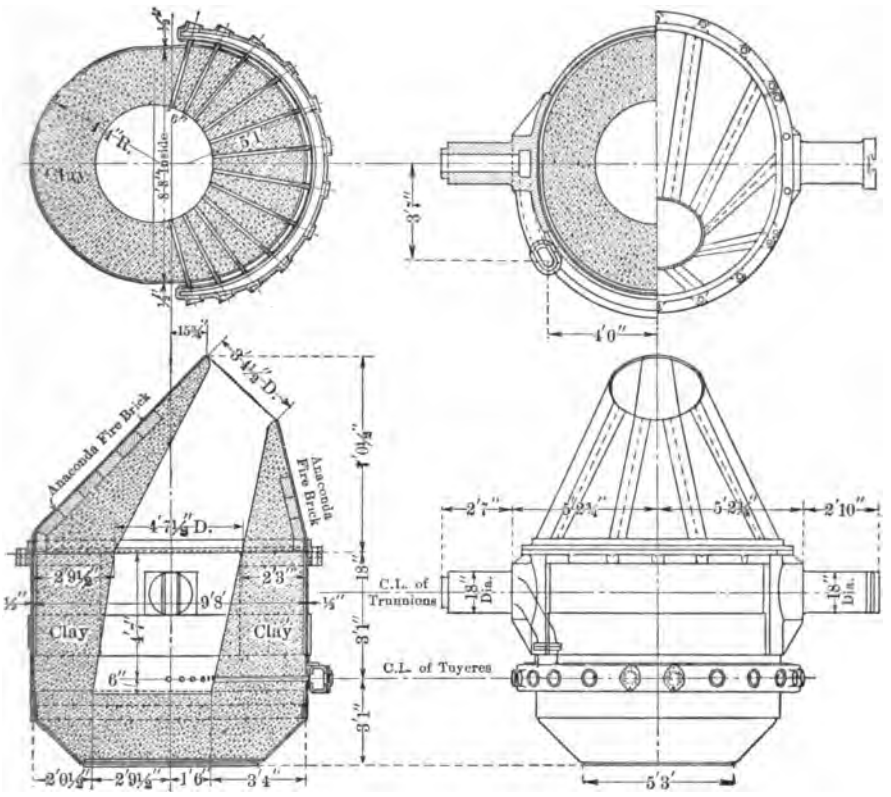



FIG. 11.—CLASS II CONVERTER, CLAY LINING.

length of time and with the same or less labor. It should be borne in mind that at this time acid linings were still in use. The accomplishment of the desired results therefore involved the following:

A converter must be made to take and finish a larger charge; therefore, it must be larger in its inside diameter when newly lined and ready for use.

It must perform the work of oxidation more rapidly, therefore air must be supplied at a faster rate, and consequently either more tuyeres or larger ones must be used.

It must make at least as many, and preferably more, charges on a

lining as the Class I, therefore it must contain more available lining material than the Class I.

It was decided that we would try to make a converter which would produce twice as much copper per charge, produce as many charges on a lining, and do this in the same aggregate time, that is, in one-half the time per ton of copper, as the Class I.

TABLE III.—*Tons Copper Produced per Lining.*

| Month | Class I. | Class II. | |
|---|---|---|---|
| December, 1904 | 15.7 | 22.3 | |
| January, 1905 | 16.8 | 27.8 | |
| February, 1905 | 16.1 | 31.5 | |
| March, 1905 | 16.4 | 31.4 | |
| April, 1905 | 17.8 | 35.1 | |
| May, 1905 | 19.1 | ..... | |
| June, 1905 | 17.2 | 40.0 | |
| July, 1905 | 16.2 | 37.0 | |
| August, 1905 | 16.2 | 29.2 | |
| September, 1905 | 18.2 | 39.5 | |
| October, 1905 | 18.6 | 39.7 | |
| November, 1905 | 18.2 | 48.9 | |
| December, 1905 | 19.2 | 65.0 | only 1 lining |
| January, 1906 | 17.2 | 44.9 | |
| February, 1906 | 18.0 | 56.7 | |
| March, 1906 | 19.3 | 51.8 | |
| April, 1906 | 16.1 | 46.3 | |
| May, 1906 | 18.1 | 47.1 | |
| June, 1906 | 16.7 | 48.0 | |
| July, 1906 | 16.3 | 37.7 | |
| August, 1906 | 18.1 | 42.7 | |

The first study was the distribution of the lining in the converter. It had been observed that the lining always wore away more rapidly at the tuyeres than anywhere else. The next most rapid wear seemed to be directly opposite the tuyeres, and led to the conclusion that the wear at the tuyeres was both mechanical and chemical, while that at the opposite side was largely mechanical, due to the circulation of the molten bath. The least wear appeared to be at the sides. Therefore an elliptical converter was designed with the long axis at right angles to the trunnions, that is, passing through the center of the wind box. The long diameter was 9 ft. 8 in. and the short diameter 8 ft. 8 in. The converter was not a true ellipse, but was made up of two half circles 8 ft. 8 in. in diameter, the perimeter between these being straight.

The converter is shown in Fig. 11. These figures show the shape and dimensions of the lining.

The mechanical arrangement of trunnions turning in fixed bearings was retained, as it was not considered that the weight of the converter had passed beyond the safe point for this construction. The old hydraulic tilting was, however, abandoned, and electric power substituted.

There were twelve 1⅛-in. tuyeres instead of eight ⅞-in. tuyeres, as in Class I. The height of the converter was practically the same as Class I. This was because it was not believed a greater height would be of any advantage. The converter was put into service with acid lining on Nov. 29, 1904, and its first run was very satisfactory.

The first Class II converter was operated regularly and studied as to its economy and performance as compared with the Class I converter for about 15 months before instructions were given on Apr. 5, 1906, to install the second Class II stall. This second stall was put into service on July 28, 1906, at which time we had four bowls or bottoms for the two stalls.

During the trial period preceding the starting of the second stall the work of the Class II showed rather uniform improvement as can be seen from Table III, which contains, for comparison, the performance of Class I converters during the same time.

From this it will be seen that we had succeeded in our plan to produce twice as much copper per lining as in the Class I.

As a comparison of the speed of the two classes the following can be noted:

| | Minutes Blowing Per Ton Copper | |
|---|---|---|
| Month, 1906 | Class I. | Class II. |
| January | 40.7 | 28.9 |
| March | 40.1 | 24.5 |
| July | 45.8 | 32.7 |
| August | 46.9 | 29.8 |

Of course the grade of the matte affects the time required per ton of copper for blowing, but in the above comparison the two classes for any particular month are comparable as they both worked on the matte from the same sources at the same times. It will be observed that while we had not absolutely succeeded in producing copper in one-half the time per ton as in Class I, we had approached fairly near to it.

This marked the determination to abandon the Class I converters, the plan being to run each one until such time as it came to require an extensive repair, and then discard it in favor of a larger converter.

Although the abandonment of the Class I was definitely determined, the exact converter with which to replace it was not decided. A study of the work of Class II showed possibilities of improving it. For one thing, we now found that the first points at which the lining wore through were on the sides under the trunnions. We also felt that with more tuyeres we could work faster. These two things led to the design of the Class III. This is shown in Fig. 12.

It will be observed that this is an elliptical converter with the long

diameter on the line of the trunnions instead of at right angles to them as in the Class II. The long diameter is 12 ft. and the short diameter 10 ft., outside measurement. The height over all is 13 ft. 8.5 in., which is practically 1 ft. less than the Class II. There were 15 1⅛-in. tuyeres as against 12 1⅛-in. tuyeres in the Class II. It will be observed that the shape of the inside of the lining is quite different from that of the Class

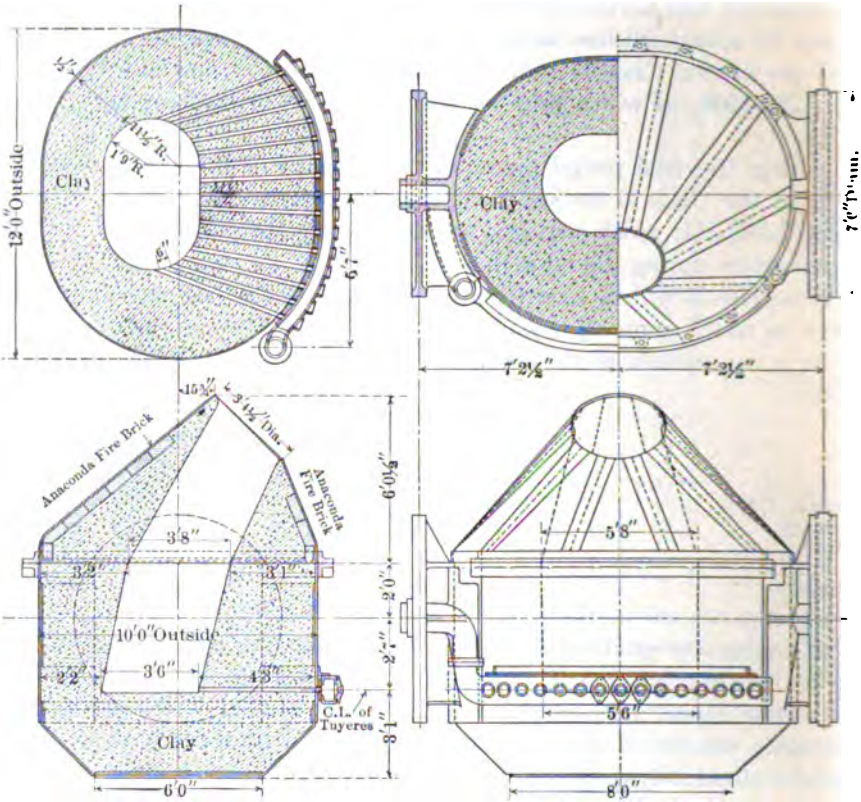



FIG. 12.—CLASS III CONVERTER, CLAY LINING.

II. This was the subject of much discussion before the adoption of the converter, as it was feared that the straight sides on the front and back would not stand up well in service. However, it was determined to try it and accordingly a new stall for this class was put in. The converter was built, and started on its first campaign on July 18, 1907, with the usual acid lining of mixed ore and B. & M.* sand rock.

The first linings were put into the Class III converters by hand, as the converter would not go on the tamping machine. These runs did not show the expected increase in production per lining. It was not until March, 1908, that the converter was started on machine-tamped linings,

it having required this time to find opportunity to make the necessary changes in the tamping machine. Two hand-tamped and five machine-tamped linings were run up to the time that the teeth of the tilting gear failed on Mar. 30, 1908.

The record of the five machine-tamped linings was: Copper per lining, 70.5 tons; time blowing per ton of copper, 26.3.

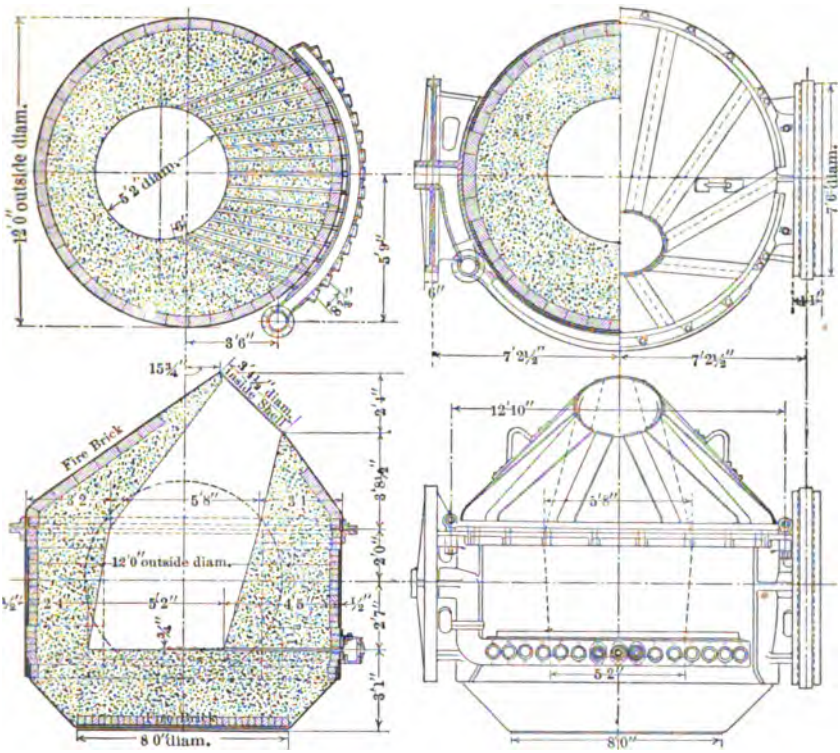



FIG. 13.—CLASS IV CONVERTER, CLAY LINING.

This was a great improvement over the work of Class II with respect to the copper produced per lining, and some improvement as regards the speed of blowing, but it is not to be considered as a fair average of the work which could be expected over a long period, as we had learned by experience with the Class II that extraordinary tonnages per lining were sometimes recorded for individual runs, due sometimes to particularly careful manipulation and sometimes to the use of a large quantity of electrolytic anode scrap which is melted up in undue proportion to the charge. Over a long period of averages this anode scrap is small in its effect on tonnages.

Study of the linings and operation of the Class III showed that the

lining wore out first at the tuyeres as in the Class I, and the next step was to retain the thickness of the Class III lining at the trunnions and thicken it up at the tuyeres and opposite them. This brought us to the Class IV, which is circular in horizontal section 12 ft. in diameter, 13 ft. 8.5 in. high (the same as Class III) with 15 1⅛-in. tuyeres (the same as Class III). This is shown in Fig. 13. The particular diameter of 12 ft. was used, as this was the same as the long diameter of the Class III and it would therefore fit in the same stall.

The date of the finished drawing for this converter is Mar. 24, 1909, and the first converter was started on Feb. 4, 1910. This class of converter was run and studied for some time with the result that on Oct. 8, 1910, instructions were given to abandon all converters except Classes III and IV as fast as they came to serious repair, it being considered that in this way the equipment would perform all converting work until basic linings which had been discussed should be demonstrated.

The work of seven of the early linings in Class IV converter was as follows:

| | |
|---|---|
| Tons copper produced per lining | 54.270 |
| Tons copper produced per charge | 7.448 |
| Minutes blowing on finish per ton of copper produced | 8.41 |

This brings us to the adoption of the basic-lined converter which will be discussed under a separate heading.

The latest development is the Class V converter but as this has been from the beginning strictly basic lined it will be discussed under that heading.

The development so far described has resulted as applied to acid-lining practice in: Increased size of individual charges; increased tonnage from one lining; decreased time blowing per ton of copper produced; decreased labor per ton of copper.

It is rather difficult to secure data which will be comparable on the time blowing for the different classes, as the grade of matte affects this item materially. In our recent studies we have adopted, as the proper comparison of the speed of the converters, the minutes blowing on the finish from white metal to copper per ton of copper produced. We call this "Minutes per ton of copper on finish." We have adopted this because no matter what has happened in converting from matte to white metal, the work to be performed per ton of copper produced from white metal to copper is the same for all charges.

We can make the general statement that from Class I to Class IV, inclusive, the speed has increased, but in the discussion of the basic-lined converter we will give some comparisons of different conditions in recent investigations.

As to labor per ton of copper produced, we can say that the Class I converters first had crews of four men per converter. This was afterwards cut down to three. In the later types the crew has been two men per converter per shift. As the tonnage produced per crew shift has increased in absolute weight, it necessarily follows that the labor per ton has decreased considerably. The change has been made from 12-hr. shifts to 8-hr. shifts, and the wages per man increased, but nevertheless the cost per ton of copper for labor has decreased.

## Basic Lining.

An early attempt was made to use a basic lining and a little later a neutral lining. Early in 1897, the exact date cannot be located, but probably in April, one of the side-blast converters then in use, which we now call Class I, was lined with 9-in. magnesite brick. The converter must have been equipped with eight ¾-in. tuyeres, and was undoubtedly in all respects, except the lining, the same as the converter shown in Figs. 1, 2, and 3. The tuyeres were pipes, expanded into the wind box and shell, and extending into and through the brick lining. This converter when dried and ready was charged with molten matte, turned into the stack and blown. As soon as the converter was blowing into the stack, second-class ore which had been placed on a platform behind the converter, and at a convenient height, was shoveled into the mouth. According to the recollection of A. R. McKenzie, the converter foreman at the time, and of which he made a record on Mar. 24, 1905, as much as 6 tons of ore was sometimes smelted per charge, and he further states that something like 164 tons of copper was produced by the converter before the lining was worn out.

The date is definitely fixed as not later than the early part of April, 1897, by the following note from Frank Klepetko, Superintendent at the time, to John J. Case, Assistant Superintendent, dated Apr. 12, 1897. "Have you had any of the ordinary slag from basic lined converter assayed, and the silica and iron and copper determined? This should be done regularly on each charge in order for you to get data that will be of any value." This original note is now in the company files. Unfortunately a recent fire in the laboratory destroyed our early records and these slag assays are not now available.

In February, 1901, while running a bottom-blast converter, a magnesite-brick bottom was used as a backing for the clay bottom. This clay bottom came up while operating the converter and therefore, in a way, the run was partially a basic-lining run. This has been described under the heading of "Bottom Blast".

In April, 1906, a chrome-brick lining was tried, in one of the Class I side-blast converters such as is shown in Figs. 1, 2, and 3. A 9-in. chrome-

brick lining, backed up between the brick and the converter shell, with well-tamped clay, such as was used for acid linings, was put in with the tuyere pipes extending from the wind box into and through the brick lining.

The converter was fired for several days and when thoroughly dried was put into commission at 2 P. M., Apr. 2, 1906. The first charge was reverberatory matte assaying 36 per cent. copper. When blowing had commenced, ore from the company's Mountain View mine, Butte, was dumped into the converter. Nine charges were blown before the lining was destroyed, converting 219,225 lb. of matte and smelting or using as flux 113,309 lb. of ore. The matte probably averaged about 40 per cent. copper.

In the early part of 1910 the question of basic lining came prominently to the front, being at that time advocated by the Pierce-Smith people after their demonstrations at Baltimore and Garfield.

It was finally determined to try one of our Class IV converters without change as to shape and mechanical arrangements, with a basic lining of magnesite brick.

The converter, which was known as bottom B, was lined as shown in Fig. 14, and the first matte charged into it at 3.50 P. M., Mar. 9, 1911. It had 15 tuyeres of 1.25-in. extra strong tubing having an internal cross-section of 1.27 sq. in. The converter started off well and continued to run well, so that on Apr. 29, 1911, the second converter was put into service. This was an exact duplicate of the first converter and was known as Class IV, bottom C. The third converter, a duplicate of the first two, and known as Class IV, bottom D, was put into service on May 7, 1911.

In the meantime the question of the number and size of tuyeres had been discussed with the result that the fourth converter known as Class IV, bottom A, was a duplicate of the first three except that it had 26 tuyeres, each with an internal cross-sectional area of 1.27 sq. in. This was put into service June 22, 1911.

We then considered that we would need a spare converter and therefore lined a Class III bottom, putting in 24 tuyeres, each with 1.27 sq. in. internal cross-section.

These are all the bottoms which have been lined, except the Class V, 20-ft. converter. Some of them have finished the first campaign, been repaired and started on the second. Some were shut down before completing a campaign and changes made for experimental purposes, and some are still running on the first campaign.

A brief resumé of each lining is as follows:

(A tabulation of these figures with some calculations from them will be found in Table IV.)

*Class IV, Bottom B.*—Started with 15 tuyeres each with 1.27 sq. in. internal cross-section; first charge, Mar. 9, 1911; ran until June 3, 1912, an elapsed time of 451 days; made 1,026 charges, producing 12,216.1 tons of copper; the calculated matte treated is 35,700 tons; ore smelted,

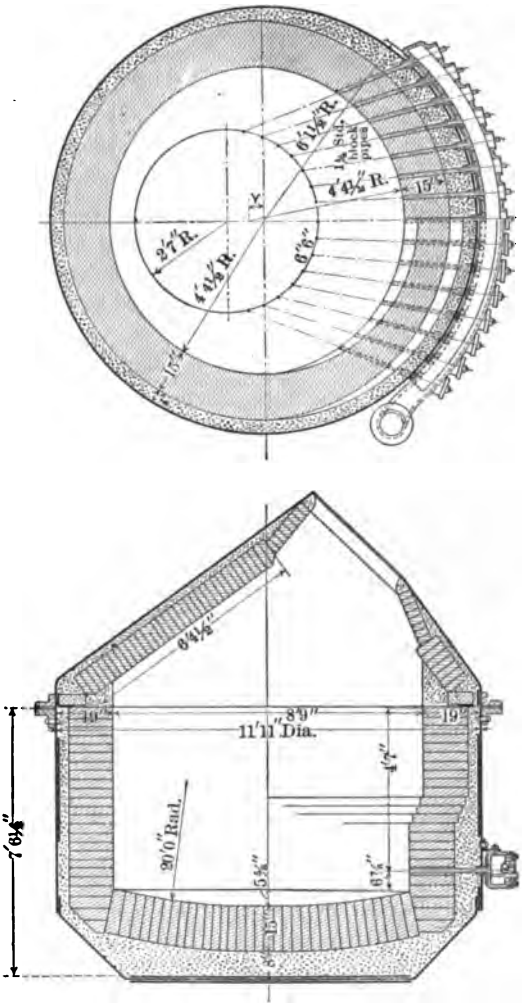



FIG. 14.—CONVERTER B, BASIC LINING.

8,398.8 tons; time operating, 7,509 hr. 45 min.; shut down to repair lining.

*Class IV, Bottom C.*—Started with 15 tuyeres, each having 1.27 sq. in. internal cross-section; first charge, Apr. 29, 1911; ran until Jan. 4, 1912,

an elapsed time of 251 days; made 580 charges, producing 7,479.1 tons of copper; the calculated matte treated is 23,500 tons; ore smelted, 5,222.5 tons; time operating, 4,149 hr. 40 min. The converter needed only a slight repair when it was shut down but was shut down in order to put in 34 tuyeres for experimental purposes. This converter was then fitted with 34 cast tuyeres, as described under the heading "Cast Copper Tuyeres."

It was put into service with these cast tuyeres Mar. 17, 1912, and run on them experimentally until Mar. 27, 1912, when the tuyeres burned out. These cast tuyeres were then replaced with 34 tuyeres, each having an internal cross-section of 1.27 sq. in. The converter was run on these tuyeres for a few days, during which it threw out its charge badly and therefore instructions were given to plug the two end tuyeres on each end, leaving 26 tuyeres in action. Therefore practically all of this second run was made on 26 1.25-in. tuyeres, and this will be considered as its tuyere equipment.

Its record on this run is as follows:—Charged Mar. 17, 1912; up to Apr. 13, 1913, elapsed time 392 days; made 445 charges producing 5,643 tons copper; the calculated matte treated is 16,900 tons; ore smelted, 3,176 tons; time operating 2,960 hr. 10 min.; still in service, after some slight repairs to lining.

*Class IV, Bottom D.*—Started with 15 tuyeres, each having 1.27 sq. in. internal cross-section; first charge, May 7, 1911; ran until Dec. 22, 1912, when it was shut down to change the tuyeres for experimental purposes; elapsed time up to Dec. 22, 1912, 596 days; made 1,014 charges, producing 13,414 tons copper; calculated matte treated, 40,250 tons; ore smelted, 10,157 tons; time operating, 7,663 hr. 55 min.

The converter was shut down to try an experiment with large tuyeres, and was equipped with 15 tuyeres of 3-in. boiler tube with an approximate inside diameter of 2.75 in. and an area of 6.079 sq. in. each.

The converter was started on Dec. 28, 1912, but only seven of the tuyeres were used, the others being plugged with clay. The converter ran with these tuyeres until Apr. 2, 1913, when it was decided that the tuyeres were too large. The elapsed time was 96 days, in which 85 charges were made, producing 968.2 tons of copper. The estimated matte treated is 2,900 tons; ore used, 816 tons; time operating, 601 hr. 45 min.

When the converter was shut down it was equipped with 15 tuyeres of 2.5-in. boiler tubes with an internal area of 4.09 sq. in. each, and started again on May 12, 1913, and is still in operation.

During this time of May 12 to June 1, it has made 28 charges, producing 369.3 tons of copper; estimated matte treated, 1,110 tons; ore used, 294 tons; time operating, 184 hr.

All of these three runs can really be considered as one campaign, as in

no case was the converter shut down from necessity to repair. The shutdowns were all for purposes of changing the tuyeres for experimental purposes.

Considering these as one campaign, the record is:—Started May 7, 1911; running at present, the elapsed time to June 1, 1913, being 756 days, although a few days were lost in making tuyere changes; made 1,127 charges producing 14,751.5 tons copper; estimated matte treated, 44,260 tons; ore used, 11,267 tons; time operating, 8,449 hr. 40 min.

*Class IV, Bottom A.*—This was the first trial of an increased number of tuyeres. The converter was equipped when it started with 26 tuyeres, each having 1.27 sq. in. internal cross-section. It was started June 22, 1911, and is still operating, the elapsed time up to June 1, 1913, being 722 days; made 1,170 charges, producing 16,407.3 tons of copper; estimated matte treated, 49,200 tons; ore used, 10.118 tons; time operating, 8,198 hr. 40 min.

*Class III, Bottom 3.*—This was lined up as a spare bottom and therefore has not been operated continuously, which should be borne in mind in considering its record. It was first equipped with 24 tuyeres, each having 1.27 sq. in. internal cross-section. It was started Nov. 16, 1911, and continued with this tuyere equipment until Dec. 25, 1912, when it was shut down to change tuyeres for experimental purposes. The elapsed time to Dec. 25, 1912, is 406 days, in which 640 charges were made, producing 8,990.7 tons of copper; estimated matte treated 27,000 tons; ore used, 5,171 tons; time operating, 4,523 hr. 50 min.

When the converter was shut down it was for the purpose of equipping it with 11 tuyeres of 2.5-in. boiler tube, each tuyere with an internal cross-section of 4.09 sq. in. It was started Mar. 28, 1913, and is still running, the elapsed time to June 1, 1913, being 70 days. In this time 33 charges were made, producing 404.2 tons of copper; estimated matte treated, 1,210 tons; ore used, 389 tons; time operating, 251 hr. 50 min.

Here again the two runs can really be considered as one campaign, and if they are combined the record is:—Started Nov. 16, 1911, still running, the elapsed time up to June 1, 1913, being 476 days; made 676 charges producing 9,394.9 tons copper; estimated matte treated, 28,210 tons; ore used, 5,560 tons; time operating, 4,775 hr. 40 min.

### *Class V, or 20-ft. Converter.*

All our investigations and experiments pointed to the probable success of a larger converter than the Class IV or 12 ft. After some discussion, it was determined to build one of about three times the capacity of the Class IV and 20 ft. was decided upon as the diameter, still adhering to the upright type.

TABLE IV.—*Record of Basic-Lined Converters.*

| Converter | | Campaign | | Elapsed Days | Tons Cu | Charges | Tons Cu Per Charge | Total Time Operating | | Tons Cu Per Converter-Hour | Tuyeres | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Class | Bottom | From | To | | | | | Hr. | Min. | | No. | Approx. Internal Diam. In. |
| IV | B | Mar. 9, 1911 | June 3, 1912 | 451 | 12,216.1 | 1,026 | 11.9 | 7,509 | 45 | 1.63 | 15 | 1.25 |
| IV | C | Apr. 29, 1911 | Jan. 4, 1912 | 251 | 7,479.1 | 580 | 12.9 | 4,149 | 40 | 1.80 | 15 | 1.25 |
| *a*IV | D | May 7, 1911 | Dec. 22, 1912 | 596 | 13,414 | 1,014 | 13.2 | 7,663 | 55 | 1.75 | 15 | 1.25 |
| *b*IV | A | June 22, 1911 | June 1, 1913 | 710 | 16,407.3 | 1,170 | 14.2 | 8,198 | 40 | 2.00 | 26 | 1.25 |
| *c*III | 3 | Nov. 16, 1911 | Dec. 25, 1912 | 406 | 8,990.7 | 640 | 14.0 | 4,523 | 50 | 1.99 | 24 | 1.25 |
| IV | C | Mar. 17, 1912 | Apr. 13, 1913 | 392 | 5,643 | 445 | 12.7 | 2,960 | 10 | 1.91 | 34 | 1.25 |
| V | A | Aug. 3, 1912 | Dec. 17, 1912 | 137 | 2,835 | 88 1/6 | 32.4 | 656 | 45 | 4.31 | 62 | 1.75 |
| *b*IV | B | Sept. 12, 1912 | June 1, 1913 | ..... | 3,904.3 | 307 | 12.7 | 2,180 | 20 | 1.79 | 26 | 1.375 |
| *a*IV | D | Dec. 28, 1912 | Apr. 2, 1913 | 96 | 968.2 | 85 | 11.4 | 601 | 45 | 1.69 | 7 | 2.75 |
| *b*V | A | Jan. 20, 1913 | June 1, 1913 | 132 | 6,223.2 | 169.9 | 36.6 | 1,306 | 0 | 4.77 | 26 | 2.25 |
| *c*III | 3 | Mar. 28, 1913 | June 1, 1913*b* | 70 | 404.2 | 33 | 12.3 | 251 | 50 | 1.60 | 11 | 2.25 |
| *a*IV | D | May 12, 1913 | June 1, 1913 | 20 | 369.3 | 28 | 13.2 | 184 | 0 | 2.01 | 15 | 2.25 |

*a* These three runs are really parts of the same campaign.
*b* Still running on this campaign.
*c* These two runs are really parts of the same campaign. The converter was shut down on Dec. 25, 1912, to change the tuyeres for experimental purposes.

The design involved many mechanical problems, but it is probably sufficient to say that the converter as built has shown no structural weakness in its operation to date of 269 days.

One of the important discussions was as to the number and size of tuyeres to be put in it. It was finally decided to equip it with 62 tuyeres.

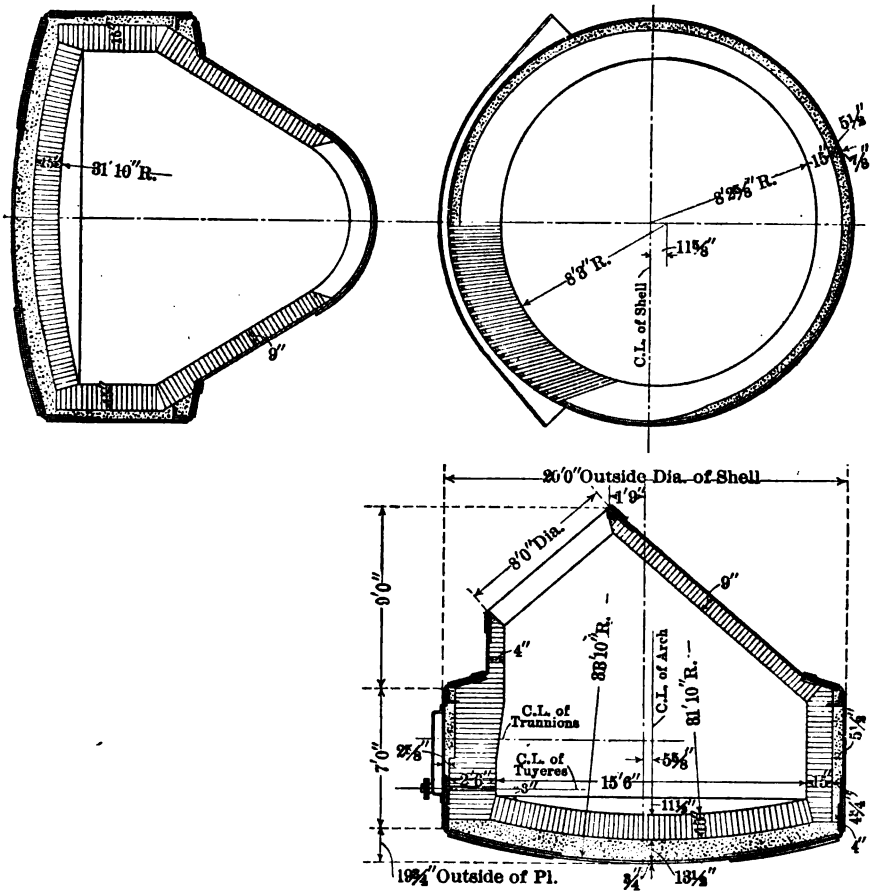



FIG. 15.—20-FT. CONVERTER, BRICK LINING.

of 2-in. boiler tube, each having an inside diameter of approximately 1.75 in. and a transverse internal area of 2.573 sq. in. The reason for this decision is related under the heading "Number and Size of Tuyeres."

The shape and principal dimensions both inside and outside are shown in Figs. 15, 16 and 17.

After running a time it was decided to change the tuyeres to a less number of a greater diameter, which will show in the following record:

*Class V, Bottom A.*—Started Aug. 3, 1912; run until Dec. 17, 1912, with 62 tuyeres, each with an internal cross-section of 2.573 sq. in.; elapsed time 137 days; made 88 charges, producing 2,835 tons copper; estimated matter treated 8,500 tons; ore used 1,368 tons; time operating 656 hr. 45 min.

On Dec. 17, 1912, it was shut down to replace the 62 1.75-in. tuyeres with 26 2.25-in. tuyeres, made from 2.5-in. boiler tube, and having an internal cross-section of 4.09 sq. in. each. Started with 26 tuyeres Jan. 30, 1913; still operating, the elapsed time to June 1, 1913, being 132

FIG. 16.—CLASS V CONVERTER. SIDE VIEW.

days; made 170 charges, producing 6,223.2 tons copper; estimated matte treated, 18,700 tons; ore used, 3,982 tons; time operating, 1,306 hr.

For the purpose of bringing the above record of the different converters together in more concise form Table IV is given, and from this tabulation another will be made bearing upon the determination of the best size and number of tuyeres.

The total time operating is the summation of the total elapsed time from charge to finish of all the individual charges and therefore includes all the usual delays of daily operation.

The copper per converter-hour is the total copper produced divided by the operating time.

It is quickly seen that the Class V converter greatly outstrips all others

as regards speed of operation, that is copper per converter-hour, and also as to copper per charge.

The comparison of the operating labor on the 12-ft., or Class IV, and the 20-ft., or Class V, is as follows, each of course operating three shifts per 24 hr.:

| Labor | Rate Per Shift | 12-Ft. Converter | | | | 20-Ft. Converter | | | |
|---|---|---|---|---|---|---|---|---|---|
| | | Per Shift | | Per Day | | Per Shift | | Per Day | |
| | | No. | Cost | No. | Cost | No. | Cost | No. | Cost |
| Skimmer.... | $4.25 | 1 | $4.25 | 3 | $12.75 | 1 | $4.25 | 3 | $12.75 |
| Helper..... | 3.25 | 1 | 3.25 | 3 | 9.75 | 2 | 6.50 | 6 | 29.25 |
| Total..... | ..... | 2 | $7.50 | 6 | $22.50 | 3 | $10.75 | 9 | $42.00 |

When it is considered that the 20-ft. converter produces three times the copper per day that the 12-ft. converter does, the comparison becomes: Labor cost one 20-ft. converter, $42; labor cost three 12-ft. converters, $67.50.

## Number and Size of Tuyeres.

In order to segregate this important subject a rearrangement of parts of the tabulation "Performance of Basic-Lined Converters" is given in the accompanying table in which the performance of the different converters is listed in approximately the order of increasing total tuyere area.

Table V.—*Converter Performance with Relation to Number and Size of Tuyeres.*

| Converter | | No. | Tuyeres | | Total Area Sq. In. | Tons Copper Produced | | |
|---|---|---|---|---|---|---|---|---|
| Class | Bottom | | Approx. Diam. In. | Internal Area Sq. In. | | Per Charge | Per Converter-Hour | Total for Campaign |
| IV | B | 15 | 1.25 | 1.27 | 19.05 | 11.9 | 1.63 | 12,216.1 |
| IV | C | 15 | 1.25 | 1.27 | 19.05 | 12.9 | 1.80 | 7,479.1 |
| IV | D | 15 | 1.25 | 1.27 | 19.05 | 13.2 | 1.75 | 13,414.0 |
| III | 3 | 24 | 1.25 | 1.27 | 30.48 | 14.0 | 1.99 | 8,990.7 |
| IV | A | 26 | 1.25 | 1.27 | 33.02 | 14.2 | 2.00 | 16,407.3 |
| IV | C | 26 | 1.25 | 1.27 | 33.02 | 12.7 | 1.91 | 5,643.0 |
| IV | B | 26 | 1.375 | 1.496 | 38.896 | 12.7 | 1.79 | 3,904.3 |
| III | 3 | 11 | 2.25 | 4.09 | 44.99 | 12.3 | 1.60 | 404.2 |
| IV | D | 7 | 2.75 | 6.079 | 42.553 | 11.4 | 1.69 | 968.2 |
| IV | D | 15 | 2.25 | 4.09 | 61.35 | 13.2 | 2.01 | 369.2 |
| V | A | 62 | 1.75 | 2.537 | 157.294 | 32.4 | 4.31 | 2,835 |
| V | A | 26 | 2.25 | 4.09 | 106.34 | 36.6 | 4.77 | 6,223.2 |

An inspection of the tabulation shows that the first increase from 15 to 24 tuyeres of the same size resulted in faster work of the converter. The increase to 26 tuyeres showed practically the same work as with 24. The 26 1.375-in. tuyeres show a little falling off which is not explained.

The Class IV, bottom B, with 26 1.375-in. tuyeres shows a little falling off in its speed as compared with converters with 26 1.25-in. tuyeres and having therefore a little less total tuyere area. This is because it

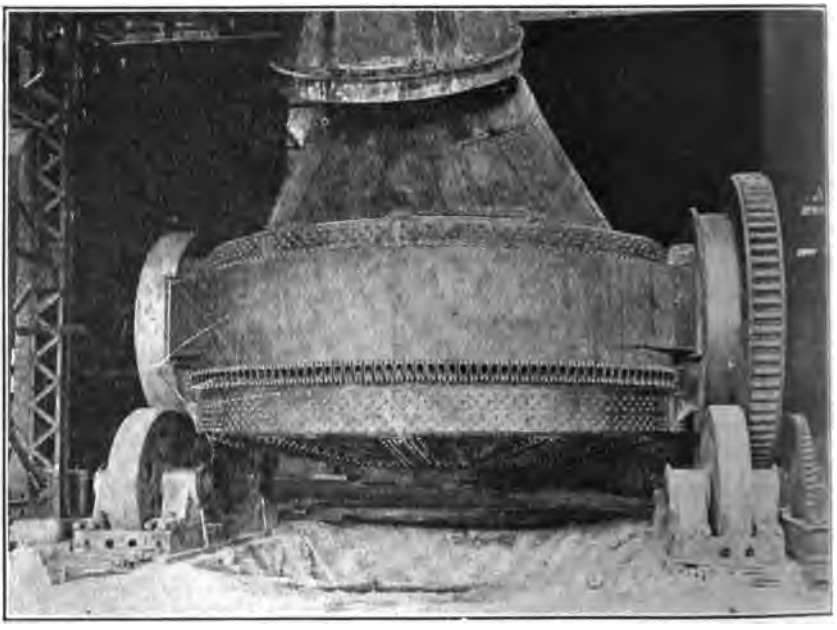


FIG. 17.—CLASS V CONVERTER. FRONT VIEW.

was operated a large portion of the time in a stall having many bends in its air-supply pipes, resulting in restriction of the air supply. Any converter used in the same stall has shown this same tendency as compared with the same converter in other stalls.

The Class III converter with 11 2.25-in. tuyeres shows a falling off in its speed, but this is explained by the fact that it was used intermittently and in the stall with many bends in the air pipe, as described above for Class IV, bottom B.

When we come to the bottom D with seven 2.75-in. tuyeres (3-in. boiler tube) we find a falling off. Observations on this converter showed a great tendency for it to throw out its charge, due evidently to the concentration of air in large volume in a few places.

This same tendency was shown by bottom C with 34 1.25-in. cast

tuyeres, and was probably due to the concentration of the air due to close spacing of the tuyeres.

When we come to bottom D with 15 2.25-in. tuyeres we find work equally as good as the bottom A with 26 1.25-in. tuyeres and the fewer tuyeres of larger size are much preferable for construction reasons, for the strength of the lining, and for the labor of punching.

The Class V converter shows the same thing, the 26 2.25-in. tuyeres showing even better work than the 62 1.75-in. tuyeres, and they are far preferable from all points of view.

Another point of superiority of a few large tuyeres, within the limits of working, over many small ones, is in the decreased resistance to the air and consequently a lower necessary blast pressure. We have found this to be true from practical observation as well as from theoretical reasoning, but have not yet carried our experiments far enough to include the results in this paper.

In order for the increase in tuyere area to be truly of the benefit the preceding figures indicate, it must be shown that the air when passing through is used economically. That it is used at as high efficiency at large volume as when used at less volumes per minute will be shown in detail under "Efficiency of Air."

## Cast-Copper Tuyeres.

In the history of Class IV, bottom B, basic-lined converter, mention was made of cast-copper tuyeres. The following discusses this arrangement a little more in detail.

There is some difficulty in fitting the magnesite brick of the lining around the circular tuyere pipes, and where tuyeres are closely spaced the brick work between them is so small that it is apt not to stay in place well.

With the idea of simplifying the construction and also of allowing the close spacing of tuyeres a Class IV magnesite-lined converter was equipped with 34 tuyeres 1.27-in. inside diameter. Each tuyere was a cast-copper block of such a shape and dimension that it would lie flat on the brick work, which was built up to just below the tuyere line, and fill the space horizontally from half way between itself and the tuyere on the right, and itself and half way between the tuyere on the left, so that when all tuyere blocks were in place they formed a solid block with a tuyere opening through each one. They were made by laying a copper pipe with an inside diameter equal to the proposed inside tuyere diameter, in a dry sand mold and casting the copper around it allowing the pipe to extend out of the end of the block so as to be expanded into the converter shell like any ordinary tuyere pipe.

The converter was tried over the period Mar. 15 to 27, 1912, with 29 different charges, some of which were blown only from white metal to copper, some from matte to white metal, and some were not finished to either.

Without going into details it can be said that it was found that the tuyeres would stand fairly well while blowing from matte to white metal, but that they melted away rapidly when blowing from white metal to copper. The converter threw out badly, due possibly to the concentration of air in large volumes at particular places because of the ring being melted back.

It was thought when the plan was first discussed that the air blast passing through the blocks would keep them sufficiently cool to prevent melting, but it evidently did not. The plan was abandoned as not holding out sufficient promise of success to warrant continued experiments.

## Efficiency of Air.

The speed with which a converter will work, that is, perform oxidation, is of course dependent upon the rapidity with which the chemical reactions will take place and the rate at which air, or, strictly speaking, oxygen, is supplied. With proper distribution of the air through the charge the speed of the reactions in the converter is undoubtedly beyond any practical rate of supplying the air. It is therefore of interest and of practical importance to know how fast air can be supplied and still be efficiently used, and how the air must be distributed.

Our first experiment along this line was on the distribution of the air. A light tin tank about 20-in. in diameter and 2 ft. 6 in. high, with a glass bottom, was made. Two wind boxes were made independent of the tank so that they could be attached to an air main and placed into the tank in the same relative position as a wind box on a converter.

One of these wind boxes had eight holes, each ¼ in. in diameter, and the other had 32 holes ⅛ in. in diameter. Thus the total area of the holes was the same in both cases.

When the tank was partly filled with water and the eight-hole wind box tried, it was found that the air penetrated the water only a few inches, rising immediately to the surface in comparatively large bubbles. When the 32-hole wind box was tried the air acted in the same way, except that the bubbles coming to the surface were much smaller and better disseminated through the liquid. Incidentally it may be mentioned that the circulation of the charge as represented by the water was upward from the tuyeres, the surface then flowing away from the tuyeres and toward the front, thence downward toward the bottom, thence across the bottom toward the tuyeres. There was not sufficient penetration of the blast into the charge along the bottom to set up a circulation in the

direction opposite to that described as had been thought probable. After several observations had been made, all showing the same results, it was decided that a large number of small tuyeres would probably result in more efficient use of the air than fewer large tuyeres.

These experiments were conducted to help on the decision as to the number of tuyeres which should be put into the Class V or 20-ft. converter which was then being built, and as a result of them, it was decided to place the tuyeres as close together as construction details would allow, thus providing as much distribution of air as possible. They were made as large as it was felt would work to advantage, so as to make the converter work as fast as possible. Consideration was given to the probable increased resistance to the passage of the air through many small tuyeres as compared with few large ones, but air distribution was given the preference, and the first Class V converter was equipped with 62 tuyeres of 2-in. boiler tube with an approximate inside diameter of 1.75 in. As described in the discussion of the Class V converter, these were afterwards changed to 26 tuyeres of 2.5-in. boiler tube with an approximate internal diameter of 2.25 in.

After this converter was in operation it was desired to know whether the large volume of air put into the converter was efficiently used and therefore experiments were outlined and carried out for this purpose.

The method adopted for the determination of the efficiency of the air was a chemical one based on the fact that the nitrogen entering the tuyeres passed out of the mouth of the converter unchanged chemically. Therefore, if the ratio between the oxygen and nitrogen in the air, and the same ratio in the outgoing gases be known, the nitrogen being a fixed quantity, the percentage of the entering oxygen which is used, or the efficiency as we call it, can be calculated from the formula:

$$\frac{\frac{O^1}{N^1} - \frac{O}{N}}{\frac{O^1}{N^1}} \times 100 = \text{per cent. efficiency.}$$

Where $O^1$ = oxygen in the air; $N^1$ = nitrogen in the air; $O$ = oxygen in escaping gases; $N$ = nitrogen in escaping gases.

This necessitates the sampling of the gases actually issuing from the mouth of the converter before any admixture of atmospheric air with them. This was done by putting the sampling pipe into the mouth of the converter.

The first samples were taken through an iron pipe the inside of which was washed with milk of lime to prevent oxidation of the iron and consequent vitiation of the sample. It did not prove satisfactory as no oxygen could be found in any of the samples of escaping gases, or in other words, the samples always showed 100 per cent. efficiency. It did not

TABLE VI.—*Efficiency of Air in Converting.*

| Date 1912 | Converter | | | Time | Sample No. | Tuyeres | | Cu. Ft. Air Per Min. Stand. Conditions | Volumetric Analysis of Gases | | | Per Cent. Efficiency of Air | Remarks |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Class | Bottom | Diameter Ft. | | | No. | App. Inter. Diameter In. | | Acid Gas | O | N | | |
| Aug. 14 | V | A | 20 | 10.00 a.m. | 1 | 62 | 1¾ | 22,400 | 28.9 | 2.7 | 68.4 | 85.2 | Sample taken after second doubling of charge. Strong $SO_2$ odor. |
| | | | | 11.10 a.m. | 2 | 62 | 1¾ | 24,800 | 10.6 | 4.8 | 84.6 | 78.3 | After charging 1 boat of ore and 1 pot of matte. |
| Aug. 15 | V | A | 20 | 11.45 a.m. | 1 | 62 | 1¾ | 18,000 | 13.6 | 1.4 | 85.0 | 94.8 | Converter charged at 8 a.m. No punching. |
| | | | | 1.45 p.m. | 2 | 62 | 1¾ | 18,000 | 13.4 | 1.3 | 85.3 | 94.1 | |
| | | | | 2.45 p.m. | 3 | 62 | 1¾ | 24,400 | 20.7 | 3.3 | 76.0 | 83.3 | Converter about 1 hr. 30 min. on finish. |
| Aug. 16 | V | A | 20 | 9.45 a.m. | 1 | 62 | 1¾ | 15,740 | 18.0 | 2.4 | 79.6 | 88.4 | 5 min. after charging 1 boat of ore. |
| | | | | 1.25 p.m. | 2 | 62 | 1¾ | 16,700 | 19.9 | 0.0 | 80.1 | 100.0 | 5 min. after charging 1 boat of ore and 2 pots of matte. |
| | | | | 2.30 p.m. | 3 | 62 | 1¾ | 20,800 | 20.8 | 0.0 | 79.2 | 100.0 | |
| | | | | 3.10 p.m. | 4 | 62 | 1¾ | 25,200 | 16.8 | 0.5 | 82.7 | 97.8 | |
| Aug. 17 | V | A | 20 | 9.45 a.m. | 1 | 62 | 1¾ | 19,200 | 7.8 | 0.5 | 91.7 | 97.8 | 5 min. after charging 1 boat of ore and 1 pot of matte. |
| | | | | 10.45 a.m. | 2 | 62 | 1¾ | 20,000 | 49.3 | 0.0 | 50.7 | 100.0 | |
| | | | | 1.25 p.m. | 3 | 62 | 1¾ | 16,850 | 27.3 | 0.5 | 72.2 | 97.6 | 5 min. after charging 1 pot of matte. |

| | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Aug. 17 | V | A | 20 | 2.10 p.m. | 4 | 62 | 1¾ | 22,800 | 14.3 | 0.9 | 84.8 | 97.0 | |
| | | | | 3.15 p.m. | 5 | 62 | 1¾ | 20,000 | 6.6 | 1.4 | 92.0 | 94.2 | 5 min. after start on finish. Charge very cool. No punching. |
| Aug. 19 | IV | A | 12 | 11.05 a.m. | 1 | 26 | 1¼ | 7,200 | 8.0 | 3.2 | 88.8 | 86.4 | 5 min. after charging 1 pot of matte. |
| | | | | 11.45 a.m. | 2 | 26 | 1¼ | 9,800 | 10.8 | 0.6 | 88.6 | 97.6 | On finish 20 min. |
| | | | | 12.40 p.m. | 3 | 26 | 1¼ | 9,700 | 14.7 | 0.6 | 84.7 | 97.5 | On finish 1 hr. 20 min. |
| | | | | 2.15 p.m. | 4 | 26 | 1¼ | 6,400 | 7.8 | 0.4 | 91.8 | 98.5 | 5 min. after charging 1 boat of ore. |
| | | | | 2.45 p.m. | 5 | 26 | 1¼ | 8,400 | 13.8 | 0.4 | 85.8 | 98.2 | |
| | | | | 3.40 p.m. | 6 | 26 | 1¼ | 6,800 | 10.5 | 0.6 | 88.9 | 97.5 | Doubled with 2 pots of matte at 3.30 p.m. |
| Aug. 20 | V | A | 20 | 9.20 a.m. | 1 | 62 | 1¾ | 19,500 | 14.0 | 0.7 | 85.3 | 97.0 | Blown in at 9 a.m. with 6½ pots inside matte, and 1½ pots trans. |
| | | | | 10.05 a.m. | 2 | 62 | 1¾ | 24,000 | 13.8 | 0.0 | 86.2 | 100.0 | After skimming ½ pot of slag, and charging 1 boat of ore. |
| | IV | A | 12 | 1.15 p.m. | 3 | 26 | 1¼ | 11,500 | 12.6 | 1.0 | 86.4 | 95.4 | On finish, 3 min. after charging 1 boat of ore. |
| | | | | 1.55 p.m. | 4 | 26 | 1¼ | 10,700 | 15.4 | 0.4 | 84.2 | 98.2 | Punching. |
| | | | | 2.20 p.m. | 5 | 26 | 1¼ | 8,850 | 16.9 | 0.1 | 83.0 | 99.7 | 2 min. after charging 1 boat of scrap. |
| | | | | 3.40 p.m. | 6 | 26 | 1¼ | 8,750 | 9.6 | 0.4 | 90.0 | 98.2 | 3 min. after charging 1 boat of ore and 2 pots of matte. |
| | | | | 4.15 p.m. | 7 | 26 | 1¼ | 7,900 | 10.0 | 0.9 | 89.1 | 95.9 | 10 min. after charging 1 boat of ore. |
| Aug. 21 | V | A | 20 | 10.40 a.m. | 1 | 62 | 1¾ | 19,400 | 10.8 | 0.6 | 88.6 | 97.4 | Charged at 7.35 a.m., doubled at 10.15 a.m. |
| | | | | 11.30 a.m. | 2 | 62 | 1¾ | 23,800 | 16.8 | 0.3 | 82.9 | 98.5 | 5 min. after charging 1 pot of matte and 1 boat of ore. |

TABLE VI.—*Efficiency of Air in Converting* (Continued).

| Date 1912 | Converter | | | Time | Sample No. | Tuyeres | | Cu. Ft. Air Per Min. Stand. Conditions | Volumetric Analysis of Gases | | | Per Cent. Efficiency of Air | Remarks |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Class | Bottom | Diameter Ft. | | | No. | App. Inter. Diameter In. | | Acid Gas | O | N | | |
| Aug. 21 | V | A | 20 | 1.25 p.m. | 3 | 62 | 1¾ | 22,400 | 13.4 | 0.6 | 86.0 | 97.4 | Charged 1 boat of scrap. |
| | | | | 2.05 p.m. | 4 | 62 | 1¾ | 31,400 | 14.6 | 0.2 | 85.2 | 98.2 | |
| | | | | 2.40 p.m. | 5 | 62 | 1¾ | 23,600 | 33.1 | 0.1 | 66.8 | 94.8 | Charged 1 boat of scrap. |
| | | | | 3.15 p.m. | 6 | 62 | 1¾ | 23,200 | 35.6 | 0.1 | 64 3 | 94.0 | |
| Aug. 22 | V | A | 20 | 8.15 a.m. | 1 | 62 | 1¾ | 25,000 | 10.9 | 0.0 | 89.1 | 100.0 | Converter charged at 8 a.m. |
| | | | | 9.25 a.m. | 2 | 62 | 1¾ | 19,500 | 9.9 | 0.1 | 90.0 | 99.8 | 5 min. after charging 3 pots matte, 1 boat ore, and 1 boat scrap. |
| | | | | 10.10 a.m. | 3 | 62 | 1¾ | 18,700 | 10.1 | 0.4 | 89.5 | 98.2 | 5 min. after charging 1 pot matte. |
| | | | | 10.55 a.m. | 4 | 62 | 1¾ | 22,800 | 9.2 | 0.2 | 90.6 | 99.3 | |
| | | | | 11.35 a.m. | 5 | 62 | 1¾ | 22,800 | 11.1 | 0.4 | 88.5 | 98.2 | |
| | | | | 12.30 p.m. | 6 | 62 | 1¾ | 21,800 | 10.6 | 0.3 | 89.1 | 98.8 | Charged ½ large pot matte. |
| | | | | 1.35 p.m. | 7 | 62 | 1¾ | 25,400 | 22.7 | 0.0 | 77.3 | 100.0 | 5 min. after charging pot of matte. |
| | | | | 2.35 p.m. | 8 | 62 | 1¾ | 31,800 | 24.4 | 0.8 | 74.8 | 96 0 | On finish 1 hr. 45 min. |
| | | | | 3.00 p.m. | 9 | 62 | 1¾ | 32,200 | 21.5 | 0.0 | 78.5 | 100.0 | Charged 1 boat of dope. |

seem reasonable to suppose that such uniformly perfect results should be obtained, and it was concluded that the oxygen in the gases was used in oxidizing the hot walls of the pipe.

The final sampling pipe was of iron, lined with porcelain tubes. The end of the iron pipe which was inserted in the converter was protected by covering the outside with a clay coating.

The samples were all taken over mercury by aspiration, the acid gases, that is the $SO_2$ and $SO_3$, absorbed by KOH, the oxygen absorbed by pyrogallic acid and the nitrogen determined by difference. Results quoted here are only for samples taken with the proper sampling tube.

For the purpose of comparison samples were also taken of the gases from a Class IV or 12-ft. converter having 26 1.25-in. tuyeres. The results are shown in Table VI.

An examination of the results shows almost uniformly high efficiencies and a study of the figures is of interest.

Picking out all the 100 per cent. efficiencies we find volumes in cubic feet per minute as follows: 16,700, 20,800, 20,000, 24,000, 25,000, and 32,200, which would show that air in volumes up to 25,000 in cubic feet per minute, and even up to 32,200 cu. ft., can be used efficiently and the readings quoted are all on the 20-ft. converter. On the other hand we find a reading of 24,400 cu. ft. with an efficiency of only 83.3 per cent. This was near the finishing of the charge, when it may be possible that air will not be used so efficiently on account of the absence of iron and the small quantity of sulphur present, and available for reaction.

The high as well as the low efficiencies are found with a wide range of volumes which would indicate that conditions may vary and affect the efficiency, but that it is entirely practical and commercial to use large volumes of air, and this is the principle which has been adopted in the design of converters for the plant.

## Size of Converter Mouth.

Considerable experimenting on this question has been done, and without going into an extended discussion of the details it may be stated that a large mouth is an advantage. The largest which has been tried on the Class IV converter is a 6 ft. 6 in. diameter opening in the shell. On the Class V converter this opening is 8 ft. diameter.

The advantages of this are: It is easier to keep clean, that is to remove the crusts; less tendency to close up; easier to put in the charge, ore, etc.

The closing up of the mouth by the building up of crusts will result in making it difficult for the waste gases to escape freely with the result that the converter will become hot and injure the lining. On the other

hand the mouth can be made so large as to allow too free an escape of the gases with the consequent too rapid escape of heat, and a decrease in the cold material which can be smelted.

## Height of Converter.

One of the Class I converters was changed by inserting an extension ring 19.75 in. high between the main trunnion section and the tuyere section, and then putting on a tall cap, making the converter 18 ft. 2.25

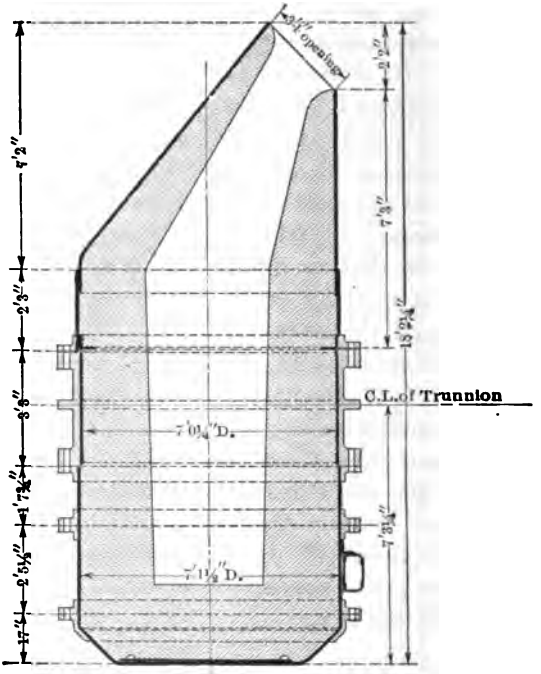



Fig. 18.—Class I Converter, Lengthened 4 ft.

in. high over all and 13 ft. 5.75 in. from the center of the tuyeres to the lowest point of the mouth. A vertical section of this converter is shown in Fig. 18.

This was done to try its effect as to the throwing out of the charge by the blast. After rather an extended trial, the details of which will not be given here, it was determined that no advantage was obtained. These trials were made in August, 1910.

In view of the fact that the Class IV converter 13 ft. 8.5 in. high, and the Class V converter 17 ft. 7.75 in. high do not throw out seriously, it would seem that no advantage would be gained by making the converter

higher. In fact there is a distinct disadvantage as was demonstrated in the trials of the tall converter, namely the clogging up of the mouth by the charge throwing against the lining in the upper part of the cap and chilling there.

## Summary.

The basic lining has shown its superiority over the acid lining and therefore the acid lining can be dismissed as not being the best practice.

A large converter is better than a small one. There is no practical limit to the size of a converter which will work metallurgically. The limits are mechanical, and depend on the size of unit a plant will stand from the point of view of quantity of production, but even in this latter instance there can be a rather wide range of choice. We have found that the Class V or 20-ft. converter can be used as a storage for matte or for finished copper for long periods if the mouth is properly covered. Therefore in a plant of comparatively small production, a combination of ample matte storage capacity in blast-furnace settlers and reverberatory furnaces, together with a large converter, can be made so as to allow all converting to be done on one shift, and the casting of the copper perhaps to be done on the next shift.

With the success of the large units even the largest plants should have only a few large units and not a large number of small units.

From the experiments and observations which have been made, we feel that the tuyeres for any converter from 12 to 20 ft. diameter should be about 2.25 in. internal diameter. Tuyeres 2.75 in. internal diameter have not proved successful. Tuyeres smaller than 2.25 in. diameter introduce undesirable construction features, particularly weakening the lining at the tuyeres, introduce unnecessary resistance to the entrance of the air, and add to the work of punching.

The mouth of the converter should be of ample size. Openings of 6 ft. 6 in. diameter for a 12-ft. converter and 8 ft. for a 20-ft. converter have not been found too large.

The bottom lining must not be too close to the tuyeres on account of a tendency of the bottom to build up and interfere with the punching of the tuyeres and also often causing a concentration of air in a few places with consequent throwing out of the charge. At least 5 in. should be allowed from the lowest point of the tuyeres to the bottom lining. There is no reason why this dimension cannot be increased, the determining factor being the size of charge it is desired to finish. There must always be copper enough at the finish to cover the tuyeres and a very deep bottom would require a large charge.

Air can be used efficiently in large volumes, for instance in the 20-ft. converter up to 22,000, or even more, cubic feet per minute. However,

the best results seem to be for this converter about 18,000 cu. ft. while slagging.

As to the height of the converter, the Class IV, 13 ft. 8.5 in. high, and the Class V, 17 ft. 7.75 in. high, have proved very satisfactory with proper attention to the details of operation, and trials of a taller converter were not satisfactory. Probably the converter should not be made appreciably lower. The proper dimension to quote in this particular is from the center of the tuyeres to the lowest point of the mouth. These dimensions for the two classes are: Class IV, large mouth, 7 ft. 11 in.; Class V, 8 ft. 9.25 in.

These we believe to be a few of the basic principles which have been demonstrated by the experimenting and practice at this plant.

We wish to acknowledge our indebtedness to Peter Thill, A. T. Elliott and James Moore, who have been members of the staff from the very early days of the plant and are so still, for their recollections and descriptions of the early construction and practice.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.
[SUBJECT TO REVISION].

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Butte meeting, Aug. 18 to 21, 1913, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1913, when Vol. XLVI. of the *Transactions* will go to press. Any discussion offered thereafter should preferably be in the form of a new paper for publication in Vol. XLVII. (with suitable cross references in both volumes).

# Shaft-Sinking Methods of Butte.

BY NORMAN B. BRALY, BUTTE, MONT.

(Butte Meeting, August, 1913.)

THE following is not offered as an extended paper on the subject of shaft sinking, but more as a description of the present practice of shaft sinking in the Butte district.

The Anaconda company is sinking at present the following shafts: Badger State, Moose, Mountain Consolidated, Mountain View, Pennsylvania and Tropic. At the High Ore and Leonard mines the shafts are being deepened by sinking two compartments and will subsequently be enlarged to the full working size by raising alongside the compartments which are now being sunk.

The following other companies are also sinking shafts: Butte-Alex Scott, Butte and Zenith City, Boston and Corbin, North Butte Mining Co. (at the Granite Mountain shaft), East Butte Co. (at the Pittsmont shaft), and the Rainbow Lode Development Co.

## *Shaft Timbering.*

The shafts of the Butte district, generally speaking, are of three compartments arranged in a straight line, and the pump compartment is usually arranged for the use of an auxiliary hoisting cage, as well as for carrying pipe lines and electric wires. The Rainbow shaft is practically square, and of three compartments. The length of the shafts outside of timber will vary from 17 to 22 ft., and the width from 6 ft. 6 in. to 7 ft.

Fir timber is used in all shaft work. This may be obtained from the Coast, when it is known as Coast fir, or from Idaho or Montana, when it is known as native fir. The native fir seems to be the tougher wood, but it is hard to obtain clear lumber in the lengths and sizes necessary for shaft work. Some of the shaft timber of the Anaconda company has been treated with creosote as a preservative, but this practice has not come into general use, as the ground of the camp is so heavy that the timbers usually crush before they rot. The size of timber most used for shaft work is 12 by 12 in., while 10 by 12 in., 10 by 14 in. and 12 by 14 in. are used for some of the special pieces around the shaft. The new Leonard shaft is timbered with 14 by 14 in. wall plates throughout.

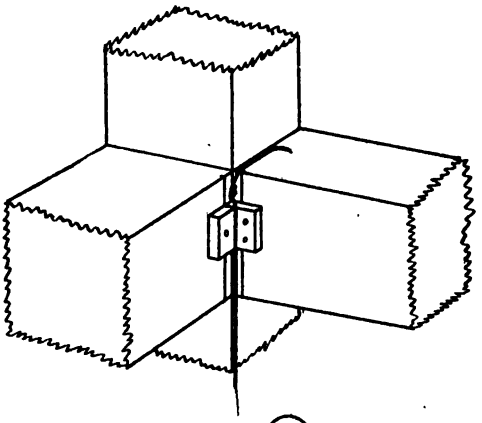
No machinery has been introduced here as yet for the framing of shaft timbers, although the North Butte company is considering such a machine, and all of the timbers are therefore hand-framed. It may be of interest to note that ½ to ¼ in. is generally held off the wall plate and end plates at each end in the framed part to make the mitre draw close together when the set is being blocked; thus the wall plate for a shaft 20 ft. long would be 19 ft. 11 in. long. In order to hold the set together when it is being blocked, 1-in. wooden pins are driven at each corner, in holes bored for that purpose. These pins are made by driving 1-in. square blocks of wood through a 1-in. round hole in iron 1 in. thick.

To keep the shafts vertical, blocks nailed together (Plate I, Fig. 2), made of boards $\frac{15}{16}$ in. thick are used, so that half the width of the line will bring the center of the line 1 in. from the corner of the set both ways. While the timber is being framed, the carpenter marks with his scratch awl a line on the timber 1 in. from each corner, as shown in Fig. 1, Plate I, and this line saves a great many rule measurements in the early blocking of the set. At about every 50 ft. these lines must be lowered to new blocks. A set is selected at a point about 50 ft. below the blocks, and measurements are taken each way from the corner line to determine how far the line is from the timbers. The new blocks are then nailed in place and the lines suspended from them. If the measurements to the timbers do not correspond to the old measurements, the block is cut with a pocket knife to bring the lines exactly to their former position. It is well to measure diagonally across between lines at about every 100 ft. to see if the shaft remains square.

A transit should be taken down before reaching a station, and readings taken from the corners down to the level of the instrument. Posts are then cut to bring the shaft timbers level. This is called a leveling-up set; this leveling may be done with a carpenter's level, but not as quickly nor as accurately.

The placing of the collar set is the first thing to be done in starting a shaft. Its height above the ground is determined by the amount of room available for the dumping of waste rock from the sinking operations. In a level place it is generally raised about 20 ft. The wall plates of the collar set are usually made 10 ft. longer than the shaft set, projecting 5 ft. at each end. This set bears the weight of the shaft sets below, and usually of the gallows frame. Posts resting on blocks are placed under each end, and diagonal braces are then placed around the shaft to carry part of the weight and prevent the set from moving. Figs. 3 and 4, Plate I, show the collar set of a two-compartment shaft, with the diagonal braces and posts.

The size of the guides varies greatly. At first 4 by 6 in. guides were used almost universally in the camp, but lately the size has been



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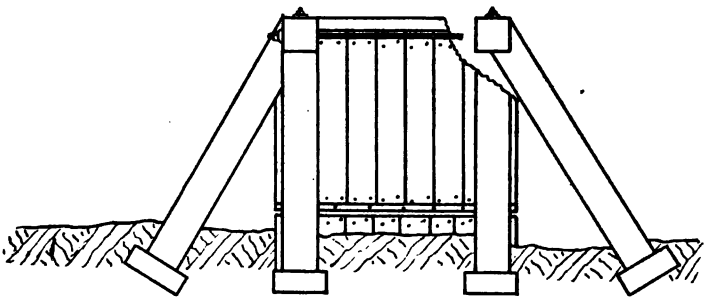

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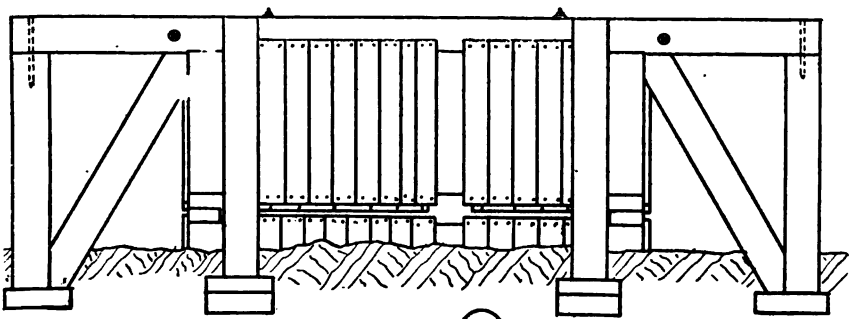

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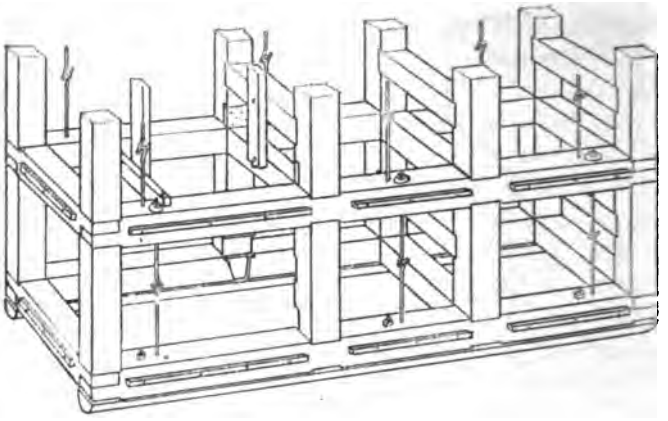
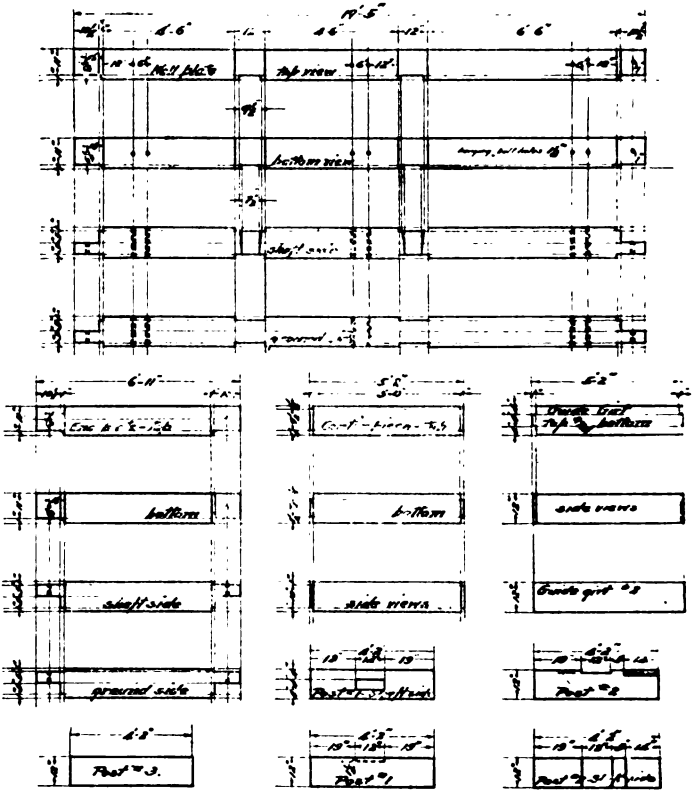
PLATE I.

1. Block Set for Lining Timber in Place.
2. Block before Nailing.
3. Collar Set, end view.
4. Collar Set, side view.






List of pieces

| | | | |
|---|---|---|---|
| Wall plates | . 2 | Post #1 | . 4 |
| End | . 2 | . #2 | . 2 |
| Center pieces | . 2 | . #3 | . 2 |
| Guide girt #1 | . 2 | | |
| . . #2 | . 1 | | |

PLATE II.—GRANITE MOUNTAIN SHAFT OF THE NORTH BUTTE MINING CO., USING GUIDE GIRTS IN THE HOISTING COMPARTMENTS. 12 BY 12 IN. USED THROUGHOUT.

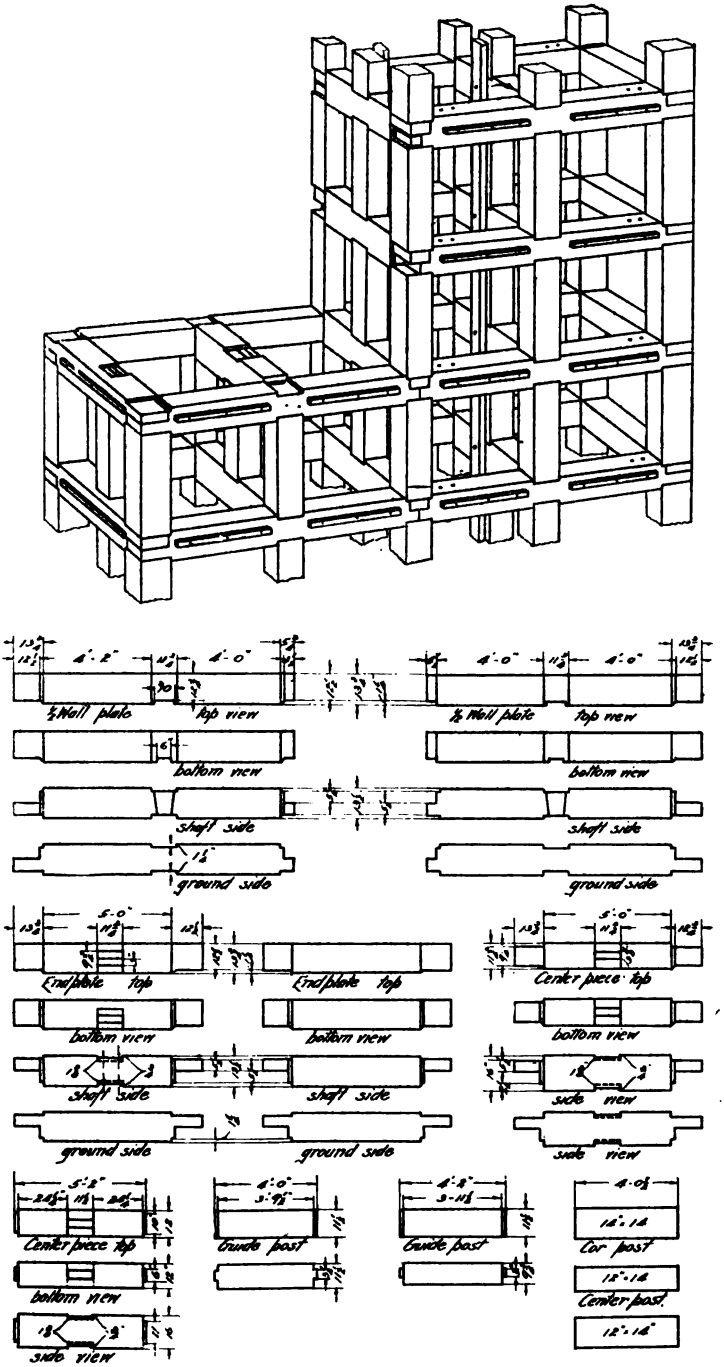



PLATE III.—LEONARD SHAFT NO. 2. SINKING TWO COMPARTMENTS, THEN RAISING TWO. 14 BY 14 IN. WALL PLATES.

increased to 4 by 9 and 5 by 9, held by ⅞ by 8 or ⅞ by 9 in. lag screws. The larger size guide can stand heavier loads and greater shocks, and two lag screws are used in each timber; the holes, two in number, are placed diagonally across the guide, and, after a number of years, they are placed diagonally across in the other direction, giving fresh timber for the screws to hold to.

Guide posts are vertical posts running behind the guides to give them more support. They may be seen in Plate III. They are framed into the centers and end plate of the shaft. Where used they are carried in the main hoisting compartments, and sometimes in the auxiliary hoisting compartment, but not in the pump compartment. They are the same width as the center, and usually 12 in. in the other direction.

Guide girts are girts running horizontally between posts and equidistant between the centers of the shaft. They are framed into the posts of the shaft, as seen in Plate II, giving support to the guide and also helping to take the side weight of the shaft. They are being used at the Granite Mountain shaft of the North Butte company, and will be used in the retimbering of the Neversweat shaft of the Anaconda company. In heavy ground the guide girt would seem to be more satisfactory than the guide post.

To allow for the end plate of the shaft pushing in, 2 in. is often allowed in the outside hoisting compartment. To make the distance between guides the same size in the two compartments, a 2-in. block of wood is placed between the guide and the end plate of the shaft. As the end plate comes in, the block is cut down or removed until the guides are again in the right distance apart. This is called a filler. Sometimes, as at the Badger State mine, the filler may be a vertical post carried up the shaft against the end plates, and bolted with ⅞-in. bolts to the guide posts. At this mine a 10 by 12 in. post, in 20-ft. lengths, is carried up alongside the guide posts, and a 6 by 12 in. post is carried up in the other hoisting compartment on the opposite side. At the Moose shaft these fillers are carried behind each guide, the same size for both sides of the hoisting compartment, and are, I believe, 4 by 12 in.

In sinking a shaft spliced wall plates are often used. In heavy ground the ground cannot be kept open to a depth to allow a full length wall plate to be turned in a shaft so as to be brought into place and, as long length timber is hard to obtain, the splice is coming into more general use. Four hanging bolts must be used on a side with spliced wall plates. The splice, shown in Plate V, Fig. 1, is the one which finds most general use. In this the center is supported by the wall plates while the set is being blocked. Fig. 2 shows the splice when one side of the shaft is to be raised. It will be noted that the mitre is left out and the framing is straight. This is because dirt getting into the straight joint will not throw it out of line

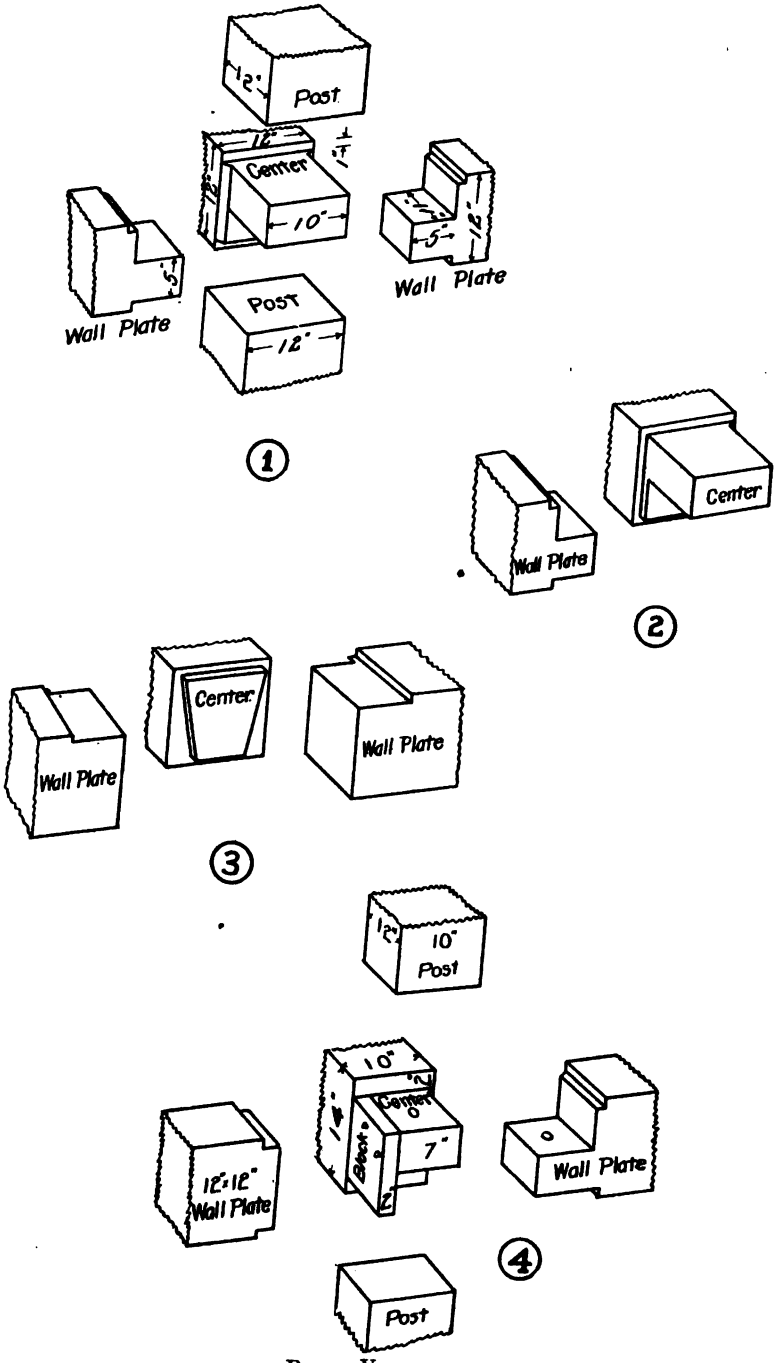



PLATE V.

1. Usual method of splicing wall plates.
2. Splice where half of shaft is raised.
3. A common method of splicing wall plate.
4. Method of splicing wall plate at High Ore mine.

as much as it would the mitred joint. When the shaft is being sunk a block of wood is placed in the opening under the center, and, when the other half of the shaft is raised, this block is chopped or picked out and the other half of the wall plate placed in position.

Fig. 3 shows a spliced joint very easy to frame. The center in this case, however, has to be supported while the set is being blocked. This is usually done by nailing it to the wall plate before the posts have been placed in position. At the Tropic shaft the set is blocked on both sides of this center temporarily; the end blocks are then set, and the blocks on each side of this center are then taken out and the set brought to line and blocked permanently. A temporary post is sometimes placed under the center to hold it up while being blocked. This method of splicing is cheaper to frame, but a little more expensive to put in place.

Fig. 4 shows the method of splicing at the High Ore mine. A 2-in. block is nailed to the center before it is sent down into the mine and the corners of this set are held together with pins. The other half of the wall plate can be easily pushed in, as the distance is small. This is not as strong a splice as the one shown in Fig. 1, but would be excellent where a small shaft is being sunk and there is a possibility that more compartments will be added. On the whole, it may be said that it is cheaper to sink the whole shaft than it is to sink part of it and raise the balance. In soft ground where it is not desired to open up too much country at once, the Anaconda company finds it an advantage to sink and raise.

The timbering takes place from the top of the muck pile whenever possible. That is, the dirt, after blasting, is mucked down to about 10 ft. below the last set. The new set is hung and blocked; then mucking is continued. The men do not have to build any staging to drive the wedges, which are driven from below. If the timber is kept close to the bottom it is sometimes found after blasting that the muck is pushed up to within a few feet of the last set of timbering. If the men could work only in one compartment mucking would be very slow; so, to avoid such a possibility and to aid in turning the timbers to get them in place, one of the centers of the shaft on either side of the hoisting compartment is left out on the bottom set. A 2-in. strip is cut from the post and the center is pushed in through this opening and hammered down into place when the set below is put in. A block made to fit the opening in the post is then nailed into place, completing the set. This may be seen in the Rainbow shaft set, Plate IV, and the Granite Mountain set, Plate II. In the Granite Mountain shaft it will be noticed that the guide girt is also left out. The guide girt is slipped in from the side; then it is wedged on the end and the side weight of the shaft soon makes it solid. The center and girt are always kept within one set of the bottom timber; this style of center is framed with a dovetail.

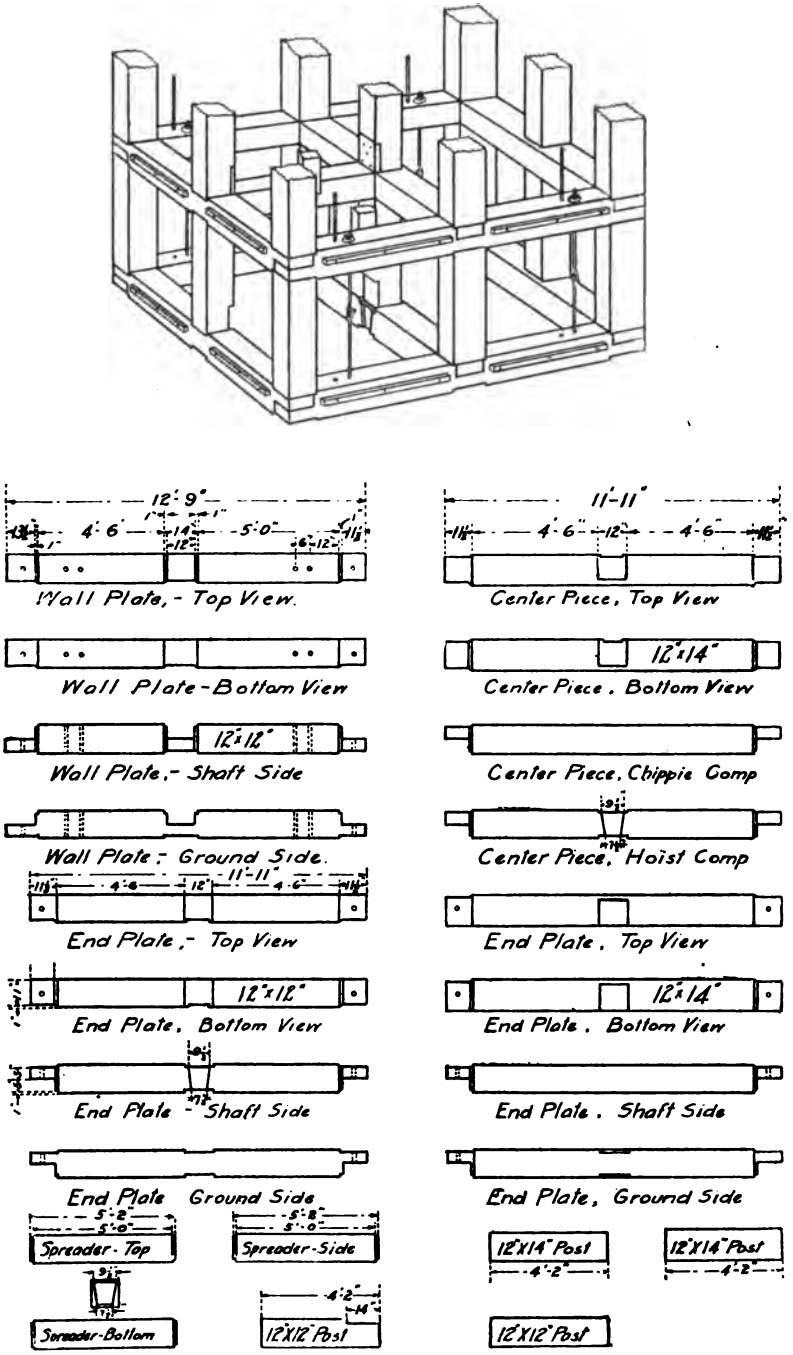
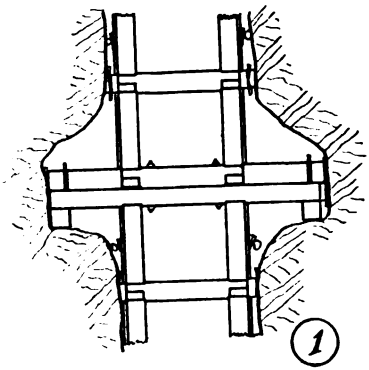
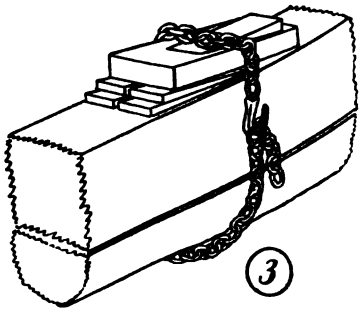
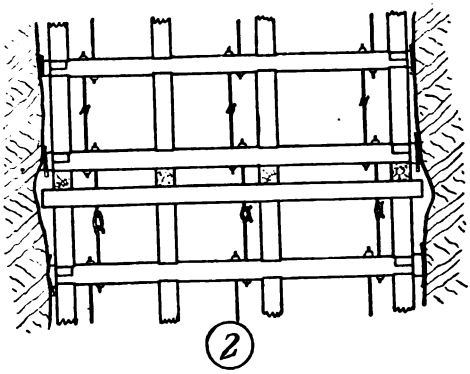



PLATE IV.—RAINBOW SHAFT. AN ALMOST SQUARE SHAFT OF THREE COMPARTMENTS. 12 BY 12 IN. AND 12 BY 14 IN. WALL PLATES.



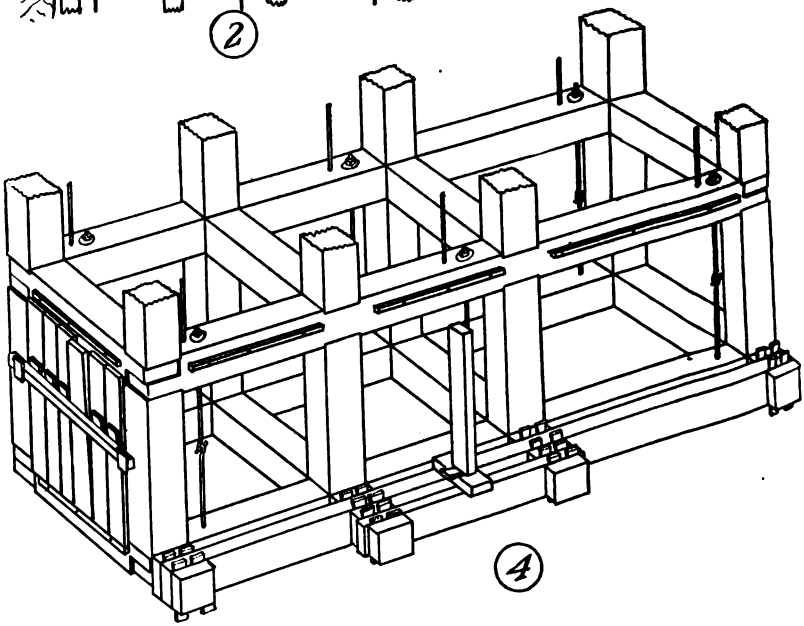




PLATE VI.

1. Shaft bearers, end view.
2. Shaft bearers, side view.
3. Method of fastening blasting logs with chains.
4. Method of blocking and lagging set.

Where the guide post is framed into the center with a mortise and tenon, the center cannot be slipped in from the top. It is then framed with straight sides and is pushed up from the bottom and held in place by the posts of the set below when they are being placed. This may be seen at the Mountain View mine. This method of pushing the center up from below is sometimes used at other shafts on account of the cheaper framing. The first method is considered the best for holding out the center. It sometimes happens that the wall plates of the shaft have been

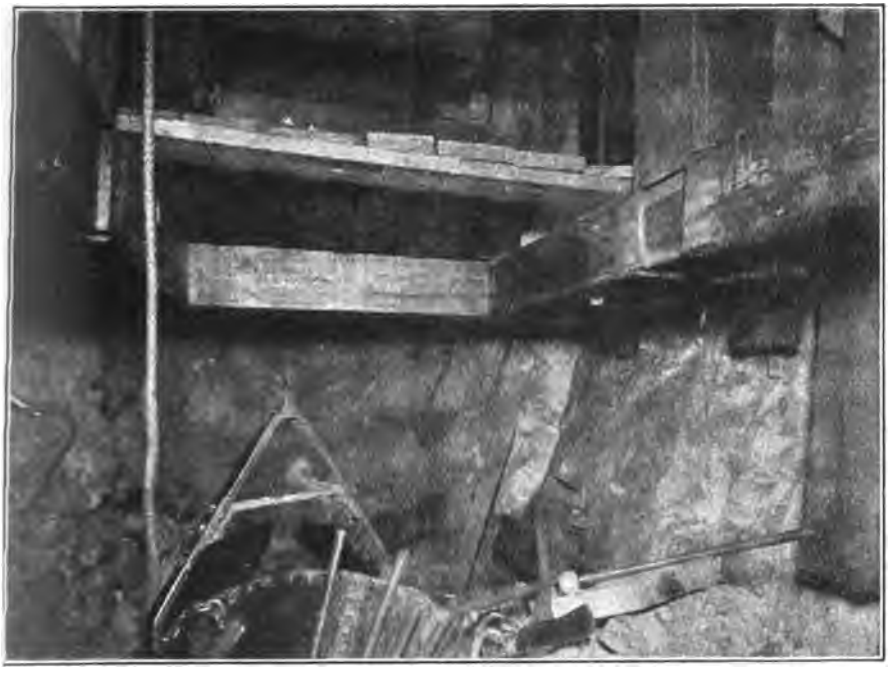


PLATE X.—TAKEN WHILE SET WAS BEING BLOCKED. GRANITE MOUNTAIN SHAFT, NORTH BUTTE MINING CO.

pushed in so that the center is hard to put in place. A small bar of 3-in. pipe, similar to a stoping bar, is then used to spread the timber.

The method of blocking and lagging the set may be seen in Plate VI. Where the distance is small,—say, not over 1 ft.,—a single block is placed between the set and the wall, the wedging being done between the block and the wall. When, however, the ground breaks back farther, a block of about 5 in. thickness is placed back of the center and toe-nailed from above. Blocks of the same size are placed along the side and toe-nailed. A stringer the length of the shaft is then placed along the side and nailed to the blocks. Blocks are again placed, and then stringers as needed. The wedging is then done between the blocks and stringers. The wedges

should be driven from below as well as above. An upright post is then put in place and wedged to the blocking above to prevent the blocking below from being knocked out by the blast. After the set has been hung and the posts are all in place, the set is generally raised up by placing a pinch bar under the hook on the hanging bolt; then the set is raised and the nut run down as far as possible by hand. In case blasting chains are used the bar may be supported by hooking the chains around

PLATE XI.—BOTTOM OF RAINBOW SHAFT, WHILE MUCKING. BLASTING LOGS HELD WITH CHAINS.

the hanging bolt above. In case bolts are used a U-shaped bar of ¾-in. iron, with hooks on the side, is made for this purpose. The top end of such a bar may be seen in Plate X, on the right-hand side of the bucket.

Two-inch fir is used for lagging. A strip is placed horizontally, back of where the lagging will come and about half-way between wall plates. The shaftman uses his ingenuity to find some method of support for this stick. The lagging timbers are then placed in and wedged from the back, the shaftman leaving one plank out for this purpose. This last

lagging is wedged from the inside of the shaft. Two-inch square strips support the lagging and are nailed to the wall plates before they are sent into the mine.

Blasting logs protect the timber from being cut up by the blasting. They are fastened to the bottom set with bolts or chains, as shown in Plate XI, which was taken in the bottom of the Rainbow shaft. Timber 8 by 12 in. is often used for this purpose; or a 14-in. stull, split lengthwise, makes an excellent blasting piece, as the round surface tends to throw off the rock. One-quarter inch iron and 3 by 3 in. angle iron, placed along the edge of the shaft timber, are also used. The iron blasting pieces, however, are twisted out of shape by heavy blasting and must be sent to the surface frequently to be straightened by the blacksmith. Where bolts are used to hold blasting pieces, they are of 1-in. size, and the head of the bolt is always placed down. Plate VI, Fig. 3, shows the method of tightening a blasting log held with a chain. The blasting log is held up, the chain is hooked over the timber, and a block is then slipped under the chain. Wedges are then driven under the block; a few blows with an ax makes them tight, and a side blow loosens them again when necessary. This is probably the best and most reliable method of holding blasting logs.

Figs. 1 and 2, Plate VI, show what is known as a shaft bearer. About every 400 or 500 ft. the shaft is widened; sills are laid along the side of the shaft as shown, and cross pieces are then bolted under the end plates and centers of the shaft, resting on the sills. The shaft set is then blocked and wedged on top of these cross pieces. Regular posts are then placed under the bearers when sinking is resumed. Shaft bearers have not come into general use, although they may be seen at the Leonard and Badger State shafts of the Anaconda company. The object of the bearer is to prevent the shaft from sinking on one side and to give additional support to the shaft below when sinking.

I will not enter into a description of any particular shaft, as it is hoped that the drawings will make themselves clear.

### *Sinking Equipment and Methods.*

The No. 3 Rand, E 44 Sergeant, and jackhammer drills are the ones most in use in shaft sinking. With the reciprocating drill the tendency is toward long rounds, 10-ft. rounds being broken where the rock is hard; with the jackhammer drill a 5-ft. round is drilled. All sinking is done with cross bars where the reciprocating drills are used, and these bars are generally made 6 in. longer than the width of the shaft timber, while a longer bar and a shorter bar are used for special set-ups. The bars are made of extra heavy 4-in. pipe.

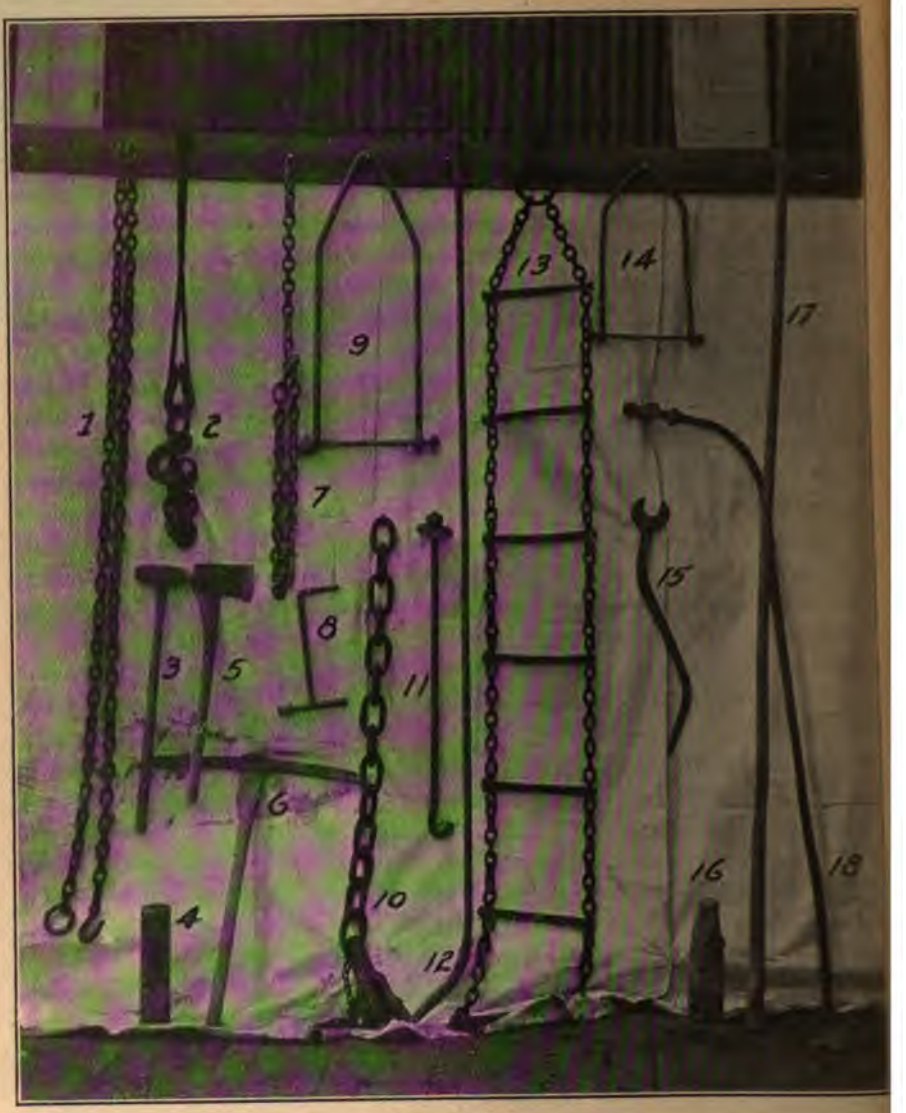



PLATE XII.

1. Rock Chain.
2. Flat Link Method of Fastening Bucket.
3. Rock Hammer.
4. Pipe cut for Collar of Drill Holes to keep Rocks from Rolling In.
5. Axe.
6. Pick.
7. Blasting Log Chain.
8. Wrench for Screwing in Guide Lag Screws.
9. Timber Clevice.
10. Drop Pin and Chain to go under Cap.
11. Hanging Bolt.
12. Sand Pump, 0.75-in. pipe, and made in different lengths.
13. Chain Ladder.
14. Guide Clevice.
15. Hanging Bolt Wrench.
16. Wooden Plugs for Drill Holes.
17. Wooden Tamping Stick for Powder.
18. Blow Pipe, 0.75-in. pipe, and made in different lengths.

For cleaning the holes sand pumps are used. (Plate XII.) These are made of ¾-in. pipe with a bend near the end. The shaftman places his hand over the end as the pipe is raised, causing a suction, taking it away as the pipe is pushed down. When this is done rapidly it lifts the sand out of the bottom of the hole. The sand pumps are made in different lengths and sizes to suit conditions. Blowpipes are used for blowing out the holes before blasting, and are usually made of ¾-in. pipe, so that small rocks will have a chance to pass.

As to the placing of holes, Plate VII will show the general method, as there are no ironclad rules. If the ground is breaking out too far on one side the holes are drawn in, or perhaps one may be cut out altogether. It is generally arranged so that each machine has the same number of holes to drill, and this often causes rivalry among the different runners. Both 40 per cent. and 60 per cent. gelatine powder are used, the 60 per cent. being usually used in the cut holes only. In some shafts the whole round is blasted together, but, with 10-ft. rounds, generally the cut holes and one or two rows on either side are blasted first; thus making sure the round will do its work. The cuts are sometimes mucked clear to bottom and the bottom of the cuts blasted with the rest of the round. At other places it is customary to muck down far enough to see that all of the holes have gone and then shoot the rest of the round. The men in this way are twice on the top of a fresh muck pile, and only pick bottoms once to a 10-ft. round. Sometimes a set of timber is placed after the cut holes are blasted and then another after the rest of the round, making both very nicely from the top of the muck pile.

The jackhammer drill is finding its application in certain kinds of rock. The air is blown out through the hollow drill, keeping the holes free from cuttings; in soft ground the holes are likely to plug. At the Pittsmont mine a hole is drilled in the steel sidewise about 1 in. from the lower end of the drill, so that, if the hole in the end becomes clogged, the air blows out through the side, keeping the hole clean above the last inch of the drill. In soft ground the drill bits are enlarged to take 1.25-in. powder. In hard rock, if the bits are enlarged sufficiently to make the hole large enough for 1.25-in. powder, the drilling becomes too slow, while the ⅞-in. powder will not break to the bottom cleanly, and then picking bottom is a slow process. The application of these drills seems to be in rock which is neither too hard nor too soft. A heavier drill of this type might overcome these difficulties if it bored a hole large enough for 1.25-in. powder fast enough in hard rock.

The fuse is doubled for blasting the holes; two nails or hooks are set the right distance apart and the fuse is wound tight around them, and then cut at one of the nails. Caps are put on both ends of the fuse; the crimp in the fuse, made by the other nail, marks the place where the fuse is

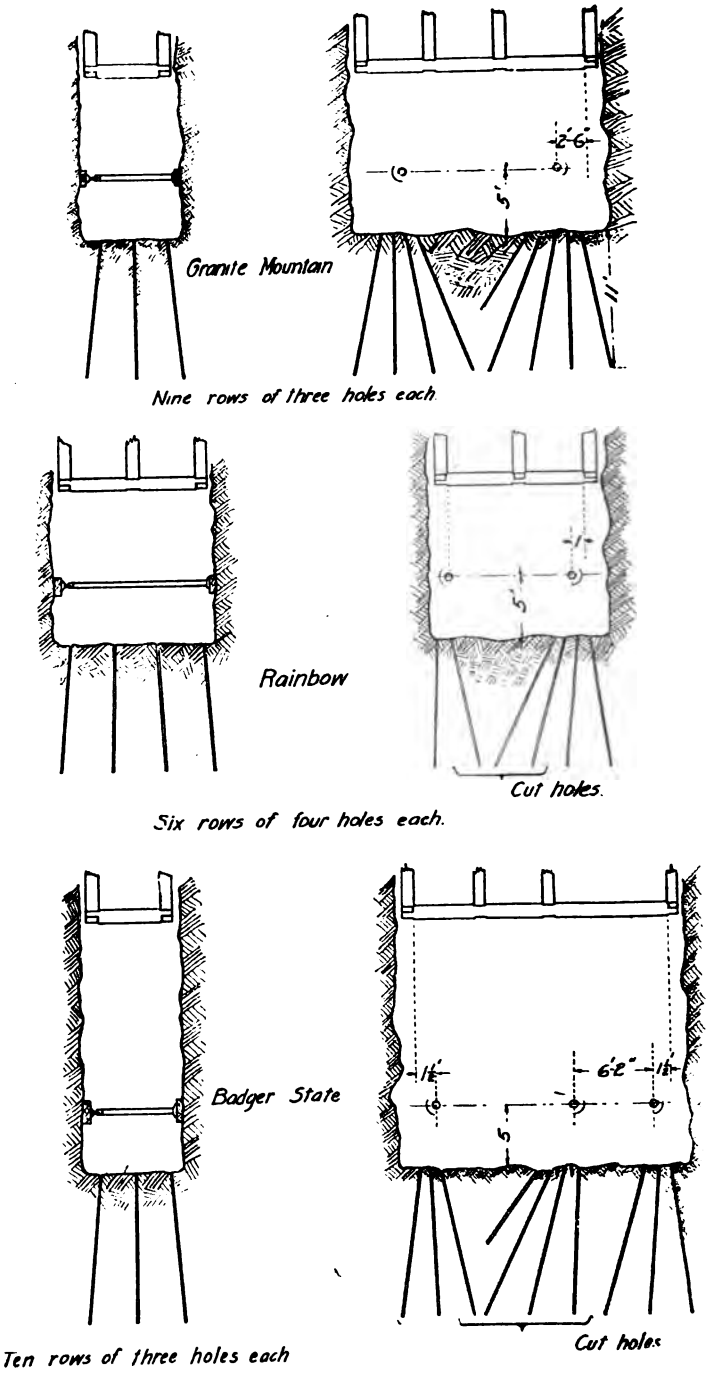



PLATE VII.—PLACING OF DRILL HOLES IN SHAFT.

to be spit, and the fuse is not cut until that particular fuse is spit. The fuse for the different rows of holes is not cut in different lengths, since the work of spitting gives plenty of relay between fuses.

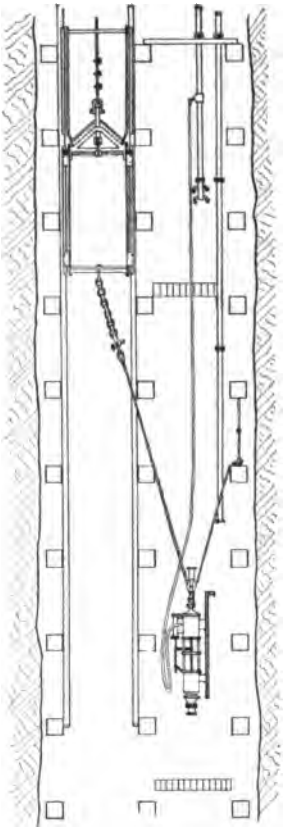
In the larger mines sinking takes place under a working shaft, the shaft being bulkheaded under the hoisting compartments and several hundred feet above under the auxiliary compartments. A lining of 2-in. or 3-in. plank is run, separating the auxiliary and main hoisting shafts between these two bulkheads. The sinking engine is then placed so it may be served by the main hoisting cages. This bulkhead is often made by crossing the shaft with several layers of 10 by 10 in. timber, removing the shaft lagging for several sets above and filling it with rock. Sometimes a timber bulkhead is placed, as in Fig. 4, Plate VIII. A good method is shown in Fig. 2, Plate VIII. A block of ground is left under the hoisting compartments for about 30 ft., when the shaft is lengthened out to the full. When sinking is finished this portion is raised, completing the shaft. This may be seen at the Badger State and Mountain Consolidated mines of the Anaconda company.

### *Handling Shaft Water.*

Some of the shafts of Butte, where the ground has been drained, are being sunk without pumps. In this case the water is bailed into the sinking buckets with pails. The bucket is then hoisted and dumped into a car. The car can be run out away from the shaft, and the plug pulled out; it will empty itself by the time the next bucket of water is ready to be hoisted.

The 10-7-5-10 and the 9 b Cameron pump are generally in use in sinking work. It is becoming the practice of the camp, where pumping with compressed air, to force the exhaust air back into the water column. A check valve is used to keep the water from running back into the cylinder of the pump when the pump is shut down. A valve is placed so that the air may be exhausted into the atmosphere. A valve should be placed in the water column, to allow the column to be drained in case the water pressure is too great to allow the pump to start. A pump so rigged, but for a short lift, and without the column valve, is shown in Plate XIII. It is shown with the 6 to 4 in. nipple on the suction, to make the flanges standard, and with the expansion joint on the water column.

Pumps run in this way are noiseless and will not freeze in a cold shaft because the air leaves the cylinders of the pump still under pressure. The advantages to be gained by this method are that the column is not under such heavy pressure and is less liable to breaks, and the pump will give less trouble with the packing. A 200-ft. lift can be made very nicely with a 10-7-5-10 Knowles pump. The column pipes are flanged in 10-ft.



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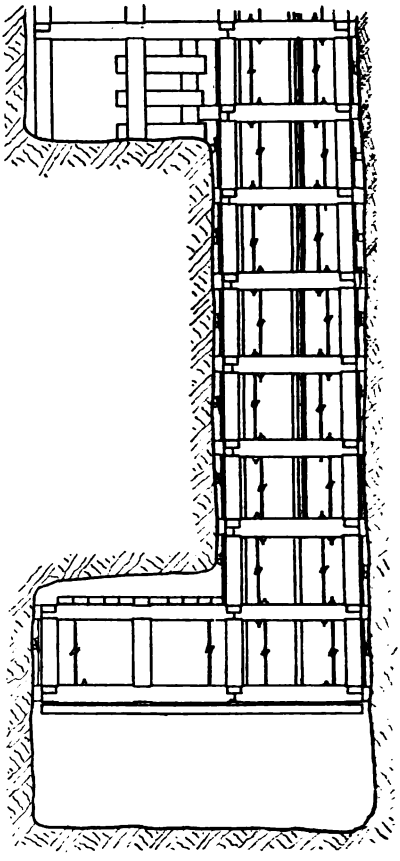

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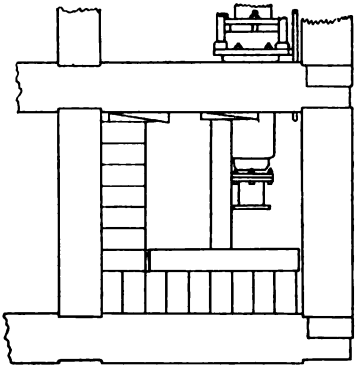

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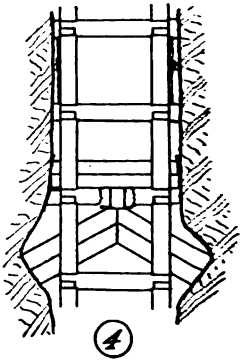

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PLATE VIII.

1. Lowering pump with rope.
2. Leaving a block of solid ground under the hoisting compartments of the sha
3. A bulkhead under the pump.
4. End view of a timber bulkhead under the hoisting compartments.

lengths, and care should be taken to keep the flanges away from the wall plates. They should be started and held vertical about half-way between sets. Expansion joints should be placed every 200 ft. in the water and air lines. The pump has an expansion, as shown in Plate XIII, which is lowered with it. This takes care of any variation in the 10-ft. lengths

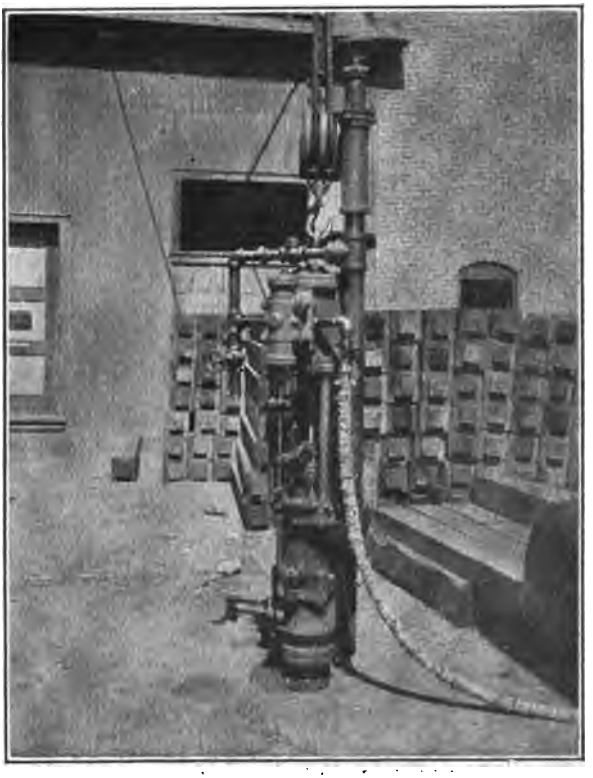

PLATE XIII.—10 BY 7 — 5 BY 10 IN. KNOWLES-PUMP. CONNECTED TO THROW EXHAUST AIR INTO WATER COLUMN.

of pipe. The clamps on the water column should be kept 30 to 40 ft. above the pump. In case the pump is not lowered to a position exactly under the pipe, the pipe can be swung over to the pump.

There are several methods of lowering pumps, the favorite one being with chain block. The chain block is fastened to a cross piece several sets above, which is carried down along with the chain block. A hand-operated winch, set in the pump shaft usually at the first station, and a rope leading down to the pump, are used. A separate bell line gives the man at the winch the signal to hoist or lower the pump. The pump may be lowered with a large-size Manilla rope, as shown in Plate VIII,

Fig. 1. This method is very fast. The rope is wound around the timber and tied after the pump is lowered, so that, if the pump is lifted off the wall plate by the blasting, it will not fall to the bottom.

To keep the pump from being blasted a bulkhead is carried along and kept under the pump. As these bulkheads have to be passed down by

PLATE XIV.—BUCKET AND HOOK ARRANGEMENT FOR SUPPORTING BUCKET. TROPIC SHAFT, ANACONDA CO.

hand every time the pump is lowered, the timber should be in as light pieces as possible, and 5 by 10 in. or 4 by 10 in. set on edge makes a nice size for this purpose. The bulkhead is designed by the pumpman to suit existing conditions. An end view of a typical bulkhead is shown in Plate VIII, Fig. 3. It is wedged tight in place, and, after the blast, it is loosened so that it may be easily taken out.

The suction of the pump extends through this bulkhead to the sump below; consequently the bulkhead is made so that it can be closed for each blast. Before the blast the suction is disconnected and pulled up above the bulkhead by a double block and tackle. One way to make

this hole for the suction is to have the pieces below the pump wedged so they can be pulled up and out; or they may be kept loose so they can be spread. Another method is to take two 10 by 10 in. timbers and cut a 5-in. hole in each, so that, when they are placed under the pump, they make a 10 by 10 in. hole for the suction. During a blast a block is placed

PLATE XV.—DROP PIN. METHOD OF SUPPORTING BUCKET. RAINBOW SHAFT.

over the hole and under the pump, to prevent rocks from flying up. In some places the suction is pulled just high enough to leave the strainer in the hole. The strainers for the suction hoses are made at the mines and consist of a piece of pipe of the same diameter as the suction, about 14 in. long, bored full of ¼-in. holes, with thread on one end, and flattened together on the plain end. Timbers 10 in. square are very heavy when they are waterlogged, and most pumpmen prefer the first method. The suction hoses are usually wound with ¼-in. Manilla rope, as in Plate XI, to prevent wear. Short lengths of flanged pipe are kept with the pump, and are placed between the suction hose and the pump as the rock is being mucked down. As the suction hose can be lowered with the block and

tackle, this is an easy matter. A leak in the suction is a great annoyance and about the only way to find it is to place the ear near the flange joints and listen for the sound of the leak.

### *Rock Handling.*

Sinking engines have every possible speed, depending upon the depth they have to work. The type of cage shown in Plate XVII is generally

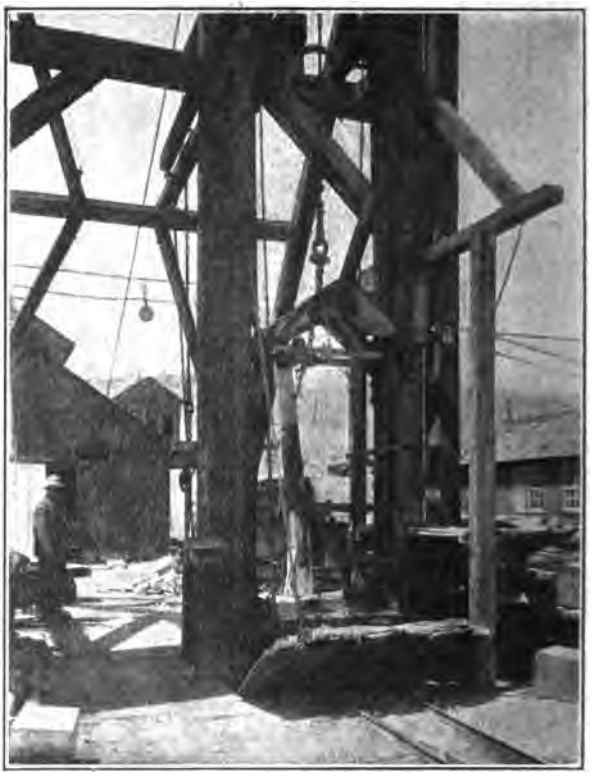

PLATE XVI.—CAGE AND SINKING SHOES. RAINBOW SHAFT.

used for sinking; the long shoes along the side and the sinking shoes above can be seen. The corner bars of the cage are drawn in so that men can step off the cage on to the timber. In Plate XVII a regular mine cage is fitted for sinking, having sinking shoes. The sinking bucket with straight sides has come into almost universal use. These buckets are about 2 ft. 6 in. across the top, and 3 ft. high, holding one miner of rock. Barrel-shaped buckets and buckets wider at the top than the bottom have not found much favor. The bucket must be wide

the bottom so that it will sit steadily on the muck below, buckets with small bottoms being likely to topple over and hurt the men. There are many novel methods of securing the bucket; the best and most reliable one, in my estimation, being shown in Plate XV, which is known as the drop pin. Another method is shown in Plate XII, Object 2, this being

PLATE XVII.—USUAL FORM OF SINKING CAGE AND SINKING SHOES. TROPIC SHAFT, ANACONDA CO.

a link fastened for about 0.75 in. on one side, and which can be passed into the hook shown, the chain being passed around the bail of the bucket. Plate XIV shows two hooks turned in opposite directions, this system is being used at the Tropic shaft of the Anaconda company.

The cars should be large and have strong bottoms. Large boulders are lifted into the bucket by means of a rock chain attached to the cage, and dumping these into the cars soon breaks them if they are light. Plate XV shows a common mine car with raised door, so that large boulders may be dumped out of it. A rock chain may be seen in Plate XII. It is well to have several of these made in different weights of chain, and

0.5-in. chain is a good average size for this purpose. A chain ladder is also shown in Plate XII, this being made of 0.5 by 18-in. bolts, with ⅜-in. chain and 0.75-in. pipe cut to go over the bolts between the chains. The ends of the bolts are riveted after the nuts are in place; in case a step is broken it is easily replaced. The chain ladder should be hung so that, in case anything should go wrong with the sinking engine, the men can climb up the chain ladder and out. In some places this is lengthened to about 35 ft., and the wooden ladders are kept down to connect with the chain ladder.

*Miscellaneous Details.*

Where heavy ground occurs in the shaft, and the timber is being pushed in and broken, jacket sets are placed outside the regular shaft timbers. (Plate IX.) This gives an opening around the shaft, where men can work easing off the timber while the shaft is in operation; these jacket sets are raised alongside the shaft, although in a few instances they have been carried down with the sinking. The four small compartments at the corners are used as chutes, and discharge at the station below. The rock is generally very fine, and is picked out in easing off the timber. If any of the outer jacket timbers break they can be easily replaced without disturbing the shaft.

Sometimes the shaft sets themselves are pushed out of line. Vertical stringers are then run up and down the shaft, being wedged to three sets all around. The jacket timber of the middle set can be then wedged and held from coming in when the blocks between them and the shaft are taken out. The shaft set is then pushed over with a jack bar to its right position and new blocks are fitted to go between the shaft set and the jacket set. This is then repeated on the next bad set. In raising alongside the shaft for the jacket sets, a set is finished around the shaft before commencing another. The place where the ground is the heaviest is selected as the starting point for the raising; what natural support the better ground can give is thus taken advantage of.

For signals, a 0.75-in. Manilla rope, attached to a bob, is generally used, and the shaftman ties a loop on the end of the rope to make it easier to pull. When the men are being hoisted, after spitting the holes, this loop is hooked on the cage and the rope is thus hoisted away from the blast. Rope bells can be used successfully for 2,200 ft. in depth, but after that, a combination of electric bell and rope bell must be resorted to.

Electric lights are a great aid in shaft sinking, as they leave the men unencumbered with candlesticks, etc. Especially is this true in drilling when the exhaust of the machines makes it hard to keep a light of any kind. When timbering, candles are used to give light back of the timber. An old reel may be used to hold the light wires. A couple of old pieces

shovel handle nailed about 5 ft. apart may be used for the same purpose. To protect the lights from rocks falling from above, a heavy reflector is kept above them. Electric lights are particularly useful where electric

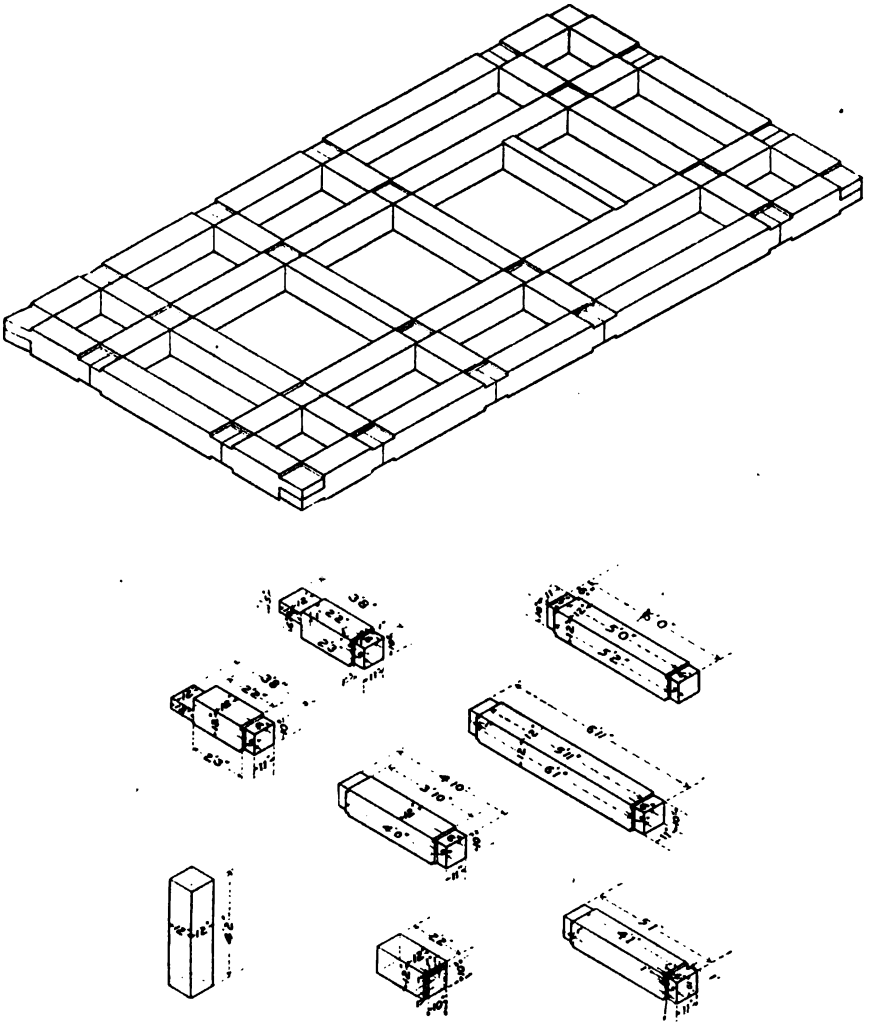

PLATE IX.—JACKET SETS, TRAMWAY MINE.

engines are used in sinking, as they give notice to shaftmen of any interruption in the current.

To ventilate the shaft, 1-in. lining boards are run down, separating two compartments of the shaft. These boards are cut to go between centers, and cleats are nailed on both their sides, at top and bottom, to hold them

in place. In order to make a circulation, a ventilating stack, called an upcast, is placed at the top of one of these compartments running up alongside the gallows frame. Where the shaft is to be sunk under hoisting shafts, the ventilation is much harder and blowers are sometimes used to furnish air. The time it takes smoke to clear away is time lost in shaft sinking. In some places the cages are run up and down through the shaft after blasting to help clear away the smoke.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.
[SUBJECT TO REVISION].

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Butte meeting, Aug. 18 to 21, 1913, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1913, when Vol. XLVI. of the *Transactions* will go to press. Any discussion offered thereafter should preferably be in the form of a new paper for publication in Vol. XLVII. (with suitable cross references in both volumes).

# Hydro-Electric Development in Montana.

BY MAX HEBGEN, BUTTE, MONT.

(Butte Meeting, August, 1913.)

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IT is estimated that the total stationary power now used in the United States, steam, water and gas, is probably over 30,000,000 h. p. The total developed water power is about 6,000,000 h. p. The water power in the United States now economically capable of development probably exceeds 25,000,000 h. p. The total developed and undeveloped power is

FIG. 1.—RELIEF MAP OF MONTANA.

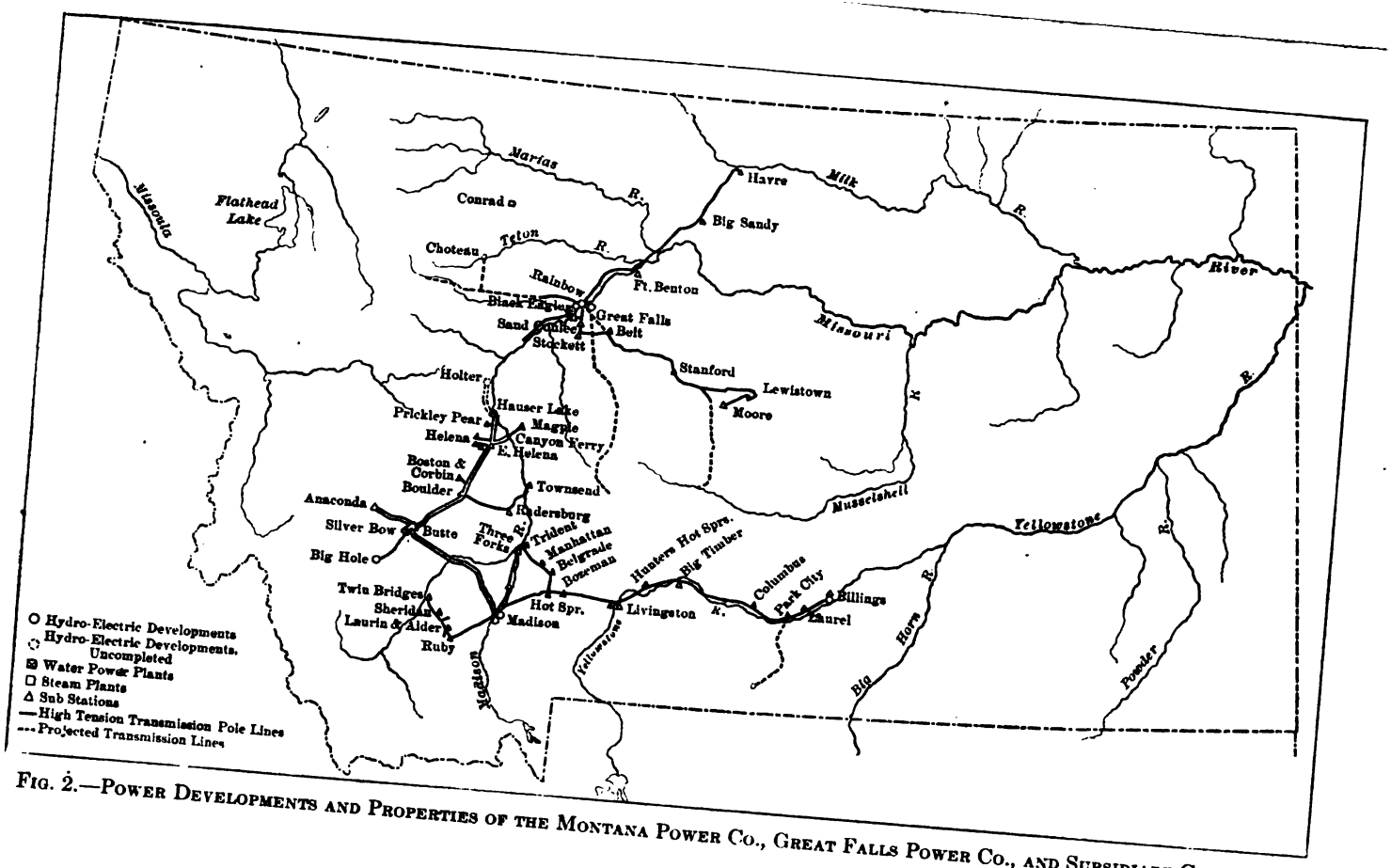


FIG. 2.—POWER DEVELOPMENTS AND PROPERTIES OF THE MONTANA POWER CO., GREAT FALLS POWER CO., AND SUBSIDIARY COMPANIES.

mainly in the far Western States, Lake States and certain Atlantic Seaboard States.

Within the State of Montana the streams rise in the high mountains at an elevation of from 5,000 to 8,000 ft. These streams leave the State line both east and west at elevations from 3,500 to 2,400 ft. It is, therefore, apparent that all these streams have an average fall of 3,000 ft. within the confines of the State, and many opportunities are presented on all streams for the development of power, by taking advantage of the natural fall.

It is probable that 1,000,000 h. p. can be developed in the State of Montana.

## I. Natural Features of State Affecting Power Development.

The main range of the Rocky mountains, forming the Continental Divide, cuts the State of Montana into two parts. The eastern part is drained by the Missouri river and its tributaries, the Madison, Jefferson, Gallatin and Yellowstone rivers. The western part is drained by tributaries to the Columbia river, the Clarks fork and the Kootenai river. The Clarks fork in turn is formed by the junction of Missoula and Flathead rivers, the latter draining Flathead lake.

The western part of the State is mountainous and the eastern part is comparatively level. As a natural result the rivers in the western part are capable of generating large amounts of power, while in the eastern part of the State natural power sites are much fewer and farther apart. It also follows that in the mountainous districts large amounts of power are required for the mining industry, while in the eastern part of the State, which is principally devoted to agriculture, power is required in smaller quantities.

## II. Early Developments.

### *1. Big Hole Plant.*

One of the first power developments in the State was made on the Big Hole river about 22 miles south of Butte. This plant was built in 1899, and has a capacity of 3,000 kw. The development was made by building a rock-filled crib dam across the narrow canyon through which the river flows and developing a head of 65 ft. This plant has been in operation for 14 years and is apparently in as good condition to-day as the day it was built.

It supplies power to the city of Butte over a wooden-pole line operating at 15,000 volts. Power is used for lighting and for the operation of the street railway. Power is also supplied to the pumping plant of the Butte Water Co. situated on the Big Hole river about one mile below

FIG. 3.—BIG HOLE DAM.

This dam is 65 ft. high and the plant has a capacity of 3,000 kw.

the power plant. A large part of the Butte city water supply is furnished by this plant through a wood-stave pipe line.

### 2. *Canyon Ferry Plant.*

This plant, situated on the Missouri river 17 miles from Helena, was built in 1898. A rock-filled timber crib dam, 39 ft. high, was built across the river and apparatus installed for 7,500 kw. Power was first transmitted to Helena at 11,000 volts, and a little later to Butte at 66,000 volts. Power from this plant is used principally to supply the requirements of the mining industry.

### 3. *Madison Plant No. 1.*

This plant was built in 1901. It is situated in the Madison River canyon, 61 miles east of Butte. This development originally consisted of a low crib dam and wooden flume about 1,000 ft. long supplying power to two 1,000-kw. units. The power was transmitted to Butte at 40,000 volts.

FIG. 4.—CANYON FERRY PLANT, 7,500 KW. CAPACITY.
Operates at 60,000 volts.

FIG. 5.—GENERATOR ROOM IN CANYON FERRY PLANT.
Ten 750-kw. generating units operating at 45 ft. head.

### 4. *Black Eagle Plant.*

In 1890 a dam was built across the Missouri river just above the Black Eagle falls, near the city of Great Falls, and power developed at this point to the extent of 8,300 h. p.; 7,500 of this was used directly by the Boston & Montana smelter and the rest was transmitted electrically and used for lighting and street-railway purposes in the city of Great Falls.

FIG. 6.—BLACK EAGLE DAM AND FALLS.

Black Eagle Plant in foreground. Boston & Montana Smelter across river, and highest stack in the world on top of hill.

These four plants supplying power to the cities of Butte, Helena and Great Falls, and being built at the time when the transmission of power by electricity was in its infancy, formed the nucleus of what has become one of the great power systems of the country.

## III. LATER DEVELOPMENTS.

### 1. *Madison River System.*

In 1906 and in the succeeding years great strides were made in power development. The Madison river dam was rebuilt and enlarged, and the Madison No. 2 plant installed with a capacity of 9,000 kw. This plant is notable in that it employs a 10-ft. and a 12-ft. wood-stave pipe 7,400 ft. long. These pipes at the time they were built were the largest pipes of this type ever constructed. The Madison No. 1 plant was rebuilt and used as an auxiliary. The voltage of the system was raised to 46,200

Fig. 7.—Madison River Dam and Pipe Lines.
These pipes at the time of their construction were the largest in the country, one being 10 ft. and the other 12 ft. in diameter; length, 7,400 ft.

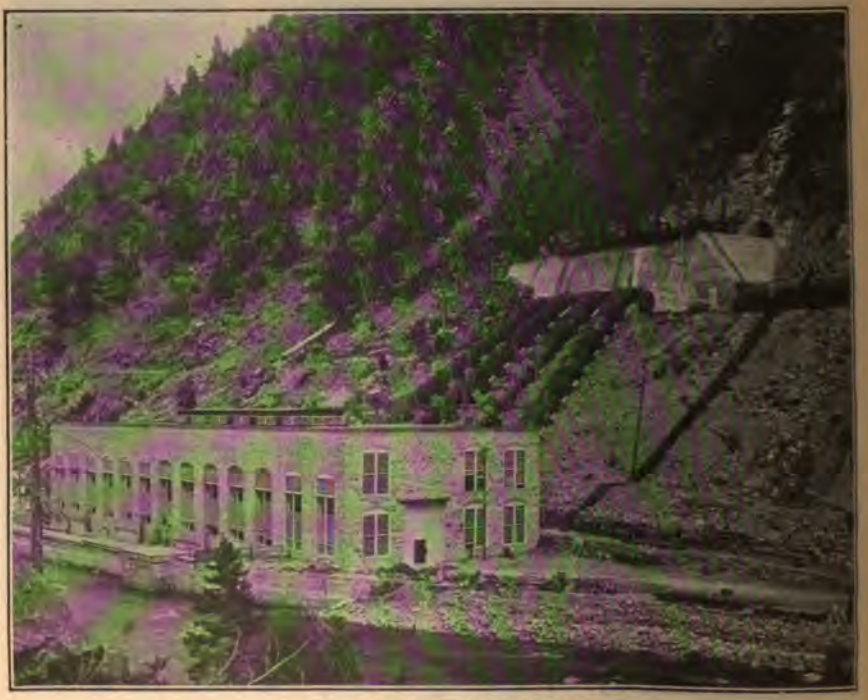

Fig. 8.—Madison No. 2 Plant.
9,000 kw. capacity, 110 ft. head.

and additional transmission lines built to Butte, to the Conrey Placer Mining Co. in Alder Gulch, to the Three Forks cement plant and to the towns of Bozeman, Livingston, and later to Billings.

### 2. *Missouri River System.*

On the Missouri river below Canyon Ferry was constructed the Hauser Lake plant with a 60-ft. dam and a capacity of 14,000 kw. This plant was connected with the transmission lines to Butte, and the lines were extended to supply power to the Washoe smelter at Anaconda. Numerous branches and extensions were also made between Butte and Helena to supply power to various small mines situated in the intervening district.

### 3. *Great Falls System.*

The greatest natural power site in the State, and one of the greatest in the country, is on the Missouri river near the city of Great Falls. In a distance of 8 miles the river falls approximately 400 ft., half of this drop being in abrupt falls, making power development particularly easy and making feasible a total development of about 125,000 kw.

In 1910 the first large development was made at this point. This development was called the Rainbow plant after the falls of the same name. A low diversion dam was built above the falls and the water conducted through two 15 ft. 6 in. steel pipes to a point below the falls where it was utilized at a head of 105 ft. Generating apparatus having a capacity of 21,000 kw. was installed, but upon test it was found that both the water wheels and generators greatly exceeded their original rating, and this plant is now conservatively rated at 25,000 kw.

Power from this plant is transmitted at 6,600 volts to the smelter at Great Falls, and to Butte over duplicate steel-tower transmission lines operating at 100,000 volts. From Butte an additional line is extended to Anaconda so that at present almost the entire capacity of both the Rainbow plant and the Missouri River plants are utilized in the mines of Butte and the smelters at Anaconda and Great Falls. In addition to these main transmission lines additional feeders of smaller capacity have been built from the Rainbow plant to Lewistown, Havre and Cascade.

### 4. *Missoula River Development.*

In 1906 and 1907 a timber-crib dam was built on the Missoula river, six miles above Missoula just below the mouth of the Blackfoot river. A 4,000-h. p. installation was made, and power from this development serves the city of Missoula. A high-tension line from this development is also constructed up the Bitter Root valley and serves the cities of Hamilton, Stevensville and Victor.

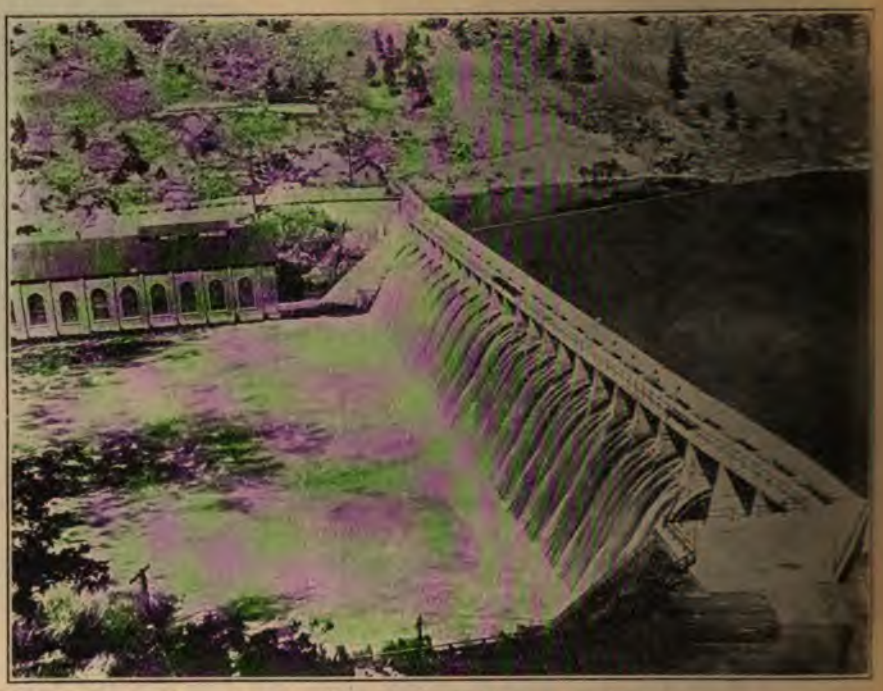

FIG. 9.—HAUSER LAKE PLANT AND DAM.
This is a solid concrete dam 60 ft. high with an adjustable flashboard system for taking care of extreme floods.

FIG. 10.—GENERATOR ROOM IN HAUSER LAKE PLANT.
Five 2,800-kw. units operating at 65 ft. head.

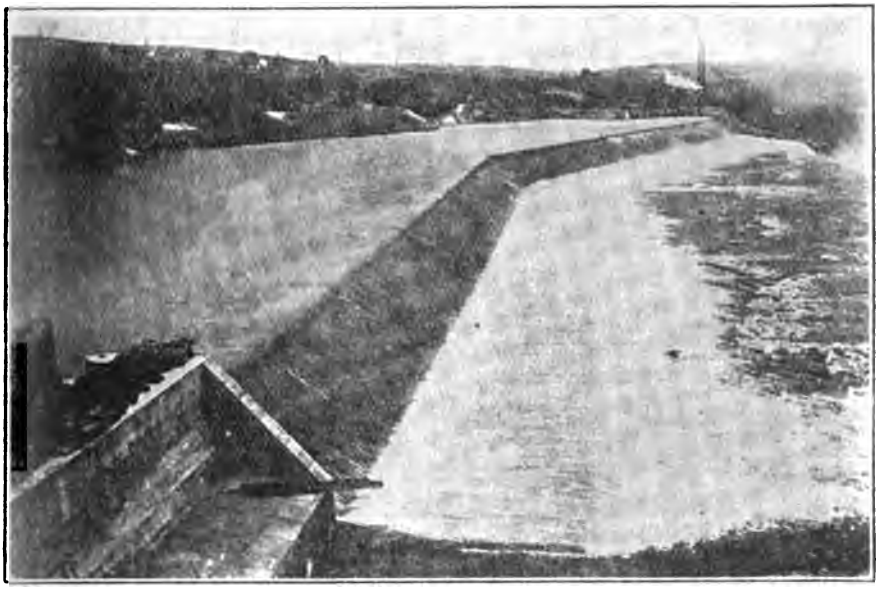

FIG. 11.—RAINBOW DAM SHORTLY AFTER COMPLETION.

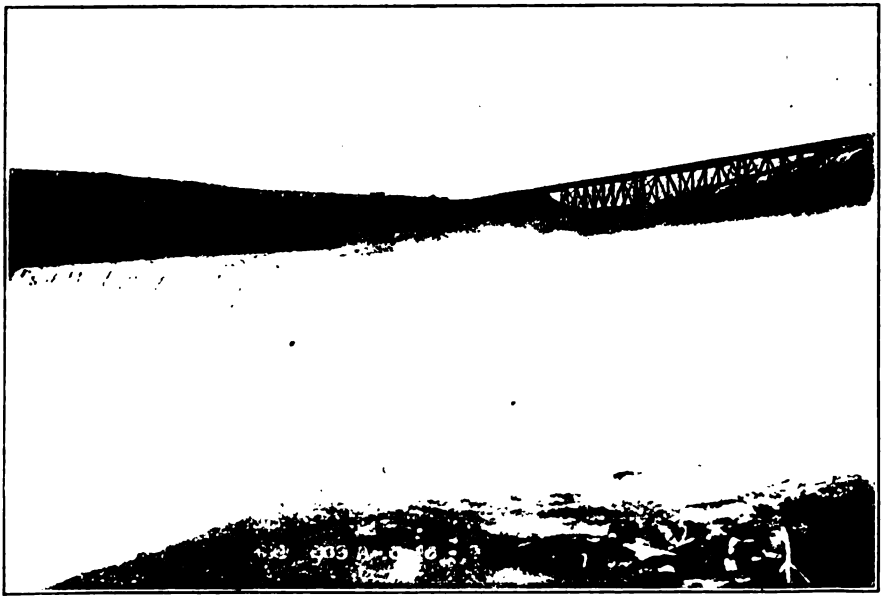

FIG. 12.—RAINBOW FALLS AND DAM DURING HIGH WATER.

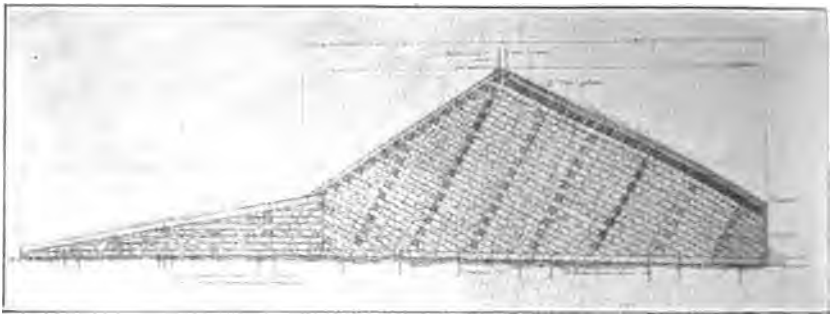

FIG. 13.—CROSS-SECTION OF DAM AT RAINBOW PLANT.

FIG. 14.—RAINBOW PLANT.
21,000 kw. capacity, 105 ft. head, operates at 100,000 volts.

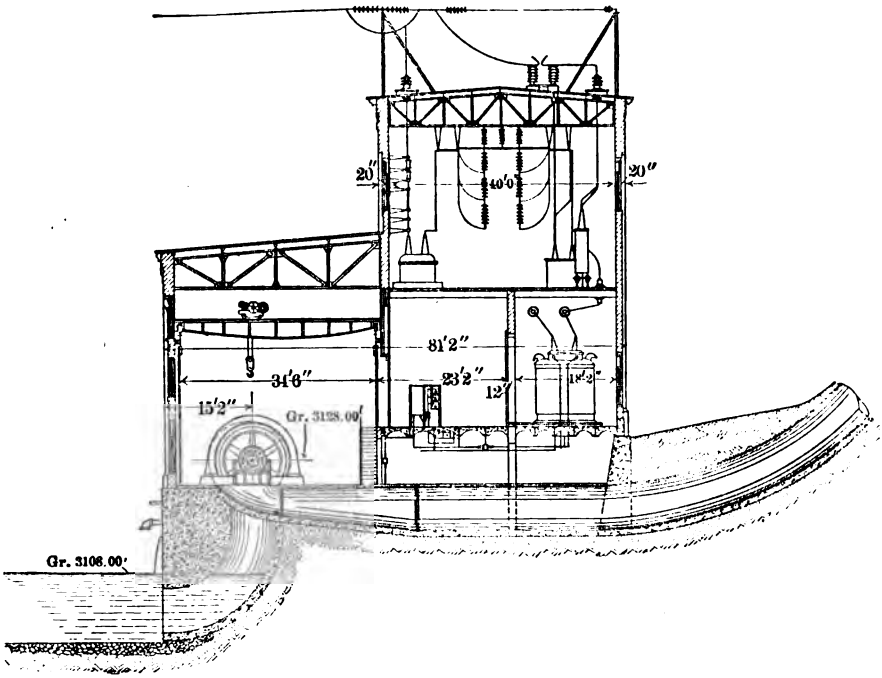


FIG. 15.—CROSS-SECTION OF RAINBOW STATION.

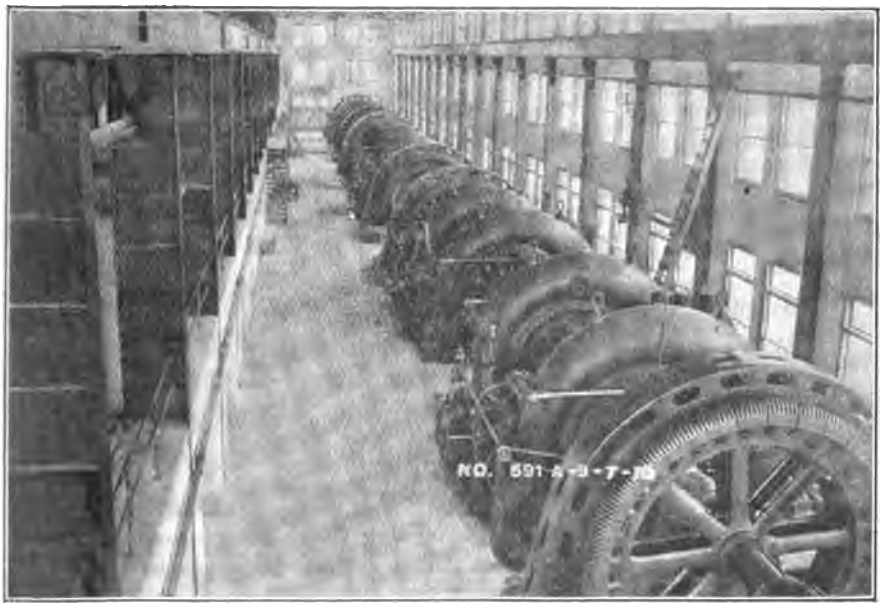

FIG. 16.—GENERATOR ROOM IN RAINBOW PLANT.
Six 4,000-kw. generating units operating at 105 ft. head.

### *5. Big Fork Development.*

A 2,500-h. p. development has for some time been in operation on the Big Fork river in the northwestern part of Montana. Power from this development serves the cities of Kalispell, White Fish, Columbia Falls and intermediate points.

In addition to the developments mentioned above there are a number of small powers scattered throughout the State.

## IV. Present Capacity of Power Developments.

In addition to the plants previously mentioned there are in operation several smaller plants which are not of sufficient capacity to require

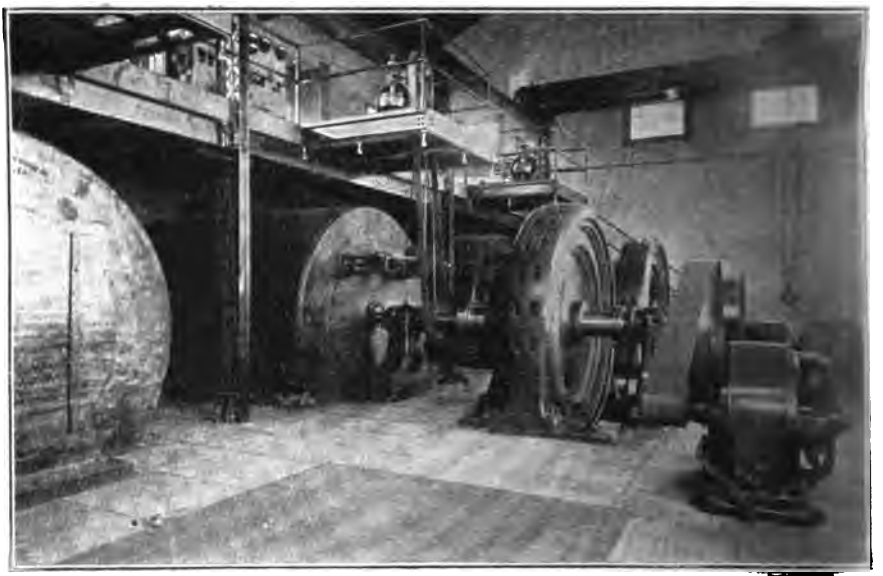

Fig. 17.—Power Plant at Billings.
Three water-wheel driven generators, 360 kw. each.

description. They will, however, be included in the list on the [illegible]ing page, which shows the total normal capacity of all the plants [illegible] are now operating on the combined systems of the Montana Pow[illegible] and the Great Falls Power Co.

## V. Transmission System.

The principal transmission lines of the combined system operate at 46,200, 66,000 and 100,000 volts. The various lines are tied together through transformers and operate as one large distributing network into which power is fed from each of the 11 plants and from which power

| Plant | Generating Capacity Kilowatts |
|---|---|
| Big Hole | 3,000 |
| Billings No. 1 | 1,000 |
| Black Eagle | 1,400 |
| Canyon Ferry | 7,500 |
| Hauser Lake | 14,000 |
| Lewistown No. 1 | 350 |
| Lewistown No. 2 | 100 |
| Livingston | 1,500 |
| Madison River No. 1 | 2,000 |
| Madison River No. 2 | 9,000 |
| Prospect Creek | 750 |
| Rainbow | 25,000 |
| Missoula River Development | 4,000 |
| Big Fork Development | 2,500 |
| Total | 72,100 |

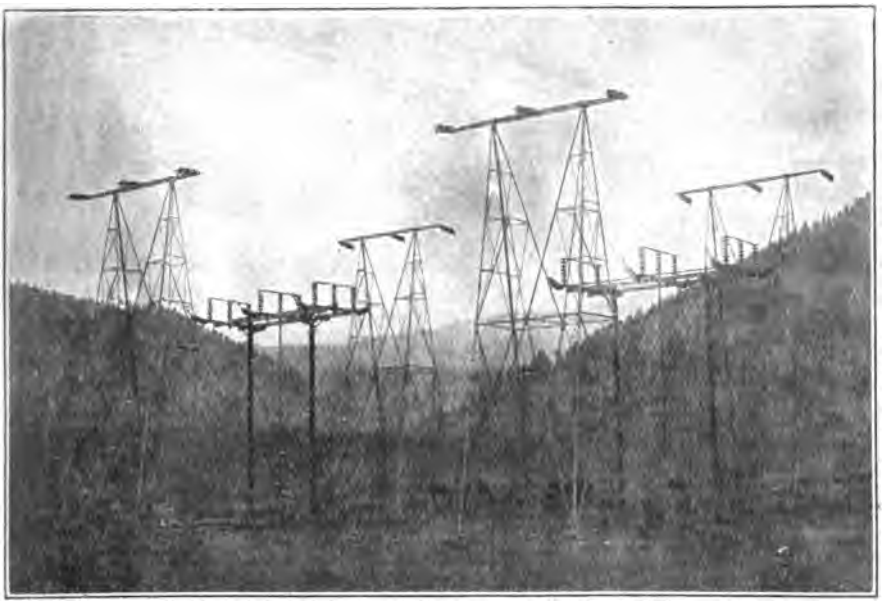

Fig. 18.—Sectionalizing Switches on Great Falls-Butte 100,000-volt Transmission Line.

These switches are used to cut the line into short sections to facilitate the location of trouble.

is taken by about 46 substations, varying in size from the Great Falls Power Co.'s substation at Butte having a capacity of 21,000 kw. down to the lowest standard high-voltage substation of 250 kw.

There is at present in operation a total of 1,300 miles of transmission line of which 340 miles is steel-tower line and the rest is wood-pole line. This transmission system covers a territory approximately 200 miles square.

## VI. Character of Load.

### *1. Lighting and Small Power Uses.*

From this one system, power is supplied for lighting 40 cities and towns in Montana. In addition to lighting, power enters into almost every phase of the industrial life of all these cities and towns. The domestic water supply in all the larger cities of the State is pumped by electric power. Something over 10,000 flat-irons are in use in the homes of the people. Cooking by electricity is becoming popular. Over 500 electric cooking stoves are in daily use. Electrically driven sewing machines and washing machines are in every-day use, and many other devices which tend to lighten the duties of the housewife are operated by electric power.

### *2. Street Railways.*

The street-railway systems of the cities of Butte, Great Falls, Missoula, Helena, Anaconda, Billings and the Gallatin Valley Railway system all receive power from the transmission lines within the State.

### *3. Mining.*

It is admitted that the average cost of steam power for mining purposes in the State of Montana is $125 per horse power per year. In many isolated mining districts this figure rises as high as $150 and even $200 per horse power per year. Electric power is purchased at $50 per horse-power-year in comparatively small quantities, and in large installations prices for power can be obtained as low as $35 per horse-power-year. The advent of electric power in mining districts of Montana has meant that many a property which formerly operated at a loss on a steam basis is now making a profit by the use of electric power.

Although the lighting, street railway and other miscellaneous industries form a very important and necessary part of the load of the power companies, yet by far the greatest amount of power which is used in Montana is devoted to mining and smelting; in fact, it may be said that

the mining industry and the power industry have developed together, each to a large extent making the other possible.

The extensive use of power for mining makes commercially feasible the development of water power in large amounts and warrants the building of high-capacity, long-distance transmission lines.

This development of power on a large scale is the principal factor in

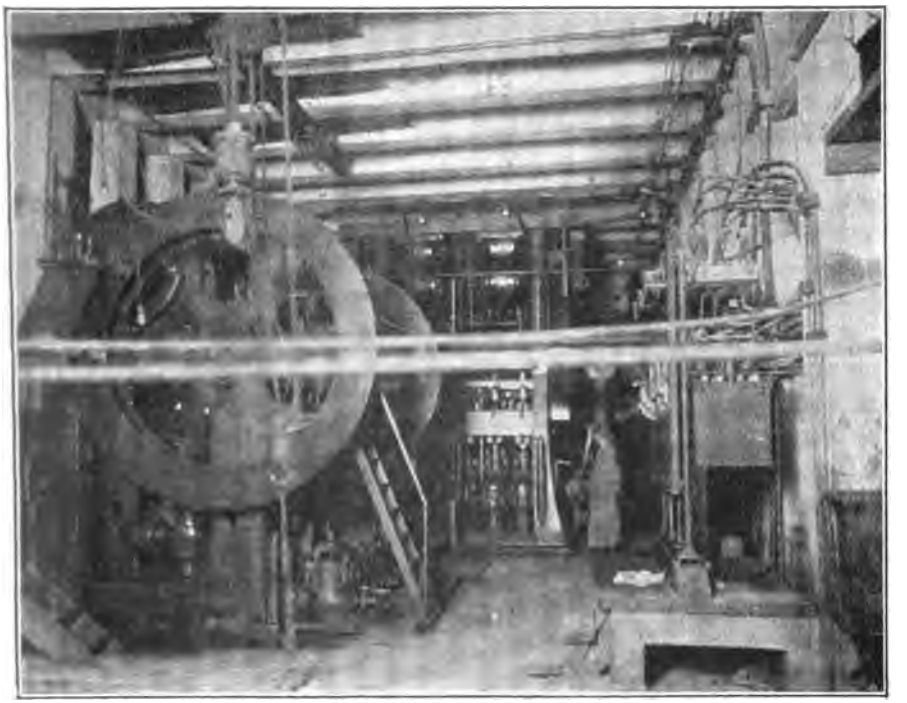

FIG. 19.—1,200-FT. LEVEL UNDERGROUND PUMPING STATION AT LEONARD MINE.

making its cost low. The low cost of power and the low price at which it is sold stimulate its use for all the miscellaneous purposes to which it is adapted. Thus the immense power possibilities of the Missouri river at Great Falls lay dormant until the demand for power in the mines at Butte made it feasible to install a large development and transmission system.

The development having once been established, it proved feasible to continue the extension of lines in all directions from Great Falls, and over these lines power is now supplied for lighting the towns of Havre, Lewistown, Cascade, and a large number of smaller points for operating flour mills, irrigating pumping plants and the coal mines in the Sand Coulee and Stockett district. For this latter purpose alone about 1,200 kw.

are being used; thus it is seen that cheap power in the northern part of the State is made possible by the mining industry in the southern part.

On the other hand the Madison River plant was originally built to supply power in Butte, principally for lighting. The installation of this plant, however, made it possible to deliver power to the Conrey Placer

FIG. 20.—COMPRESSOR PLANT AT LEONARD MINE.
Four 600-h.p. motors driving air-compressors.

Mining Co. in Alder Gulch, and made economical the operation of its numerous dredges where the operation by steam was practically prohibitive on account of the high cost of coal and increased amount of labor necessary.

The total amount of power used for mining amounts to 24,000 kw. and for milling and smelting 20,000, a total of 44,000 kw. for the mining industry as a whole. Most of this power is used in Butte, Anaconda and Great Falls.

The mines in Butte use power principally for operating compressors, for hoisting and for pumping. The compressor and pumping load has naturally a high load factor, and therefore forms a desirable load for the power company. The hoisting load has such an extremely low factor

that until very recently it was not considered worth while to operate this load with electric power. To overcome this objection, however, a comprehensive compressed-air system of hoisting has been installed embracing most of the principal mines. Compressed air is supplied to all these mines from a central compressor station which operates in connec-

FIG. 21.—HEAD FRAME AT MOUNTAIN VIEW MINE.
Hoisting at this mine is now being done by compressed air from the electrically operated central compressor plant.

tion with a large storage system. The storage system consists of a number of air receivers situated about 200 ft. below the top of the hill upon which stands the compressor plant. At the top of the hill is a water reservoir which is connected to the bottom of the air receivers so that a uniform hydrostatic head equivalent to 90 lb. pressure is maintained on the receivers, and the entire capacity of the receivers can thus be used at full normal pressure, the air being displaced by the water as it is required. When the combined demand of all the hoisting engines exceeds the supply from the compressor plant the excess air is drawn from the storage tanks and water runs down the hill into the tanks to take its place. When the demand is less than the supply from the compressor plant the excess air is forced into the storage tanks, and in turn forces the water back up the hill. In case of a temporary failure of power sufficient storage

is provided to allow the continued operation of the hoists for a sufficient period of time to cover all emergencies.

This compressed-air hoisting system was installed after various electric systems had been considered, and has the following advantages:

First. The total cost of installation was lower than the estimated cost of any other system. This was partly due to the fact that the steam hoists which were already installed were used in connection with the air system, it being necessary to install new cylinders and valve mechanism only.

Second. A larger power-storage capacity is provided than could be economically provided with a straight electric system.

Third. The central compressor plant is operated by synchronous motors, and both the power factor and load factor can be maintained at practically 100 per cent.

Fourth. Excess air from the compressor plant is used to great advantage for supplying the drill system in the mines.

This central compressor plant has a capacity of 7,200 h. p., and it is planned to operate hoisting engines for an aggregate capacity of about 40,000 h. p. That this can be accomplished is due to the fact that each hoisting engine has a load factor of about 5 per cent. and the diversity factor of all the engines taken together is correspondingly low; in other words, when one engine is operating at full load, another may be operating at part load, and several others will be standing idle. To make the combined load as uniform as possible the hoisting schedules are arranged so that ore is hoisted from the different mines at different times of day.

### *4. Railroads.*

The electrification of the Butte, Anaconda & Pacific Ry. is nearing completion. The substation at Anaconda is in operation and locomotives are working successfully at the Anaconda end of the line.

This road has a 26-mile main-line track from Butte to Anaconda and a total of 90.36 miles, the excess being made up in yards and sidings.

For this electrification, 2,400-volt direct current has been adopted, this being the highest direct current voltage thus far used for this class of work. The use of this high-voltage direct current allows advantage to be taken of the light weight and superior operating characteristics of the direct-current motor and at the same time reduces the amount of current to be collected to a point where it can easily be handled by a single pantograph trolley, it also allows placing of substations long distances apart.

This installation will have only two substations, one at Butte and one at Anaconda. Each substation will be equipped with two synchronous-motor generator sets of 1,000-kw. capacity each, giving a total capacity

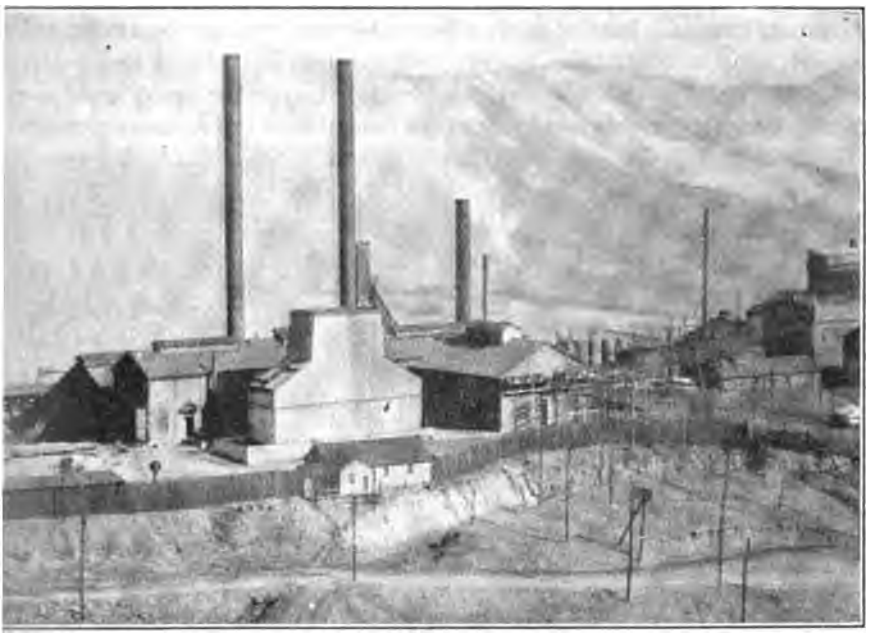

FIG. 22.—60,000-VOLT SUBSTATION AND AUXILIARY STEAM PLANT AT BUTTE.
The substation has a capacity of 10,000 kw. and the steam plant 5,000 kw. Power is delivered to this station from the Canyon Ferry and Hauser Lake Plants.

of 4,000 kw. to the entire direct-current generating equipment for the road. The generators have, however, a short time overload capacity of 200 per cent., so that 12,000 kw. may be supplied as a maximum.

The locomotive equipment will consist of 17 75-ton units. Two of these units will be operated together for heavy freight service and one unit alone used for lighter freight and passenger service.

The Chicago, Milwaukee & St. Paul Railway Co. has entered into contracts with the power companies in Montana to supply power for the electrification of its line from Harlowtown, Mont., to Avery, Idaho, a distance of 439 miles of main line with considerable excess in sidings. Work on this electrification will probably start within the next year or two, and about 25,000 kw. will probably be required.

It appears to be only a matter of time until all the transcontinental railways, in their mountain divisions especially, will find it necessary to turn to electrical operation. This method of operation not only has the advantage of making large savings in operating expenses on account of the elimination of coal and the greatly reduced maintenance expense of electric locomotives as compared with steam locomotives, but also on account of the fact that with the increase in traffic the capacity of the whole line is limited by the tonnage which can be hauled over the steep

mountain grades. Inasmuch as electric locomotives can be made with a much greater continuous power capacity than any steam locomotives yet built, it follows that trains can be hauled over the heavy grades at greatly increased speeds and the tonnage hauled over the division increased

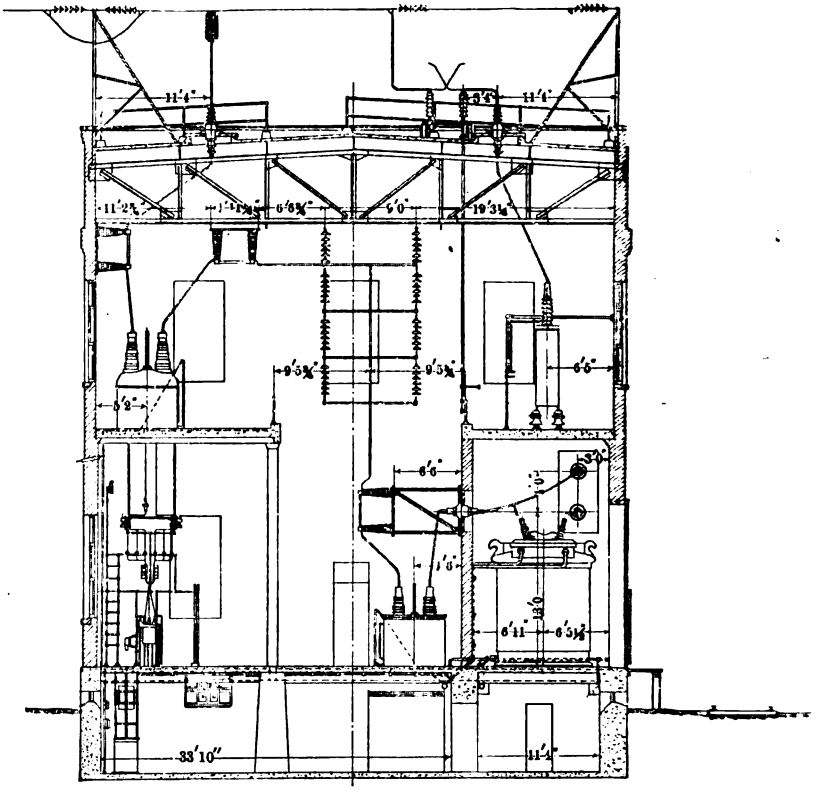


FIG. 23.—CROSS-SECTION OF 100,000-VOLT SUBSTATION AT BUTTE.
Present capacity 21,000 kw.

accordingly, while to get a corresponding increase with steam operation would require double tracking the road.

### 5. *Irrigation.*

The average value of the agricultural land in Montana is increased tenfold by irrigation. Up to the present time irrigation has been practiced for the most part only where the natural conditions were such that it was easy to get water on the land by gravity ditches. Some of the best land in the State, however, is so situated that it is impossible to supply it with water by gravity and pumping must be resorted to, in fact in a great many instances it is cheaper to pump the water than to pay fixed charges

n the necessary canals, ditches and pipe lines required to supply the water y gravity.

During the past year the first large pumping plant for irrigation was istalled in the Prickly Pear valley near Helena by the Montana Reser-

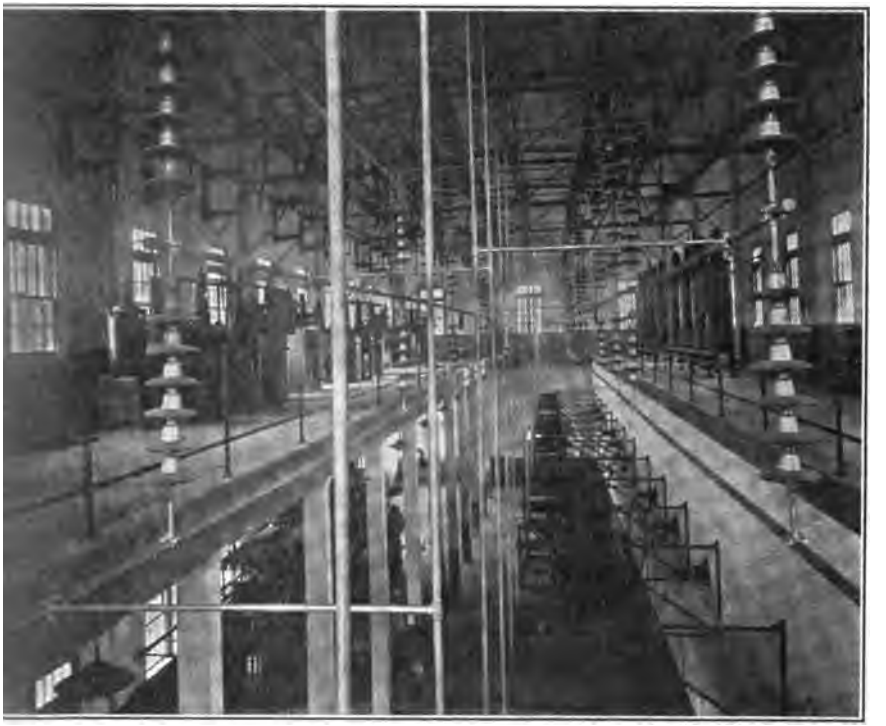

FIG. 24.—BUSBARS AND SWITCHES IN 100,000-VOLT SUBSTATION AT BUTTE. The high tension connections in this station are made of ordinary iron pipe.

oir & Irrigation Co. This plant during the present season will supply vater to 5,000 acres, and will require 1,800 h. p. for its operation. This lant forms the first unit of the Prickly Pear irrigation project, and as ast as the land becomes settled two additional units of about the same ize as the first will be installed, making a total of 5,400 h. p. power for his one project.

Near the city of Billings the Lockwood irrigation project is being es-ablished with a pumping plant of 500 h. p.

In the Sun River valley, west of Great Falls, the government has tarted work on what will be the largest irrigation project in the State. A gravity system for supplying water will be used, but for building the lams and canals the United States has contracted with the Great Falls Power Co. for a maximum of 3,484 h.p. for five years.

To supply water during the irrigation season the Montana Reser & Irrigation Co. is building a dam to form a storage reservoir near the head waters of the Madison River close to Yellowstone Park. This reservoir will have a capacity of 15,000,000,000 cu. ft., which will be sufficient to store the entire spring and summer flood of the Madison river. The water will be available for use throughout the entire length of the Madison river and in the Missouri river down to Great Falls, or even farther. The stored water will be used in two ways: First, for pumping on to land for irrigation, and second, for the generation of additional power in the present power plants, this power to be used for operating pumps. In this way the water is made to pump itself into the land.

## VII. Operation of System.

The operation of the power system composed of many plants operating in parallel is somewhat more complicated than the operation of a system having only one generating system. There are also many advantages in the operation of such a combined system which cannot be realized in the operation of a single plant. With a single plant the maximum amount of power which can be sold is ordinarily limited by the amount which can be generated at the time of minimum flow of the river upon which the plant is situated. This minimum flow can be increased by the building of storage reservoirs, but storage reservoirs are expensive and in many instances would not pay where the water is to be used by a single plant.

In the case of the power system in Montana the Hebgen reservoir of the Montana Reservoir & Irrigation Co., previously referred to, is so situated that its stored water will be utilized in turn by the two Madison River plants, by the Canyon Ferry and Hauser Lake plants and by the three Great Falls plants, being a total of seven plants, each of which will derive full benefit from the storage.

It also occurs in the development of water power that some plants are well adapted to operate on a variable load while others can only operate to advantage on a uniform load. Thus a development in which the head is produced by means of a dam ordinarily has a large pond from which large quantities of excess water can be drawn for short periods of time to supply a temporary demand for power much in excess of the average plant output. This type of development ordinarily employs very short pipe lines so that the cost of water conduits for this excess use of water is not great. This type of development when connected to a large system is called a peak-load plant.

Another type of development consists of a low diversion dam and long flume or pipe line which conducts the water to a point further down

the river where the plant is situated. With this type of development the largest item of cost is the pipe line, and inasmuch as the low diversion dam creates very little storage, and as the cost of development would be greatly increased if the pipe lines were made large enough to carry more than the normal flow of the river, this type of development is best adapted to operate at a uniform and steady load. A development of this kind is called a base-load plant.

A base-load plant operating alone to supply power for average requirements must necessarily waste a large quantity of water on account of the fact that due to the variable demand for power it cannot operate at full load continuously, and during the periods when the demand for power is light, water is wasted over the dam. On the other hand, the average peak-load plant will have a greater storage capacity than can be well utilized by the usual variations in load, which means that a certain development if it were to operate alone could be made for, say, 10,000 kw., while it might, if operated in connection with a base-load plant, be made for a development of 15,000 kw. merely by the addition of extra generators and water wheel. In this type of development, where the dam is usually the greatest item of cost, the additional power obtained only by the addition of generating units is obtained at a very low cost for the additional horse power.

Where one plant is the sole source of supply for a system every precaution must be taken to maintain continuity of service. A large number of generating units must be installed so that if one unit is out of commission for repairs the station will be reduced only a small percentage. Where the system is supplied by many plants the shutting down of a unit in any one plant is of less consequence as it reduces the capacity of the system in a percentage corresponding to the ratio between the capacity of one unit and the capacity of the entire system. For this reason the generating units in such a plant can be made larger, which will result both in a lower first cost and a higher efficiency.

When plants are situated on different rivers, which take their water from different water sheds, the minimum flow on one river will not correspond in time with that on another. For this reason when one plant has its output reduced by extremely low water another plant will have plenty of water and will be able to carry a part of the first plant's load. Also, even with plants at different places on the same river, one plant may have its output temporarily reduced by ice troubles while another plant farther down the river will be free from such trouble, and can carry an extra share of load to make up for the deficiency at the first plant.

The balancing of the variations in river flow among different plants will be carried on with great advantage between the plants on the Missouri river and on the Clarks fork. These rivers flow on opposite sides of the

Continental Divide, in parts of the State, which, although only 200 miles apart, have very different climates. The variations in the flow of the two rivers are, therefore, such that with the two developments tied together by long-distance lines power can be continually shifted back and forth from one system to the other, allowing each development to utilize the total flow of its river to the best advantage.

It is estimated that the advantage to be derived from the combined operation of many plants due to the causes above enumerated amounts to at least 50 per cent. in the economical utilization of the total available power.

To give an idea of the value of the power which is being generated by the plants of the system under consideration, which are now producing 49,500 kw., it may be stated that to produce this amount of power from coal would require the yearly consumption of 750,000 tons; at the average price of $4, which would apply at the points where the power is used, this would amount to $3,000,000 per year.

When it is remembered that this value in coal is, by means of water power, being saved for future generations, and that during the next 10 years this saving will in all probability be increased about six or seven times, it is not difficult to appreciate that true conservation consists in encouraging rather than restricting the free development of water power.

## VIII. Increase in Use of Power.

The use of electric power is increasing by leaps and bounds. The power companies must maintain a sufficient reserve capacity in power plants to accommodate increases in the demand for power as they occur. It becomes a difficult matter to predict far enough in advance the increase in load which will have to be taken care of. Such a prediction can be based on little else than past experience.

## IX. New Power Developments.

To take care of the general increase in load, especially the Chicago, Milwaukee & St. Paul railway electrification, work has been commenced on two new, large developments, one at the Great falls of the Missouri and one at Thompson falls on Clarks fork. The Great Falls development will have a capacity of 60,000 kw. and the Thompson Falls development a capacity of 30,000 kw.

The Great Falls development is situated about 8 miles down the river from the city of Great Falls, and utilizes a natural fall in the river of about 85 ft., which fall will be increased to a total of 150 ft., by the erection of a solid masonry dam immediately above the falls. The dam will have a crest line 1,000 ft. long and will have an overflow section design-

to take care of the maximum flood of the river, estimated at 100,000 sec.-ft. Generating equipment will be installed to utilize the normal flow of 6,000 sec.-ft. in six vertical single-runner water-wheel units of 10,000-kw. capacity each. To illustrate the high efficiency which can be obtained in a modern water wheel of this type, it may be stated that manufacturers have promised an efficiency of 90 per cent. for these wheels, and this efficiency has even been exceeded by similar wheels which have already been built. The power plant will be built on a rock ledge just below the falls and will take its water through six short steel pipes leading directly to the dam.

The Great Falls development will supply power to the territory covered by the present system and to the Chicago, Milwaukee & St. Paul railway from Harlowtown to Deer Lodge.

The Thompson Falls development will be similar to the Great Falls development in that approximately one-half the head exists in the form of a natural fall and the other half will be developed by a dam immediately above this fall. In the case of Thompson falls, however, the plant will be situated some little distance below the falls and the water conducted thereto through an old dry channel which will be flooded when the dam is built. The design of generating units and the general plan of the power house will be similar to that at Great Falls. A total head of about 50 ft. will be utilized.

The Thompson Falls development will supply power to the territory west of that which is covered at present and to the Chicago, Milwaukee & St. Paul railway from Deer Lodge to Avery, Idaho.

The addition of these two plants will more than double the present capacity of the system, but it is estimated that this increase will be absorbed within a very short time after the completion of the plants, which will require two or three years.

## X. Undeveloped Powers.

In addition to the plants already built and under construction there are a large number of power sites standing ready to be developed as demands for power continue to increase. Although these developments are situated in all parts of the State, the larger possibilities exist in the northwestern part of the State where developments like the Flathead lake, Kootenai falls and other large streams present possibilities for the development of at least 250,000 h. p.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Butte meeting, Aug. 18 to 21, 1913, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1913, when Vol. XLVI. of the *Transactions* will go to press. Any discussion offered thereafter should preferably be in the form of a new paper for publication in Vol. XLVII. (with suitable cross references in both volumes).

# The Great Falls Flue System and Chimney.

BY C. W. GOODALE, BUTTE, MONT., AND J. H. KLEPINGER, GREAT FALLS, MONT.

(Butte Meeting, August, 1913.)

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## I. INTRODUCTION.

In the summer of 1909 the Boston & Montana reduction department of the Anaconda Copper Mining Co. completed a new flue system, at a cost of about $1,100,000, and as this includes the largest and highest chimney in the world, and some new features of flue construction, a description will be of interest.

At the Spokane meeting of the Institute in September, 1909, a brief mention was made of this flue system, in discussing Mr. Brunton's presidential paper on the Progress of Metallurgy in the West, and a promise was given that when a long enough time had elapsed, so that operating results could be given, a paper would be offered.

The smelting plant includes blast furnaces, MacDougall roasters, gas-fired reverberatories, and converters.

## II. Character of the Ore.

The ore treated comes from Butte, where two classes are made in mining, the first class going to the blast furnaces, and the second to the concentrator. Partial analyses follow:

| | Per Cent. Cu | Oz. Ag | Oz. Au | Per Cent. $SiO_2$ | Per Cent. Fe | Per Cent. S | Per Cent. $Al_2O_3$ | Per Cent. Cu |
|---|---|---|---|---|---|---|---|---|
| First class..... | 6.06 | 2.0 | 0.015 | 51.2 | 13.6 | 17.3 | 8.1 | 0.30 |
| Second class..... | 3.55 | 1.26 | 0.008 | 58.5 | 9.4 | 11.6 | 11.7 | 0.10 |

The copper-bearing minerals include chalcocite, enargite, bornite, covellite, tetrahedrite, tennantite, and chalcopyrite,—but the most abundant is chalcocite. Sphalerite and galenite also are found, and in some of the Butte mines these minerals occur in considerable quantities. Pyrite is a very abundant mineral, ranging from 20 per cent. of the weight of second-class ore to 30 per cent. of the first class. The principal country rock of the Butte district is quartz-monzonite, and the gangue of the copper-ore deposits is made up of quartz and highly altered monzonite.

The following analysis will show the composition of the ore and of the unaltered country rock:

| | Ore Per Cent. | | Quartz-Monzonite Per Cent. |
|---|---|---|---|
| Cu..... | 3.25 | | ..... |
| As..... | 0.37 | | ..... |
| Sb..... | 0.025 | | ..... |
| S..... | 11.12 | | ..... |
| Fe..... | 9.30 | $Fe_2O_3$ | 1.96 |
| | | FeO | 2.83 |
| $SiO_2$..... | 58.45 | | 64.03 |
| $Al_2O_3$..... | 11.90 | | 15.58 |
| CaO..... | 0.15 | | 4.20 |
| MgO..... | 0.25 | | 2.15 |
| $K_2O$..... | 2.34 | | 4.11 |
| $Na_2O$..... | 0.11 | | 2.76 |
| $TiO_2$..... | ..... | | 0.60 |
| $P_2O_5$..... | ..... | | 0.18 |
| MnO..... | ..... | | 0.11 |
| BaO..... | ..... | | 0.07 |
| SrO..... | ..... | | 0.04 |
| $H_2O$ below 110° C..... | 0.02 | | 0.20 |
| $H_2O$ above 110° C..... | 0.15 | | 0.73 |

The sample of ore taken for this analysis covers about 90,000 tons second-class or concentrating ore, received in Great Falls during the month of March, 1913, from the Mountain View, Pennsylvania, Leonard and

[1] Walter Harvey Weed, Geology and Ore Deposits of Butte, Mont., *Professional Paper No. 74, U. S. Geological Survey*, p. 35 (1912).

Tramway mines, and may therefore be taken as representative in character. Leaving out the copper, arsenic, antimony, sulphur and iron, we have:

| | *Per Cent.* |
|---|---|
| $SiO_2$ | 58.45 |
| $Al_2O_3$ | 11.90 |
| CaO | 0.15 |
| MgO | 0.25 |
| $K_2O$ | 2.34 |
| $Na_2O$ | 0.11 |
| $H_2O$ above 110° C | 0.15 |
| | 73.35 |

From this it is seen that the gangue minerals of the ore amount to 73.35 per cent. of its weight. Recalculating this as 100 per cent., we have the following approximate composition of the gangue:

| | *Per Cent.* |
|---|---|
| $SiO_2$ | 79.70 |
| $Al_2O_3$ | 16.22 |
| CaO | 0.20 |
| MgO | 0.34 |
| $K_2O$ | 3.19 |
| $Na_2O$ | 0.15 |
| $H_2O$ above 110° C | 0.20 |
| | 100.00 |

From the concentrator the product of the coarse jigs, amounting to about one-third of the entire output of the mill, goes to the blast furnaces, but the fine concentrate goes to MacDougall roasters, and thence into the reverberatories. Partial analyses of these products are given below:

| | Per Cent. Cu | Oz. per Ton | | Per Cent. $SiO_2$ | Per Cent. Fe | Per Cent. S | Per Cent. $Al_2O_3$ | Per Cent. CaO |
|---|---|---|---|---|---|---|---|---|
| | | Ag | Au | | | | | |
| Coarse concentrate | 12.16 | 3.9 | 0.019 | 20.7 | 25.9 | 33.8 | 4.4 | 0.3 |
| Fine concentrate.. | 8.84 | 3.0 | 0.021 | 18.1 | 28.5 | 35.9 | 5.3 | 0.3 |

Tables I and II give the results of screen tests on first-class ore, on fine concentrate, and on flue dust, from the main flue chamber.

The test on the flue dust shows that all of it is finer than 0.5 mm., and when we consider the tests on the ore and find that from 7 to 10 per cent. is less than 0.5 mm. in size, it is apparent that much fine material would inevitably pass out of the blast furnaces into the flues.

In the case of the MacDougall roasters, a considerable production of flue dust would naturally be expected, as they treat fine concentrate, 55 per cent. of which will pass through a screen with 0.5 mm. holes.

TABLE I.—*Sizing-Test Results on First-Class Ore.*

| Screen Sizes mm.—Mesh | Pennsylvania Mine | | | Mountain View Mine | | | West Colusa Mine | | |
|---|---|---|---|---|---|---|---|---|---|
| | Individual Per Cent. | Cumulative Per Cent. | Assay Cumulative Individual Per Cent. | Individual Per Cent. | Cumulative Per Cent. | Assay Cumulative Individual Per Cent. | Individual Per Cent. | Cumulative Per Cent. | Assay Cu Individual Per Cent. |
| On 50.8 | 42.80 | 42.80 | 4.78 | 33.09 | 33.09 | 5.89 | 45.00 | 45.00 | 8.17 |
| On 45.3 | 1.75 | 44.55 | 4.58 | 1.39 | 34.48 | 3.42 | 1.34 | 46.34 | 7.38 |
| On 32.0 | 7.97 | 52.52 | 4.30 | 7.53 | 42.01 | 4.88 | 6.42 | 52.76 | 8.44 |
| On 22.6 | 7.78 | 60.30 | 4.15 | 8.17 | 50.18 | 4.84 | 6.48 | 49.24 | 8.09 |
| On 16.0 | 5.82 | 66.12 | 4.21 | 6.63 | 56.81 | 5.25 | 5.06 | 64.30 | 8.88 |
| On 9.51 | 7.13 | 73.25 | 4.18 | 8.36 | 65.17 | 5.36 | 6.24 | 70.54 | 9.07 |
| On 5.66 | 5.21 | 78.46 | 4.38 | 6.30 | 71.47 | 5.27 | 4.74 | 75.28 | 9.57 |
| On 4.00 | 2.99 | 81.45 | 4.45 | 3.78 | 75.25 | 5.17 | 2.98 | 78.26 | 9.75 |
| On 2.38 | 3.28 | 84.73 | 4.63 | 4.24 | 79.49 | 5.11 | 3.56 | 81.82 | 9.53 |
| On 1.41 | 3.22 | 87.95 | 4.32 | 4.12 | 83.61 | 4.64 | 3.85 | 85.67 | 8.59 |
| On 0.841 | 2.81 | 90.76 | 4.11 | 3.70 | 87.31 | 4.53 | 3.64 | 89.31 | 9.02 |
| On 0.500 | 2.24 | 93.00 | 4.12 | 2.92 | 90.23 | 4.65 | 2.89 | 92.20 | 8.87 |
| On 0.350—50 | 0.41 | 93.41 | 4.49 | 0.74 | 90.97 | 5.10 | 0.61 | 92.81 | 9.48 |
| On 0.166—90 | 2.12 | 95.53 | 4.88 | 2.64 | 93.61 | 5.66 | 2.19 | 95.00 | 11.15 |
| On 0.070—200 | 1.05 | 96.58 | 5.70 | 1.70 | 95.31 | 6.81 | 1.35 | 96.35 | 12.15 |
| Through 0.070—200 | 3.42 | 100.00 | 3.89 | 4.69 | 100.00 | 4.92 | 3.65 | 100.00 | 7.93 |
| TOTAL | 100.00 | ..... | 4.51 | 100.00 | ..... | 5.32 | 100.00 | ..... | 8.62 |

TABLE II.—*Sizing-Test Results on Fine Concentrates Fed to the MacDougall Roasting Furnaces and on Main-Chamber Flue Dust.*

| Screen Sizes mm.—Mesh | Fine Concentrates to MacDougalls | | | Main-chamber Dust | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| | | | | South End of Chamber | | | North End of Chamber | | |
| | Individual Per Cent. | Cumulative Per Cent. | Assay Cu Individual Per Cent. | Individual Per Cent. | Cumulative Per Cent. | Assay Cu Individual Per Cent. | Individual Per Cent. | Cumulative Per Cent. | Assay Cu Individual Per Cent. |
| On 2.83 | 9.6 | 9.6 | 13.85 | ..... | ..... | ..... | ..... | ..... | ..... |
| On 2.38 | 1.5 | 11.1 | 14.10 | ..... | ..... | ..... | ..... | ..... | ..... |
| On 2.00 | 0.9 | 12.0 | 12.50 | ..... | ..... | ..... | ..... | ..... | ..... |
| On 1.68 | 3.4 | 15.4 | 10.85 | ..... | ..... | ..... | ..... | ..... | ..... |
| On 1.41 | 3.3 | 18.7 | 9.50 | ..... | ..... | ..... | ..... | ..... | ..... |
| On 1.19 | 4.0 | 22.7 | 8.80 | ..... | ..... | ..... | ..... | ..... | ..... |
| On 1.00 | 2.0 | 24.7 | 7.70 | ..... | ..... | ..... | ..... | ..... | ..... |
| On 0.841 | 6.6 | 31.3 | 7.40 | ..... | ..... | ..... | ..... | ..... | ..... |
| On 0.707 | 5.1 | 36.4 | 6.85 | ..... | ..... | ..... | ..... | ..... | ..... |
| On 0.595 | 2.8 | 39.2 | 6.55 | ..... | ..... | ..... | ..... | ..... | ..... |
| On 0.500 | 6.0 | 45.2 | 6.75 | ..... | ..... | ..... | ..... | ..... | ..... |
| On 0.350—50 | 5.4 | 50.6 | 6.40 | 0.5 | 0.5 | ..... | 0.1 | 0.1 | ..... |
| On 0.166—90 | 21.3 | 71.9 | 7.08 | 2.8 | 3.3 | ..... | 1.9 | 2.0 | ..... |
| On 0.070—200 | 12.2 | 84.1 | 8.45 | 15.8 | 19.1 | ..... | 3.2 | 5.2 | ..... |
| Through 0.070—200 | 15.9 | 100.0 | 9.75 | 80.9 | 100.0 | ..... | 94.8 | 100.0 | ..... |
| TOTAL | 100.0 | ..... | 8.71 | 100.0 | ..... | 9.86 | 100.0 | ..... | 6.60 |

South end of main dust chamber = entrance; north end = outlet.

## III. Old Flue System.

The arrangement of flues in use prior to 1909 is shown in Plate I.[1] Waste gases from the various furnaces entered three main flues, through which they were conducted to the chimney which is 186 ft. high by 20 ft. inside diameter, and stands on a hill, about 1,300 ft. from the main smelter building. These flues were built about three-fourths underground, the walls being of rubble masonry, and having a 9-in. semi-circular brick arch roof (see Fig. 1). The cross-sectional area of flues Nos. 1 and 2 was 119 sq. ft. each; of No. 3 flue 152.6 sq. ft., making a total of 390.5 sq. ft., against a chimney area of 314 sq. ft.

Under normal conditions of operation, flues Nos. 1 and 2 carried the blast-furnace gases and a part of the reverberatory gases. Flue No. 3 carried the MacDougall and converter gases and the remainder of the reverberatory gases.

Door openings 3 ft. 2 in. by 5 ft., with cast-iron frames and doors, were built in the walls of each flue at irregular intervals of from 100 to 200 ft., for use in cleaning out the dust, which, after accumulating to a depth of several feet, would seriously throttle the draft on furnaces. No provision was made for removing the dust while the furnaces were in operation, so that it was necessary to shut them down at least once per year in order to clean out the flues. At such times all men available were pressed into service and the work continued on three shifts, night and day, until it was completed. The flues were always hot, causing the men to perspire; the dust stirred up in shoveling was very irritating to the skin, and the work was in general always a source of annoyance to the management and of dissatisfaction among the men.

In order to hasten the work, the dust was simply wheeled to the outside of the flue and dumped in piles on the ground. Later, it was taken to the furnaces, involving extra handling and expense.

The so-called dust chambers near the MacDougall furnaces and blast furnaces were little more than header flues. Only the heavier particles of dust settled in them, and, while this amounted to a considerable tonnage, it was only a small percentage of the dust produced by the furnaces. The same may be said of the main dust chamber, through which the gases from the MacDougall furnaces and converters passed. Each of these chambers was so arranged that the dust could be drawn out while the furnaces were in operation.

The dust from the blast-furnace chamber (see Fig. 2) was drawn into wheelbarrows, elevated and wheeled to a bin, from which it could be drawn into reverberatory charge cars. Dust from the MacDougall-

[1] This is shown as Fig. 2 of the paper of Corwin and Rodgers, Increasing the Efficiency of MacDougall Roasters, p. 1395, July *Bulletin*.

furnace flue (see Fig. 3) was drawn into regular calcine cars, pushed by hand to a transfer car and delivered to reverberatory charge hoppers. Dust from the main chamber (see Fig. 4) was drawn from chutes on either side into push cars which were then lowered on an elevator to the reverberatory charge-floor level and pushed by hand to a transfer car and delivered to the charge hoppers. This main chamber was so wide that only a part of the dust could be drawn out, making it necessary to shovel a large amount of it in order to give the chamber a thorough cleaning out.

From this it will be seen that, not only were the flues and dust chambers on the old flue system inadequate to settle and collect the dust being produced by the furnaces, but the method of handling the dust was crude and expensive.

Numerous observations were made to determine the draft, temperature and velocity in the old flues. The results varied widely, depending upon the number of furnaces in operation, the atmospheric conditions, and the amount of dust in the flues; but the following table may be taken as a fair average under normal operations.

| Flue No. | Point A—Near Dust Chamber | | | Point B—Near Base of Chimney | | |
|---|---|---|---|---|---|---|
| | Draft Reading In. Water | Temperature F.° | Velocity in Feet per Min. | Draft Reading In. Water | Temperature F.° | Velocity in Feet per Min. |
| 1 | 0.97 | 511 | 2230 | 0.61 | 476 | 2330 |
| 2 | 1.06 | 555 | 1730 | 0.33 | 516 | 1650 |
| 3 | 0.66 | 467 | 2840 | 0.40 | 458 | 2500 |

Temperature of atmosphere 39° F.; barometer, 26.8.

During a 21-day test on No. 3 flue gases, observations were taken every hour on the velocity, temperature and draft in each flue, at about the location of the point C, in Fig. 2 of the paper of Corwin and Rodgers already referred to, with results as follows:

| Flue No. | Draft Reading In. Water | Temperature F.° | Velocity Ft. per Min. | Volume Gas Calculated to 400° F. Cu. Ft. per Min. |
|---|---|---|---|---|
| 1 | 0.883 | 490 | 2,470 | 247,800 |
| 2 | 0.96 | 555 | 1,750 | 172,000 |
| 3 | 0.624 | 465 | 3,000 | 377,000 |
| Total | ..... | ..... | ..... | 796,800 |

Average temperature of atmosphere, 39° F.; average barometer, 26.94.

The total draft energy in the old flue system, when all flues were in operation, was used up in about the proportion indicated in the following tabulation:

| | Loss In In. Water | | |
|---|---|---|---|
| | Flue No. 1 | Flue No. 2 | Flue No. 3 |
| Loss from furnace inlet to point A | 1.53 | 1.69 | 1.04 |
| Loss from point A to point B | 0.53 | 0.24 | 0.78 |
| Loss from point B to top of chimney | 0.40 | 0.58 | 0.67 |
| Head equivalent of velocity of escaping gases | 0.23 | 0.23 | 0.23 |
| Total theoretical draft | 2.69 | 2.74 | 2.72 |

During conditions described above the gases from four blast furnaces and one reverberatory were being handled jointly by flues 1 and 2, while No. 3 flue was handling gases from 22 MacDougalls, two reverberatories, five converters and three Brückner dryers.

## IV. Tests on Old Flue System.

### 1. *Aspiration Tests.*

In the year 1900, soon after the capacity of the plant was increased, it became evident that a considerable amount of dust was passing out through the chimney and being lost. In 1903, some preliminary tests were made to determine, if possible, the amount and value of this material. Small quantities of gas were aspirated from the chimney and also from the flues. A large number of samples were taken, the amount of gas handled in each varying from 50 to 1,000 cu. ft. The velocity in the flues at this time was estimated by inserting pieces of newspapers in the flue and noting the time required for them to appear at the top of the chimney. Paper was inserted at different times at various points along the flue and an average of all observations taken. These estimates were later found to be only about 60 per cent. of the true velocity.

On account of the small quantity of gas handled, and the method of sampling, the results were not considered reliable, and in 1904 tests were started on a larger scale.

### 2. *Tests with Steel Settling Chamber.*

By means of an exhaust fan, gas was drawn from the flue, through a steel pipe 30 in. in diameter and 105 ft. long. This pipe was considered to be the settling chamber, and was connected to the flue by a smaller

pipe, on the end of which an elbow was so arranged that its open end faced the current of gas in the flue. In all, 31 tests were made with this arrangement, including tests on each of the old flues. The actual continuous running time of each varied from 28 to 42 hr., the average being about 35 hr. The velocity in the settling chamber for a given test was kept as nearly uniform as possible, but was varied for the different tests from 30 to 300 ft. per minute. After each test run was finished, the settling chamber was cleaned out, the dust weighed, sampled and assayed, and a new test started.

While valuable information was obtained from these tests, the results were still unsatisfactory, owing to the great amount of moisture which condensed in the pipe. The form and construction of this type of settling chamber being so radically different from the construction that would be necessary in a chamber of practical size made it hard to draw any conclusions as to what would be required to collect the dust properly from all the smelter gases.

In order, therefore, to make tests under conditions which would more nearly approximate actual working conditions, and at the same time to try out several suggested plans for hastening or increasing the settlement of dust, an experimental flue was constructed of brick. The general plan and arrangement of this flue is shown on Plate II. It was 4 ft. wide by 4 ft. 6 in. high and 304 ft. long, inside dimensions. As in the case of the steel pipe, the gases were drawn through this flue by means of an exhaust fan.

The inlet was through a steel pipe 22 in. in diameter. This pipe entered the old flue through the side wall and was fitted with an elbow at the inner end, the opening in which faced the current of gas at a point near the center of the flue.

With this general arrangement, 10 tests were made, numbered from 32 to 41 inclusive. The first eight of these, 32 to 39 inclusive, were made on gases drawn from No. 3 flue. During the last two, gases from No. 2 flue were tested, No. 1 flue being closed.

Before discussing these tests in detail, it may be of interest to explain the general method of conducting them, together with a description of the apparatus used for determining the amount and character of the gases handled and other physical conditions.

### *Testing Apparatus Used.*

Ordinary mercury thermometers were used for determining temperatures. Those used on the experimental flue were regular glass-stem thermometers, 12 to 16 in. in length, and were inserted through a hole in the roof, and held by a rubber stopper. Those used on the main flues

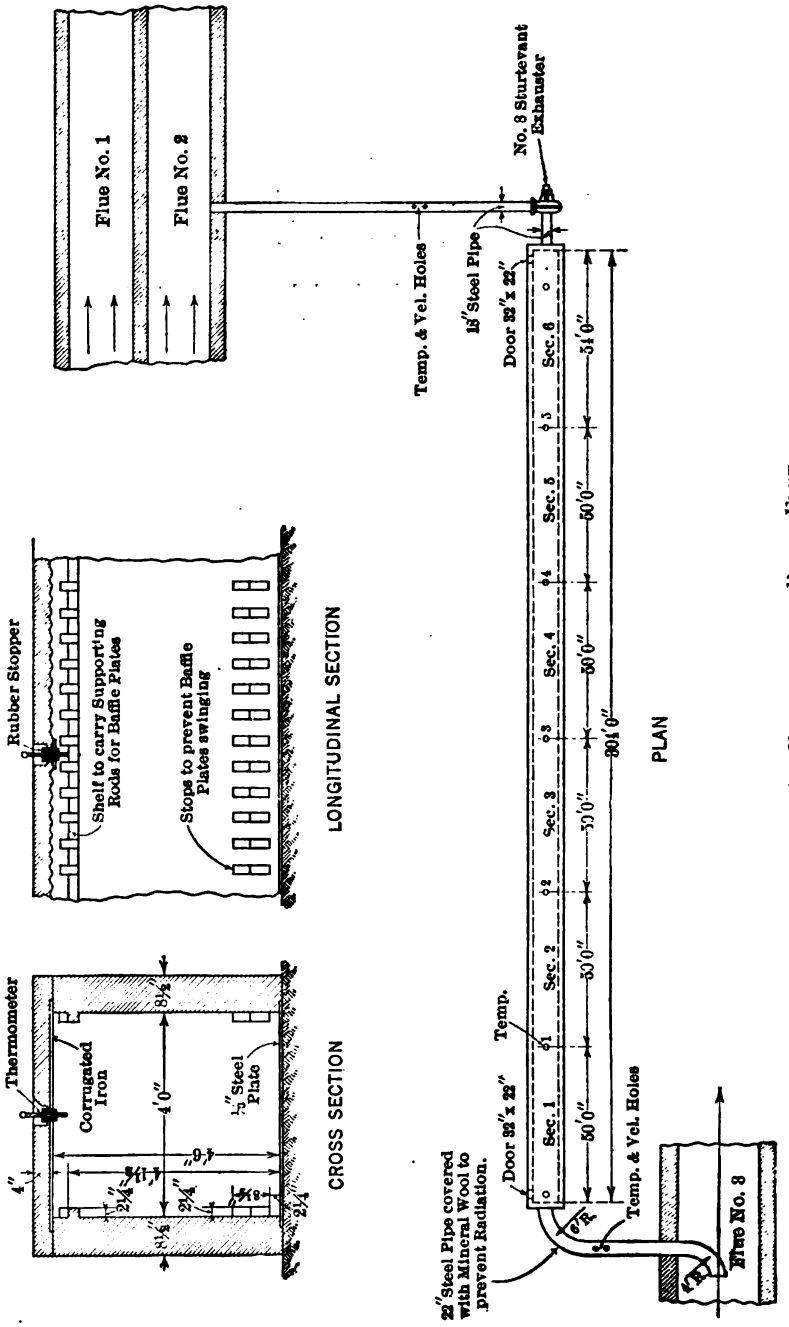



PLATE II. - EXPERIMENTAL BRICK FLUE.

were long-stem mercury flue thermometers, the stems being about 36 in. long.

In measuring velocities, Pitot tubes were employed, a special type being developed for this work. In recent years this instrument has come into such general use, and its principles have been so thoroughly discussed and explained, that any consideration of this phase of the subject seems unnecessary for the purposes of this paper.

While the underlying principle is the same in all its forms, there are almost as many types of the instrument as there are experimenters using it. It is a well recognized fact that the dynamic- or impact-pressure element of the tube is very easy to construct and manipulate, and that it will accurately indicate the velocity of a moving stream or current of fluid, provided the open end squarely faces the direction of flow. The static-pressure element is the one which requires careful attention. To measure the static pressure properly, a form of tube must be used which will not be affected by the impact or velocity pressure of the moving stream of fluid.

Several types of static-pressure end were tested by us. Air from a chamber in which the pressure was maintained practically constant was passed through a well-rounded orifice into a pipe 12 in. in diameter and about 40 ft. long.

Pitot tubes with different forms of static-pressure end were placed in this pipe, and the readings from the static-pressure side were compared with each other and also the volume of air indicated by each type of Pitot tube was compared with the volume calculated as passing through the orifice.

The form of pressure end finally adopted by us is shown in Plate III. It consists of an oval brass plate 2.5 by 4 in. by about $\frac{1}{16}$ in. thick, having the edges beveled on top. In the center and on the long axis of this plate, a slot is cut $\frac{3}{64}$ in. wide by 2 in. long. This slot communicates with a chamber in a light brass casting which is riveted or brazed to the top of plate as shown. The tube leading from this chamber to the manometer may be of any suitable form. For use in small flues or pipes all parts of the instrument should be kept as small and light as possible, consistent with strength, so as to offer the least possible resistance to the flow of gases. When placed in a jet of air from a nozzle in the side of a chamber in which the pressure was 12 lb. per sq. in. gauge, which corresponds to a velocity of about 850 ft. per second, this form of pressure end shows no appreciable reading on the manometer.

Assuming that the pressure energy under such conditions is all converted into velocity, the conclusion is reached that this type of pressure end will correctly indicate the static pressure and is not affected by velocity. It is to be understood, of course, that the tube is to be so placed

that the direction of flow of the moving stream of gas or air is parallel to the bottom plane of the plate and to the slot in the plate.

Other types of pressure end tested, when similarly placed in an air jet, showed readings on the manometer of from minus 84 in. of water for a straight open-end tube, to plus 3 in. of water, for a tube having on its end a thin circular plate 4 in. in diameter.

The type of pressure end designed and used by Capt. D. W. Taylor naval constructor, in his Experiments with Ventilating Fans and Pipes,[1]

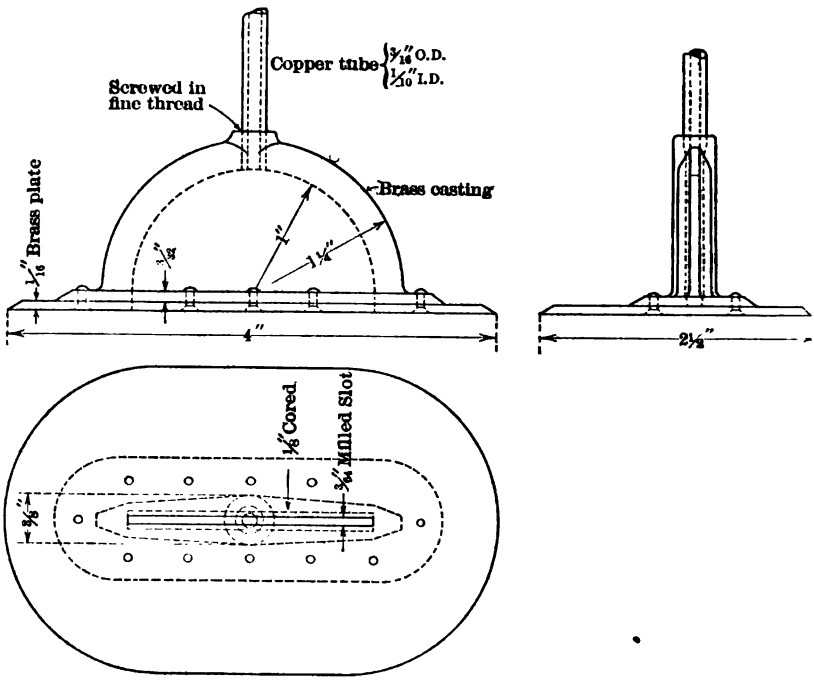



PLATE III.—PITOT TUBE, STATIC PRESSURE END.

was among those tested and was found to be very accurate and, in fact, has been frequently used by us in determining the velocity of air in pipes, etc., but this type was not considered as well adapted for work in connection with flue gases as the type which we used.

In connection with the work of testing Pitot tubes, we also made some observations to determine the average velocity over the whole section area of a pipe, and the percentage which this velocity is of the velocity at the center of pipe. The pipe was assumed to be divided into a number of concentric rings, the area of each being multiplied by the average velocity in it as determined by the Pitot tube. These observations indicate

[1] Transactions Society of Naval Architects and Marine Engineers, Vol. XIII (190

that for a 12-in. pipe the average velocity of the whole cross-section is 89.6 per cent. of the central velocity.

This value, however, was seldom used for calculations on our tests. Instead of taking one reading at the center of a pipe or flue, we used a Pitot tube having five impact points and one static-pressure point,

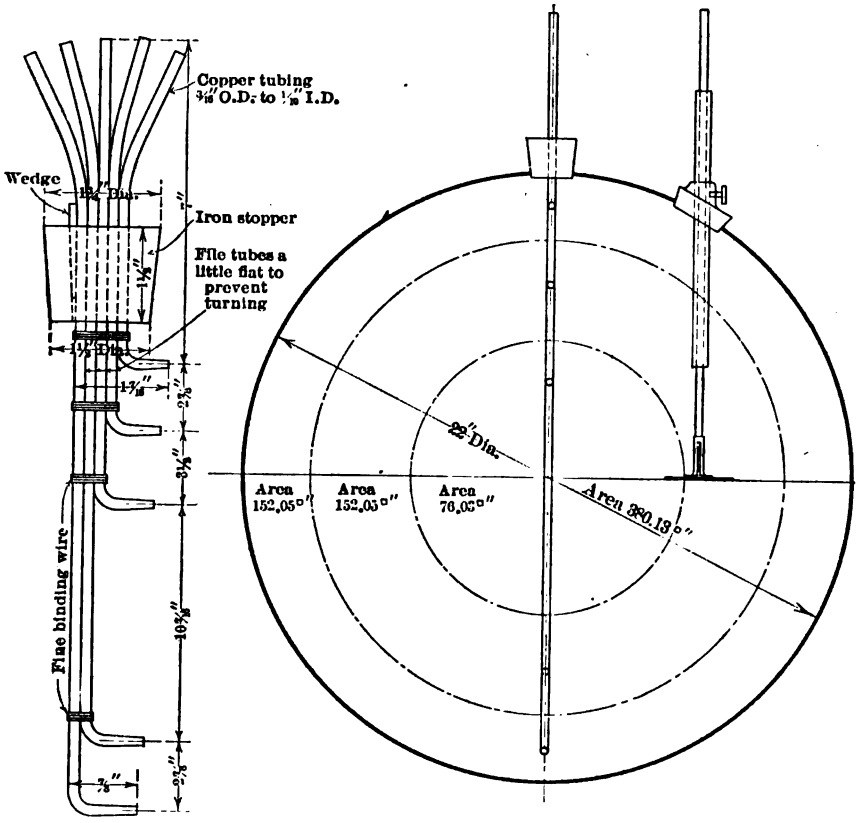



PLATE IV.—IMPACT END OF PITOT TUBE FOR 22-IN. PIPE.

the assumption being that the static pressure was practically the same at all points in the flue or pipe. Plate IV shows the arrangement and construction of the tube as designed for use in a 22-in. pipe. For the main flues, 0.25-in. gas pipes of the required length, having a 90° bend at the lower end, were used. These were not nested together as in the case of the tube for pipe work, but were inserted independently at points where it was thought a good average velocity would be obtained. Plate XXXVIII shows the location and arrangement of these tubes.

For use in connection with this multiple-point Pitot tube, we designed and, so far as we know, put into use for the first time what we have called

an automatic averaging manometer. This instrument is shown in Plate V, Fig. 1. The construction of this manometer is clearly indicated in the illustration and needs little further explanation. The first instrument was made of cast iron, but this soon corroded and filled the liquid with rust, which so clogged and clouded the glass tube that accurate readings could not be made, so that it was necessary to construct it of brass. Each of the chambers in the main body of the instrument must be of the same and of uniform diameter. The diameter of these chambers is made large in proportion to the diameter of the glass tube on which readings are taken, so that, within reasonable limits, the variation in level in these chambers may be neglected and the total reading obtained on one leg of the instrument. The various points of the Pitot tube were connected to the manometer by means of pure gum-rubber tubing, care being taken to see that good joints were made. This instrument is easy to operate and requires little attention. It is defective in that the velocity head which it indicates is the arithmetical average of velocity heads at the several points of the Pitot tube, while the true average velocity varies as the square root of the heads. The difference in these two values, however, is small for velocities such as those with which we were dealing, and we thought it more accurate to use five points in this way than to use one central point and apply an arbitrary factor for obtaining an average.

To calculate the velocity from the reading given by a Pitot tube, the general formula

$$V^2 = 2gh$$

may be used for velocities up to 5,000 or 6,000 ft. per minute, unless extreme accuracy is desired. For higher velocities or for very accurate work a more exact and complicated formula may be used.[1]

In the above formula

V = velocity in feet per second.
g = acceleration due to gravity = 32.16.
h = the height of a column of air or gas, the velocity of which is being measured, equivalent in weight to a column of like sectional area of the liquid used in the manometer, and of height equal to the manometer reading.

In our work, we used grain alcohol in the manometer. Being less dense, this liquid works more freely than water in a small glass tube and for the same reason gives a larger reading which is an advantage where low velocities are being measured.

In connection with single-point Pitot tubes, and for taking draft readings, frictional resistances, etc., we used a manometer similar to that illustrated by Fig. 2, Plate V. It will be noted that one leg of this

[1] See article on Experiments on Ventilating Fans and Pipes, by D. W. Taylor, Transactions Society Naval Architects and Marine Engineers, Vol. XIII (1905).

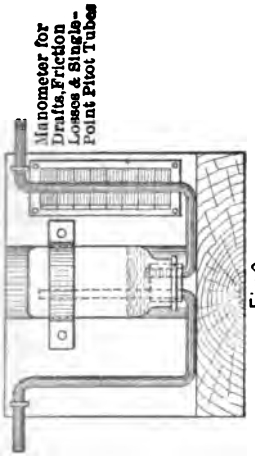



Fig. 2

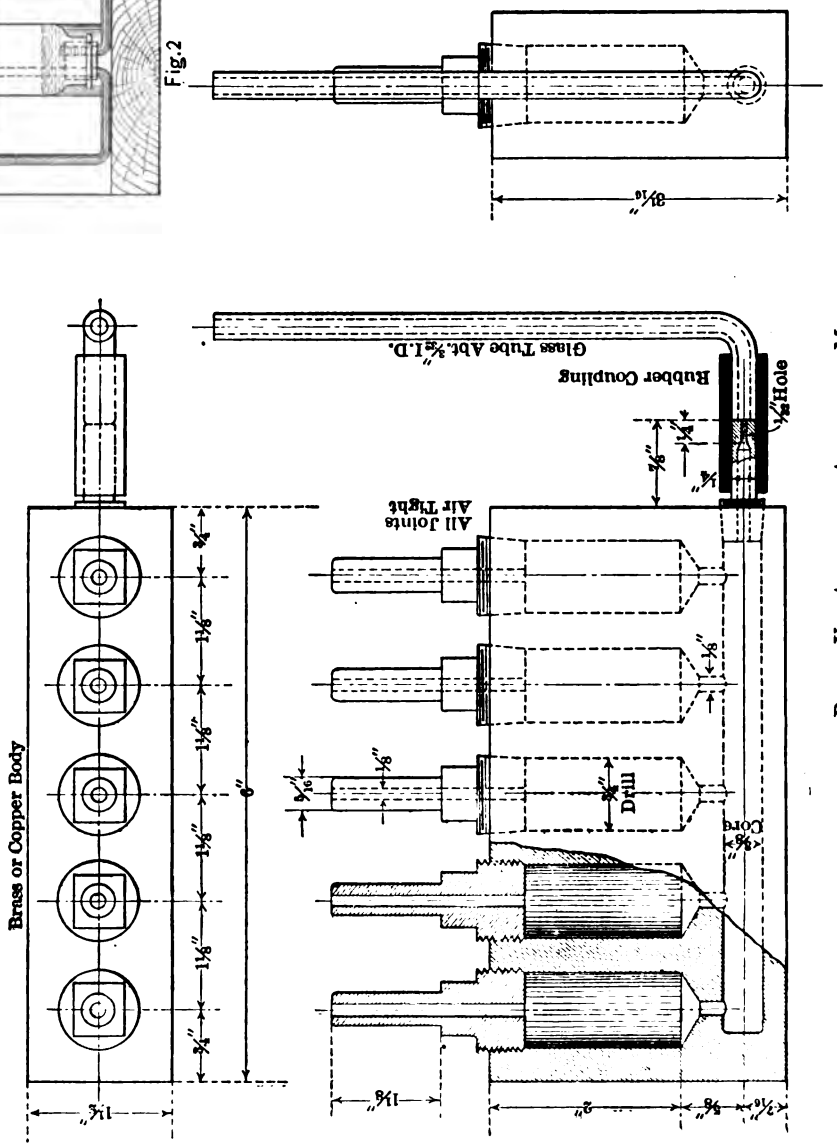



PLATE V.—AUTOMATIC AVERAGING MANOMETER.

manometer is made larger in diameter, so that the movement in it may be neglected, the total reading being obtained on the small leg.

Barometric pressures were obtained from a standard recording aneroid barometer.

### 3. *Test with Experimental Brick Flue.*

Referring again to Plate II., showing general arrangement of experimental brick flue: Thermometers were inserted through the roof of flue as shown, at intervals of approximately 50 ft. Temperatures as well as velocities were taken in the inlet and outlet pipes at points indicated, as well as in the main flue. After a test was started the exhaust fan was kept in continuous operation, day and night, until it was completed, observations of temperature, velocity, etc., being taken every hour.

For the purpose of determining the rate of settlement as well as the character of the dust at different points in the flue, it was divided into six sections, the first five of which, numbered from the inlet end, were 50 ft. long each, the last or sixth section being 54 ft. long. At the end of each test the dust was cleaned out, weighed and sampled by sections.

During test No. 32, there were no baffles or obstructions of any kind placed in the flue, the full cross-sectional area being clear and open as built.

During test No. 33, surface plates similar to Freudenberg plates[1] were hung in the flue for a distance of 170 ft., beginning about 8 ft. from the inlet end. The number and arrangement of these plates is shown on Plate VI, Fig. 1. These plates were thin copper sheets such as are used in our electrolytic refinery for starting sheets. It will be noted that in the space filled with sheets, a baffle plate 8 in. high was placed on the floor every 10 ft.

Test No. 34 was with an open flue, the conditions in every respect being as nearly as possible a duplicate of those in test No. 32.

For test No. 35, baffle plates were hung in the flue, the arrangement being as indicated on Plate VI, Fig. 2. There was a total of 184 ft. of the flue filled with these plates, arranged in two divisions.

The first division started 8 ft. from the inlet end of the flue and extended for a distance of 92 ft., or to the end of Section 2. Next there was 100 ft. of clear flue, covering Sections 3 and 4; followed by 92 ft. of baffles covering Section 5 and the greater part of Section 6.

These baffles were copper sheets 6.25 in. wide, being sheared from the surface plates used in test No. 33. A clip was attached to the lower end of each plate, through which a rod was passed to engage the stops on flue walls and prevent the baffles from swinging.

In test No. 36, 3⅛-in. baffle plates were used throughout the flue, with the exception of about 8 ft. at each end. The number and arrange-

[1] For description see Hofman's *Metallurgy of Lead*, Sixth Edition, p. 389.

ment of baffles for a given section was the same as that shown for the 6.25-in. baffles; that is, the free area of the flue was reduced only about one-fourth instead of one-half.

Test No. 37 was made to determine the effect on the settlement of dust of expansions and contractions of the gases. Two bulkheads were built

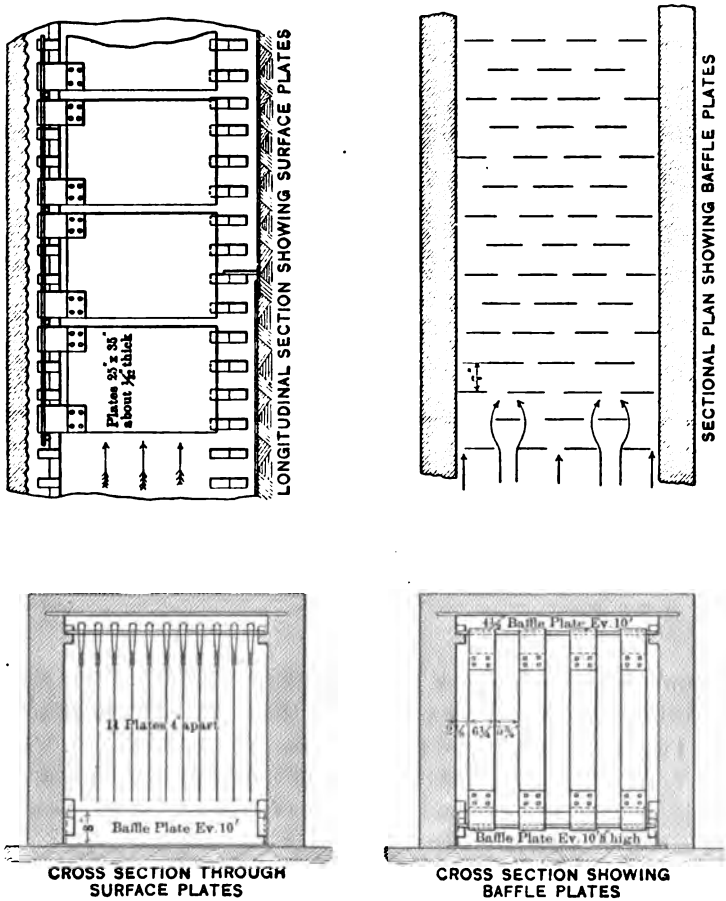


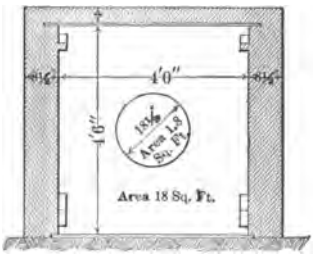

PLATE VI.—EXPERIMENTAL BRICK FLUE BAFFLING ARRANGEMENT.

across the flue at 100 ft. and 200 ft., respectively, from the inlet end. In the center of each of these bulkheads, a pipe 18.5 in. in diameter was placed, all as shown on Plate VII. The inlet pipe, together with these two bulkhead pipes, made in effect, three expansions and contractions. Other portions of the flue were clear of any baffles or obstructions.

Test No. 38 was of a preliminary nature, for the purpose of determining the effect of wire baffles[1] in the settlement of flue dust. This method of

[1] See Hofman's *Metallurgy of Lead*, Sixth Edition, p. 392.

settling flue dust was invented[1] by Dr. Bernhard Rösing, of Friedrichshütte, near Tarnowitz, Prussia, Germany. It was used with good success in settling the dust produced by a smeltery at Tarnowitz, but, up to this time, had not been used on a large scale in this country. As originally designed by Dr. Rösing and used in Germany, the wires were hung in




CROSS SECTION AT CONTRACTED AREA

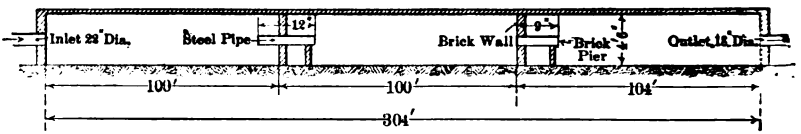



PLATE VII.—ARRANGEMENT OF EXPERIMENTAL FLUE FOR EXPANSION AND CONTRACTION TESTS.

small chambers through which the gas passed parallel to the wires, but in our work we passed the gas through them at right angles to their length. Two 50-ft. sections of the flue, viz., No. 3 and No. 6, were hung with No. 7, B. & S. gauge copper wires, which extended from near the roof to near the floor of the flue and were supported by common poultry netting. They were spaced at from 2 to 2.25 in. centers.

For test No. 39, which was the longest test run made, the flue was hung with copper wires throughout, with the exception of about 9 ft. at each end, making a total of 286 ft. of wire baffles. The wires were supported in the same way as for test No. 38, and were spaced the same, at 2 to 2.25 in. centers.

For tests Nos. 40 and 41, which were made on gases drawn from No. 1 flue at a time when it was handling gases from blast furnaces only, the number and arrangement of wire baffles was the same as for test No. 39.

During all tests, the 22-in. inlet pipe was boxed in and lagged with

[1] Flue-Dust Collector, U. S. Letters Patent 432,440.

TABLE III.—*Summary of Tests Made in Rectangular Brick Flue.*

| Test No. | Duration of Test Hr. | Volume of Gas Handled per Test Cu. Ft. at 400° F. | Total Dust Collected Lb. | Dust per Million Cu. Ft. Gas at 400° F. Lb. | Average Assay of Dust | | Average Velocity in Ft. per Min. | | | Average Temperature F°. | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | Per Cent. Cu | Silver Oz. per Ton | Experimental Flue | Main Flue | Inlet Pipe to Exp. Flue | Main Flue | Inlet End Exp. Flue | Outlet End Exp. Flue |
| 32 | 138.8 | 89,090,000 | 3,429.0 | 38.49 | 7.69 | 3.93 | 568 | 2,509 | 3,986 | 368 | 362 | 250 |
| 33 | 143.3 | 90,016,000 | 4,336.0 | 48.17 | 7.35 | 3.944 | 561 | 2,570 | 3,987 | 406 | 369 | 254 |
| 34 | 117.6 | 74,070,000 | 2,924.0 | 39.48 | 9.45 | 4.716 | 553 | 2,590 | 3,994 | 389 | 355 | 258 |
| 35 | 143.0 | 42,400,000 | 4,039.8 | 95.27 | 6.83 | 3.511 | 266 | 2,735 | 1,971 | 440 | 373 | 187 |
| 36 | 163.2 | 65,540,000 | 5,064.6 | 77.25 | 8.02 | 4.094 | 344 | 2,819 | 2,740 | 376 | 336 | 187 |
| 37 | 113.9 | 49,324,000 | 3,992.3 | 80.94 | 7.42 | 3.5 | 414 | 2,803 | 3,493 | 491 | 427 | 258 |
| 38 | 108.8 | 53,291,000 | 4,832.4 | 90.68 | 7.75 | 3.98 | 477 | 2,964 | 3,909 | 496 | 444 | 305 |
| 39 | 491.4 | 221,039,000 | 21,825.0 | 98.69 | 6.46 | 3.46 | 440 | 3,000 | 3,069 | 465 | 449 | 311 |
| 40 | 165.4 | 73,372,000 | 6,730.4 | 91.73 | 6.1 | 2.99 | 454 | 3,084 | 3,203 | 549 | 490 | 297 |
| 41 | 164.9 | 71,626,000 | 18,251.6 | 254.82 | 6.7 | 3.15 | 470 | 3,277 | 3,307 | 623 | 540 | 335 |

NOTE: Tests 32 to 39 inclusive made on gases from No. 3 main flue. Tests 40 and 41 made on gases from No. 1 flue when handling blast furnace gases only, four furnaces being in operation during each test.

*Conditions of Test.*

| Test No. | | Average Number of Furnaces Discharging Into Flue | | | | |
|---|---|---|---|---|---|---|
| | | Blast | MacDougall | Converter | Reverberatory | Brueckner |
| 32 | Clear flue, no baffles | ..... | 19.0 | 3.9 | 1 | 3 |
| 33 | Surface plates for length of 170 ft., beginning 8 ft. from inlet end | ..... | 17.5 | 5 | 1 | 3 |
| 34 | Clear flue, no baffles | ..... | 18.4 | 4.6 | 1 | 3 |
| 35 | 6.25 in. baffle plates in sections 1, 2, 5 and 6; sections 3 and 4 clear flue | ..... | 21.3 | 5 | 1 | 3 |
| 36 | 3.125 in. baffle plates throughout the flue, except about 8 ft. at each end | ..... | 20.0 | 3.6 | 1 | 1.9 |
| 37 | Expansions and contractions, three stages | ..... | 21.6 | 5.3 | 1.8 | 3 |
| 38 | Wire baffles in sections 3 and 6, other section clear flue | ..... | 22 | 6 | 2 | 3 |
| 39 | Wire baffles throughout flue except about 9 ft. at each end | ..... | 22 | 4.8 | 2 | 3.5 |
| 40 and 41 | Same as No. 39 | 4 | ..... | ..... | ..... | ..... |

mineral wool, so that the gases would be delivered to the experimental flue at as near normal temperature as possible.

Table III gives a general summary of the results of tests 32 to 41 inclusive. In comparing the results of tests in which different kinds of baffles were used, consideration must be given to the fact that in tests 33, 35 and 38, only a part of the flue was equipped with baffles. Direct comparisons may be made among tests 32, 34, 36 and 39. Tests 40 and 41 will be discussed a little later in this paper.

Table IV presents in more detail the results of test 39. It shows the average temperature in each section, as well as the amount of dust collected in each, with the analysis of the same. It will be noted that the amount of copper decreases from 6.4 per cent. in section 3 to 2.6 per cent. in section 4. At the end of this test (No. 39) it was noted that the wires in section 1, and about one-half of section 2, were clean and free from dust. Beginning at the middle of section 2, there was a deposit of dust on the wires, the amount of which gradually increased, the maximum deposit being at about the division line between sections 3 and 4, at which point the overall diameter of wire and dust was about $\frac{5}{8}$ in.

A moderate rapping or shaking of the wires was effective in removing a larger part of the dust adhering to them.

A marked difference was noted in the character of the dust adhering to the wires in the different sections. Up to about the end of section 4, this deposit was of a crystalline character, while, in the following sections, it was of a light amorphous nature.

In addition to the assays shown in Table IV, a more complete assay was made on a combined sample made up from the several sections in proportion to the amount of dust collected in each. This assay was as follows:

| | |
|---|---|
| Cu, per cent | 6.2 |
| Ag, oz. per ton | 3.4 |
| Au, oz. per ton | 0.025 |
| Insoluble, per cent | 38.2 |
| $SiO_2$, per cent | 26.0 |
| Fe, per cent | 5.7 |
| $Al_2O_3$, per cent | 13.3 |
| CaO, per cent | 0.8 |
| S, per cent | 7.1 |
| $SO_3$ (free and combined), per cent | 14.1 |

In order to determine the amount of dust escaping from the experimental flue during test 39, a measured quantity of gas was aspirated, two methods of collecting the dust being used. In one of these methods the gas was filtered through a chamber filled with mineral wool, a comparatively large volume being handled, and in the other, it was aspirated through absorption bottles which were preceded by a bulb filled with mineral wool.

TABLE IV. TEST NO. 39.—*General Summary of Observations and Results.*

| Section of Flue | Temperature of Gas F°. | Dust Collected, Lb. | | Pounds Dust per Million Cu. Ft. Gas Vol. at 400° F. | | Per Cent. of Total Dust Collected | | Assay of Dust | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | Per Section | Cumulative | Per Section | Cumulative | Per Section | Cumulative | Cu Per Cent. | $SO_3$ Free and C'm'd Per Ct. | Silver Oz. per Ton |
| Inlet Pipe | 470 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| No. 1 | 449 | 4,404 | 4,404 | 19.87 | 19.87 | 20.18 | 20.18 | 9.3 | 6.8 | 4.4 |
| No. 2 | 418 | 6,205 | 10,609 | 28.06 | 47.93 | 28.43 | 48.61 | 7.8 | 10.1 | 3.8 |
| No. 3 | 387 | 6,277 | 16,886 | 28.37 | 76.30 | 28.76 | 77.37 | 6.4 | 13.0 | 3.2 |
| No. 4 | 358 | 2,557 | 19,443 | 11.56 | 87.86 | 11.72 | 89.09 | 2.6 | 20.5 | 2.2 |
| No. 5 | 332 | 1,102 | 20,545 | 5.06 | 92.92 | 5.05 | 94.14 | 2.2 | 24.6 | 2.5 |
| No. 6 | 311 | 847 | 21,392 | 3.81 | 96.73 | 3.88 | 98.02 | 2.1 | 30.9 | 3.2 |
| Exstr. & Con. Pipes | 283 | 433 | 21,825 | 1.94 | 98.67 | 1.98 | 100.00 | 1.6 | 44.3 | 2.8 |

GENERAL DATA ON EXPERIMENTAL FLUE:

| | |
|---|---|
| Duration of test | 491 hr. 22 min. |
| Average velocity in inlet pipe | 3,070 ft. per min. |
| Approximate mean velocity of gases in flue | 440 ft. per min. |
| Average temperature of gases in flue | 376° F. |
| Average temperature of atmosphere | 39° F. |
| Frictional and other losses in first 100 ft. of flue | 0.4 in. of water |
| Frictional and other losses in second 100 ft. of flue | 1.2 in. of water |
| Frictional and other losses in third 100 ft. of flue | 0.7 in. of water |
| Total gas handled per test, volume at temperature in inlet pipe | 238,887,000 cu. ft. |
| Total gas handled per test, volume at 400° F | 221,039,000 cu. ft. |
| Pounds dust per million cu. ft. gas | 98.69 |

GENERAL DATA ON MAIN FLUE:

| | |
|---|---|
| Average temperature of gases | 465° F. |
| Average velocity of gases | 3,000 ft. per min. |
| Effective area of cross-section | 135 sq. ft. |
| Draft reading at point of velocity measurements | 0.624 in. of water |
| Cu. ft. gas passing up main flue per min., at observed temperature | 405,500 |
| Cu. ft. gas passing up main flue per min., volume at 400° F | 377,000 |
| Lb. dust carried by gases in main flue per 24 hr., as indicated by test | 53,580 |
| Average number of furnaces discharging gas into main flue, 22 MacDougalls, 2 reverberators, 4 8 converters, 3 5 Brueckners | |

The dust collected in each case was weighed and analyzed, and the results were found to check very closely.

Based on these tests, it was estimated that, if all gases being carried by No. 3 flue were passed through a wire-baffled chamber similar to the experimental flue, there would be a loss of 7,320 lb. of dust in 24 hr. Therefore,

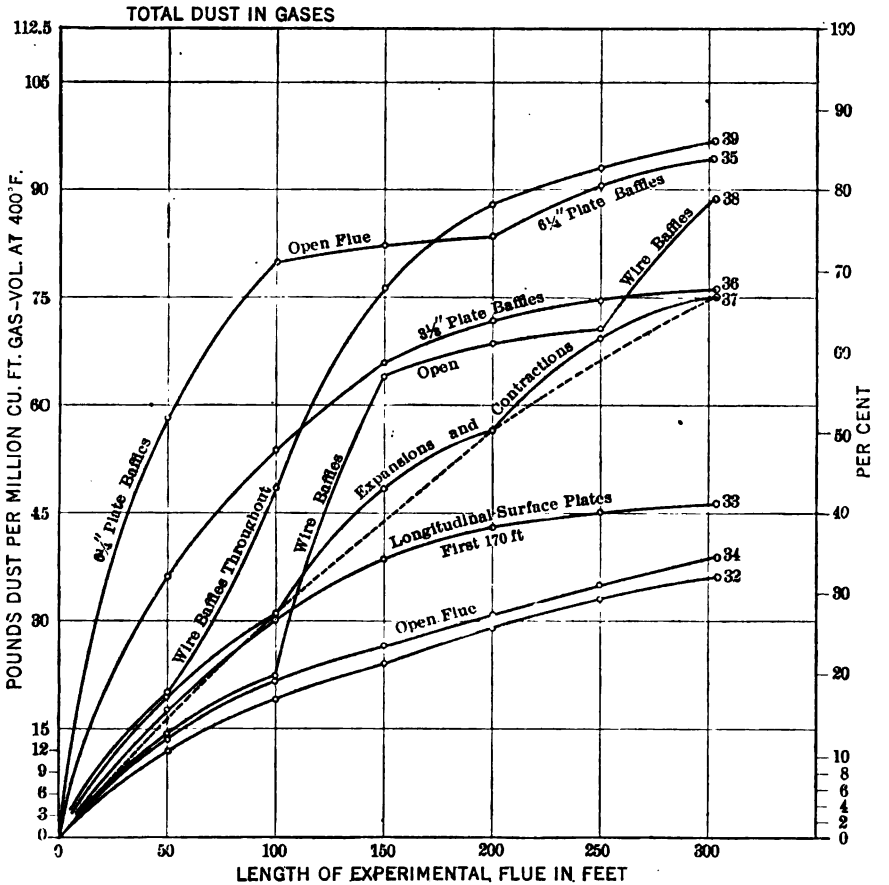



PLATE IX.—RELATIVE EFFICIENCY OF DUST-ARRESTING SCHEMES.

the total amount of dust estimated as passing out through No. 3 flue per 24 hr. would be 53,580 + 7,320 = 60,900 lb., containing 3,580 lb. of copper. It was further estimated, based on the results of test 39, that, if all the gases carried by No. 3 flue should be passed through a wire-baffled flue or chamber under the same conditions of velocity and temperature, the recovery would be 86 per cent. of the dust and 96 per cent. of the copper.

The curves on Plate IX show more clearly than a table the relative merits of the various dust-arresting schemes tested. The horizontal line at top designated as "Total Dust in Gases" is based on the results of test 39. Assuming that this line represents the total average dust in the gases at all times, it may be used as a basis for comparison of the efficiency of the various dust-arresting schemes tried out. From these curves an estimate may also be made as to the probable length of flue necessary to collect a given amount of dust, by the use of any particular scheme, and shows that with an open flue an almost infinite length of flue of uniform cross-section would be required to effect a recovery equivalent to that obtained by the use of wire baffles. It is interesting to note the trend of curves representing tests 35 and 38, in which only certain sections of the flue were filled with baffles. In the open sections for these tests the curves are approximately parallel with the curves for the open-flue tests.

Another observation to be made is that the baffle plates effect a more rapid settlement of dust than other methods; that is, a higher percentage of the total dust is settled in a shorter length of flue, but this is at the expense of draft energy, as will be pointed out later.

The favorable showing made by expansions and contractions in the settlement of dust is of considerable interest and indicates that additional tests along this line might develop some valuable information. The simplicity of the scheme commends it over the more complicated baffling methods, the only serious objection being the draft hindrance which it introduces. It is possible, however, that a greater number of stages, with a lesser percentage of contraction, might give equal efficiency and at the same time offer less resistance to the flow of gas.

Plate X, Fig. 1, shows the relative frictional resistance to the passage of gases per 100 ft. of flue, under the conditions of the test. In comparing these resistances, consideration must be given to the volume of gas handled per unit time for the different tests, the relative amounts of which are shown in Fig. 3, while Fig. 4 shows the relative dust collected per unit volume of gas and Fig. 2 shows the relative draft readings at outlet end of flue, in inches of water. During all of these tests the exhauster was run at practically the same speed, the average being about 1,250 rev. per min.

To determine how the frictional resistance in the wire baffles increased as the test progressed, observations were taken from time to time during test 39, the results of which are presented in Plate XI. The exhauster was run at a practically constant speed of 1,250 rev. per min. The increase in frictional resistance was effective in reducing the volume of gas handled from 9,500 cu. ft. per min. at the beginning of test to 7,100 cu. ft. on the twentieth day. This loss in volume was partially due to a reduction in sectional area on account of the accumulated dust on the floor of the flue, which amounted to nearly a foot in depth in some sections.

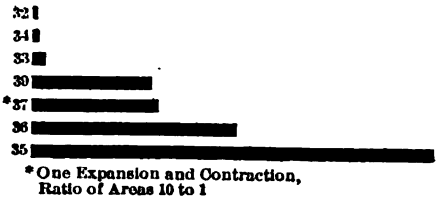



FIG. 1.—Relative Friction Losses per Hundred Feet of Flue.

FIG. 2.—Relative Draft Readings at Outlet End of Flue, Inches Water.

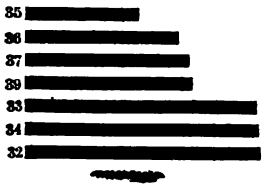



FIG. 3.—Relative Volume of Gas per Unit Time.

FIG. 4.—Relative Weight of Dust per Unit Volume of Gas.

Test No.
32. Clear Flue.
33. Surface Plates.
34. Clear Flue.
35. Six ¼-in. Baffle Plates.
36. Three ⅛-in. Baffle Plates.
37. Expansions and Contractions.
39. Wire Baffles.

PLATE X.

Based on velocity determinations made on all main flues during test 39, the total gas discharged from the smelter at that time was as follows:

| FLUE NUMBER | Average Observed Temp. F°. | Cu. Ft. per Min. at Observed Temp. | Weight Lb. |
|---|---|---|---|
| 1 | 490 | 273,730 | 10,200 |
| 2 | 555 | 203,000 | 7,080 |
| 3 | 465 | 405,500 | 15,520 |

If all these gases were combined in one main flue, the resulting temperature would be 490° F., at which the volume would be 880,300 cu. ft. per min.

### 4. *Tests on Blast-Furnace Gases.*

As already stated, tests 40 and 41 were made on blast-furnace gases only. The experimental brick flue, hung with wire baffles throughout, was used in these tests, the arrangement being similar in every respect to that employed in test 39, with the exception that baffle plates 8 in. high were placed on the floor at 10-ft. intervals, in the last 100 ft. of flue.

Information on two points was sought in these tests. First, a determination of the efficiency of wire baffles in the settlement of blast furnace flue dust; and second, to determine, if possible, what percentage of the briquettes being treated in the blast furnaces was being lost in flue dust.

Previous preliminary tests on blast-furnace gases had indicated that the dust carried by them was more difficult to collect than that contained

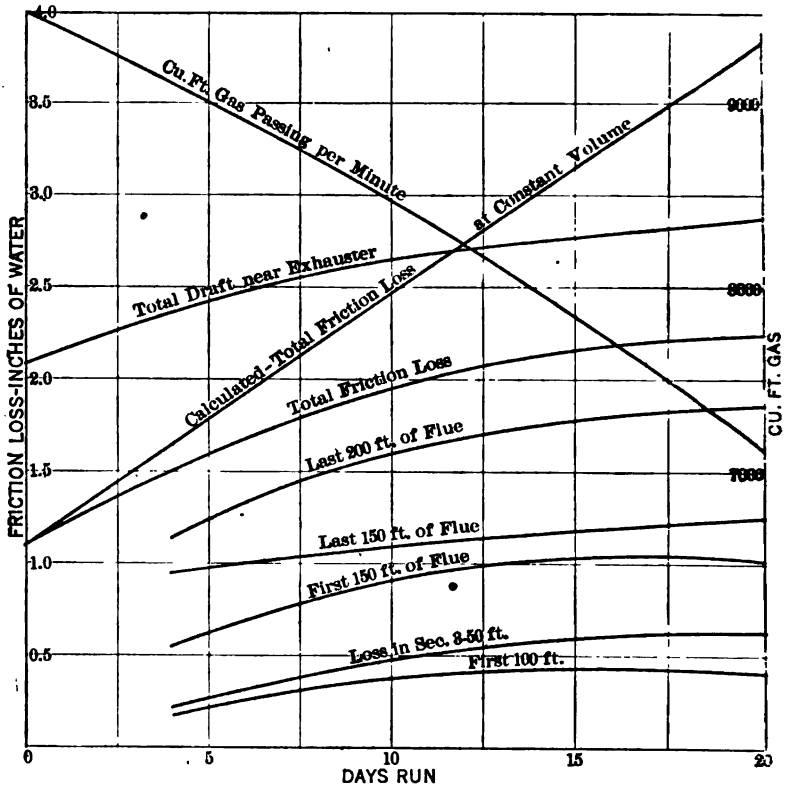



PLATE XI.—FRICTION LOSS THROUGH WIRE BAFFLES, EXPERIMENTAL BRICK FLUE

in the roaster gases. This conclusion was further confirmed by the present tests. It was evident that some dust was escaping from the experimental flue. To determine the amount so escaping, tests were made similar to those carried on during test 39. Even after filtering the gases through mineral wool, there seemed to be a slight amount of dust still escaping. Assuming, however, that practically all of the dust was recovered in the mineral-wool filter, calculations were made showing the percentage of total dust which was collected in the experimental flue. These results are shown in Table V, in parallel comparison with similar results for test 39, which was made on roaster gases.

TABLE V.—*General Comparison of Results and Conditions for Tests 39, 40 and 41.*

| | Tests | | |
|---|---|---|---|
| | 39 | 40 | 41 |
| Average temperature in main flue, F° | 465 | 549 | 623 |
| Average temperature in Sec. 1, Exp. flue, F° | 449 | 490 | 540 |
| Average temperature in Sec. 6, Exp. flue, F° | 311 | 297 | 335 |
| Average temperature of atmosphere, F° | 39 | 45 | 55 |
| Average velocity in Exp. flue, ft. per min | 500 | 450 | 470 |
| Dust recovered in Exp. flue, per cent. of total | 86 | 81 | 80 |
| Copper recovered in Exp. flue, per cent. of total | 96 | 90 | 91 |
| Average per cent. Cu in dust escaping from Exp. flue | 1.65 | 2.9 | 2.5 |
| Average per cent. $SO_3$ in dust escaping from Exp. flue | 45.1 | 11.7 | 9.2 |

It will be noted that the initial temperature of the gases is higher for tests 40 and 41 than for No. 39, which is believed to materially reduce the rate of settlement of dust. While no screening tests were made, the blast furnace dust seemed to be decidedly finer and dryer than the roaster dust, and did not adhere to the wire baffles as readily. The wires were practically clear of dust up to about the middle of the flue, and had only a light deposit for the last, or upper, half of flue. This deposit was readily dislodged by rapping or shaking the wires. The final temperature in tests 40 and 41 was low on account of an excess air leakage into the experimental flue.

*Briquettes as Dust Producers.*

During test 40 the furnaces were operated under normal conditions, no briquettes being included in the charge. During test 41 the operating conditions were as nearly like those for test 40 as possible, but about 12.5 per cent. of the charge was composed of briquettes. During both tests the furnaces were working well, the tonnage smelted being in general above the average. Practically all briquettes were charged to the furnaces directly from the machine, which was a Chambers auger brick machine with automatic wire cutoff. There was a small quantity which had been on hand for several days, but these were piled so that they would dry out very little. The average amount of moisture contained in the briquettes was 13 per cent., and they were of the following general composition:

| | *Per Cent.* |
|---|---|
| Copper precipitate | 1.31 |
| Slimes | 47.17 |
| Fine concentrate | 51.52 |

The following table shows the amount of each class of material treated during the tests:

| | Test 40 | Test 41 |
|---|---|---|
| Time represented by material treated, hr. | 167.75 | 166.00 |
| Actual running time of test, hr. | 165.42 | 164.92 |
| Average blast pressure, oz. | 38.3 | 41.3 |

| | Wet Weight Tons | Per Cent. of Charge | Wet Weight Tons | Per Cent. of Charge |
|---|---|---|---|---|
| First-class ore | 3,635.5 | 32.07 | 3,148.6 | 26.57 |
| Coarse concentrates | 2,783.9 | 24.56 | 2,433.8 | 20.54 |
| Briquettes | ..... | ..... | 1,477.2 | 12.46 |
| Converter slag | 1,332.0 | 11.75 | 1,362.9 | 11.50 |
| Lime rock | 3,585.5 | 31.62 | 3,428.7 | 28.93 |
| TOTALS | 11,336.9 | 100.00 | 11,851.2 | 100.00 |
| Coke | 1,042.7 | 9.19 | 1,025.4 | 8.65 |

The average weight of new cupreous material treated per 24 hr. during test 40 was 918.4 tons. The amount of dust being carried along by the main-flue gases per 24 hr. was therefore equivalent to 2.57 per cent. of this weight.

Applying this percentage to the weight of new cupreous material treated per 24 hr. during test 41, exclusive of briquettes, we obtain 41,484 lb. Deducting this amount from the total dust per 24 hr. as indicated by the test gives a balance of 90,616 lb. of dust per 24 hr. produced by the briquettes.

Since the average dry weight of briquettes charged during the same period was 371,660 lb., the conclusion is reached that 24.4 per cent. of the material of which they were composed was lost in flue dust. An even greater percentage was doubtless carried out of the furnace, some of which was recovered from the dust chamber.

Tables VI and VII show the detailed results of these tests. Based on these results and on results of tests on gases leaving the experimental flue, the total amount of dust in the main flue gases would be as follows:

TABLE VI. TEST 40.—*General Summary of Observations and Results.*

| Section of Flue | Temperature of Gas F.° | Dust Collected, Lb. | | Pounds Dust per Million Cu. Ft. Gas Vol. at 400° F. | | Per Cent. of Total Dust Collected | | Assay of Dust | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | Per Section | Cumulative | Per Section | Cumulative | Per Section | Cumulative | Per Cent. Cu | Per Cent. $SO_3$ Free and Combined | Ounces Silver per Ton |
| Inlet Pipe | 523 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 1 | 490 | 1,816.3 | 1,816.3 | 24.76 | 24.76 | 26.98 | 26.98 | 9.0 | 6.0 | 3.4 |
| 2 | 430 | 1,493.6 | 3,309.9 | 20.36 | 45.12 | 22.19 | 49.17 | 8.4 | 7.1 | 3.6 |
| 3 | 385 | 1,229.9 | 4,539.8 | 16.76 | 61.88 | 18.27 | 67.44 | 4.4 | 10.5 | 3.3 |
| 4 | 351 | 975.6 | 5,515.4 | 13.29 | 75.17 | 14.50 | 81.94 | 3.5 | 17.6 | 2.0 |
| 5 | 321 | 737.4 | 6,252.8 | 10.05 | 85.22 | 10.96 | 92.90 | 3.0 | 20.3 | 2.3 |
| 6 | 297 | 449.3 | 6,702.1 | 6.12 | 91.34 | 6.68 | 99.58 | 2.4 | 19.7 | 1.8 |
| Exstr. & Con. Pipes | 270 | 28.3 | 6,730.4 | 0.39 | 91.73 | 0.42 | 100.00 | 2.5 | 17.3 | 1.8 |

GENERAL DATA ON EXPERIMENTAL FLUE:

| | |
|---|---|
| Duration of test | 165 hr. 25 min. |
| Average velocity in inlet pipe | 3,200 ft. per min. |
| Approximate mean velocity of gases in flue | 454 ft. per min. |
| Average temperature of gases in flue | 380° F. |
| Average temperature of atmosphere | 45.5° F. |
| Frictional and other losses per 100 ft. of wires based on total friction through all wires | 0.52 in. of water |
| Total gas handled per test, volume at temperature in inlet pipe | 83,920,000 cu. ft. |
| Total gas handled per test, volume at 400° F | 73,372,000 cu. ft. |
| Pounds dust per million cubic feet gas | 91.73 |

GENERAL DATA ON MAIN FLUE:

| | |
|---|---|
| Average temperature of gases | 549° F |
| Average velocity of gases | 3,084 ft. per min. |
| Effective area of cross-section | 111 sq. ft. |
| Draft reading at point of velocity measurements | 0.67 in. of water |
| Cu. ft. gas passing up main flue per min., at observed temperature | 342,300 |
| Cu. ft. gas passing up main flue per min., volume at 400° F | 291,700 |
| Pounds dust carried by gases in main flue per 24 hr., as indicated by test | 47,230 |

Average number furnaces discharging gas into main flue—4 blast furnaces.

TABLE VII. TEST 41.—*General Summary of Observations and Results.*

| Section of Flue | Temperature of Gas F.° | Dust Collected, Lb. | | Pounds Dust per Million Cu. Ft. Gas Vol. at 400° F. | | Per Cent. of Total Dust Collected | | Assay of Dust | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | Per Section | Cumulative | Per Section | Cumulative | Per Section | Cumulative | Per Cent. Cu | Per Cent. $SO_2$ Free and Combined | Ounces Silver per Ton |
| Inlet Pipe | 577 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 1 | 540 | 5,559.2 | 5,559.2 | 77.61 | 77.61 | 30.46 | 30.46 | 7.6 | 4.0 | 3.2 |
| 2 | 472 | 3,875.7 | 9,434.9 | 54.11 | 131.72 | 21.24 | 51.70 | 7.4 | 4.0 | 3.2 |
| 3 | 425 | 2,351.8 | 11,876.7 | 32.84 | 164.56 | 12.88 | 64.58 | 6.6 | 4.4 | 3.2 |
| 4 | 393 | 2,121.0 | 13,907.7 | 29.61 | 194.17 | 11.62 | 76.20 | 6.2 | 4.8 | 3.0 |
| 5 | 365 | 2,584.5 | 16,492.2 | 36.08 | 230.25 | 14.16 | 90.36 | 5.7 | 5.7 | 3.0 |
| 6 | 335 | 1,687.6 | 18,179.8 | 23.56 | 253.81 | 9.25 | 99.61 | 4.8 | 7.5 | 2.8 |
| Exstr. & Con. Pipes | 304 | 71.8 | 18,251.6 | 1.00 | 254.81 | 0.39 | 100.00 | 2.1 | 15.9 | 1.6 |

GENERAL DATA ON EXPERIMENTAL FLUE:

| | |
|---|---|
| Duration of test | 164 hr. 55 min. |
| Average velocity in inlet pipe | 3,300 ft. per min. |
| Approximate mean velocity of gases in flue | 470 ft. per min. |
| Average temperature of gases in flue | 420° F |
| Average temperature of atmosphere | 55° F. |
| Frictional and other losses per 100 ft. of wires, based on total friction through all wires | 0.45 in. of water |
| Total gas handled per test, volume at temperature in inlet pipe | 86,382,000 cu. ft. |
| Total gas handled per test, volume at 400° F | 71,626,000 cu. ft. |
| Pounds dust per million cu. ft. gas | 254.81 |

GENERAL DATA ON MAIN FLUE:

| | |
|---|---|
| Average temperature of gases | 623° F. |
| Average velocity of gases | 3,277 ft. per min. |
| Effective area of cross-section | 111 sq. ft. |
| Draft reading at point of velocity measurements | 0.68 in. of water |
| Cu. ft. gas passing up main flue per min., at observed temperature | 363,700 |
| Cu. ft. gas passing up main flue per min., volume at 400° F | 288,800 |
| Pounds dust carried by gases in main flue per 24 hr., as indicated by test | 132,100 |
| Average number of furnaces discharging into main flue | 4 blast furnaces |

| | Test 40 | Test 41 |
|---|---|---|
| Dust in gases per 24 hr., based on total recovery | 38,530 | 105,970 |
| Dust in gases per 24 hr., based on escaping solids | 8,700 | 26,130 |
| Total per 24 hours | 47,230 | 132,100 |

Total extra dust per 24 hr., due to briquettes, 132,100 − 47,230 = 84,870 lb. The average dry weight of briquettes charged per 24 hr. was 371,660 lb., showing that about 23 per cent. of the material of which they were composed was lost in flue dust.

The curves on Plate XII show the rate of deposit and amount of dust collected for these two tests. The curve for test 39 is included for comparison.

The average weight per cubic foot of dust for tests 39, 40 and 41 is as follows:

| | Test | | |
|---|---|---|---|
| | 39 | 40 | 41 |
| Section 1 | 55.5 | 60.8 | 63.3 |
| Section 6 | 30.5 | 47.6 | 35.6 |

## V. Design of New Flue System.

Having determined the amount and character of gas to be handled and tried out various styles of baffles for hastening the settlement of dust, the practical problem was then presented of designing a flue system, including a chimney, which would carry all furnace gases, effectively settle the dust and provide for economically handling the same, allow for additional smelter capacity, if desired, and improve the draft conditions existing with the old system.

While all of these improvements were highly desirable and necessary from an operating standpoint, a further problem of considerable importance was the treatment and disposal of waste gases in such a manner as to avoid complaints from surrounding property owners. The advantages are well recognized of discharging smelter gases into the atmosphere at as high an elevation as possible, and for the condensation and settlement of fume, low temperatures are required, thus making a tall chimney a practical necessity.

An important limiting element in the design was the space available for locating a flue and dust chamber, of sufficient size to settle the dust

and at the same time be reasonably accessible for the removal and transfer of dust to the furnaces.

Considering the various methods tested, having in mind efficiency combined with minimum draft hindrance and general practicability, it

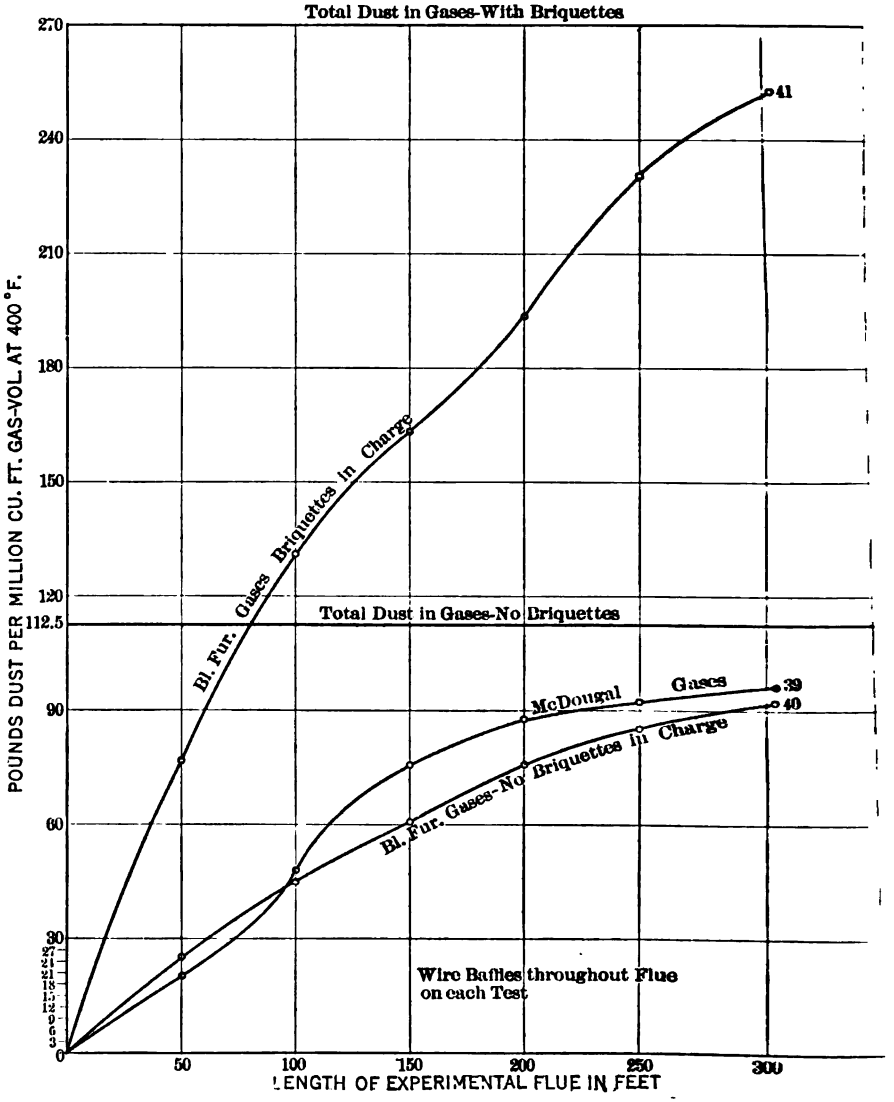



PLATE XII.—CUMULATIVE DUST CURVES.

was decided to use wire baffles and to make the cross-sectional area of the main dust chamber such that the velocity through it would not exceed 450 ft. per min.

Tests for volume and temperature made on individual department flues indicated that, if the plant were operated at full capacity and all gases were combined in one flue near the furnaces, the volume would be about 945,000 cu. ft. per minute at an average temperature of 550° F. In addition to this, an allowance was made for an increase of about one-third in the capacity of the plant, making a total of 1,260,000 cu. ft. of gas per minute based on old draft conditions. To cover an increased volume due to better draft conditions with the new system, we added an additional 25 per cent., making a total estimated volume to be handled of 1,575,000 cu. ft. of gas per minute. At a velocity of 450 ft. per minute this would require a flue having a cross-sectional area of 3,500 sq. ft. The dust chamber, as constructed is 176 ft. wide by 21 ft. high inside, there being 11 bays 16 ft. wide. The size of the flue connecting the main dust chamber to the chimney was established more or less arbitrarily. A velocity of about 1,500 ft. per minute was considered desirable, thus requiring an area of about 1,000 sq. ft. It was therefore decided to build a flue 48 ft. wide by 21 ft. high inside dimensions.

It can scarcely be said that the size of chimney, in so far as the diameter is concerned, was based on any very definite calculation. Some estimates were made on the amount of cold air necessary to bring all flue gases down to about 300° F., in case it was desired to condense the arsenic fumes; an allowance was then made for doubling the capacity of the plant, if required, and the diameter finally established at 50 ft. inside at the top, on the general principle that, in our experience, the greater part of smelter structures had proved too small after a few years' service and had been replaced by larger ones.

In arriving at the height of the chimney, however, a more definite calculation was made. From tests on the old flue system it was estimated that, with the better arrangement of flues contemplated in the new system, there would be required to overcome the frictional and other losses up to the point of entrance to the new dust chamber, a draft energy of 0.97 in. of water. In like manner, from tests on our experimental flue and other observations on the old system, the following total draft requirements were estimated:

| | *In. of Water* |
|---|---|
| Draft energy required at entrance of new dust chamber | 0.97 |
| Draft energy required in wire-baffled dust chamber | 0.85 |
| Expansion and contraction loss entering and leaving dust chamber | 0.30 |
| Chimney and flue from dust chamber | 0.96 |
| Total | 3.08 |

An average temperature of 450° F. was assumed from inlet of dust chamber to base of chimney, the height being 146 ft.

An average temperature of 350° F. was assumed in the chimney; atmosphere temperature 70° F. From these data, and assuming the gases to be air, it is found that the 146-ft. rise in flue from dust chamber to base of chimney will produce a draft energy of 0.78 in. of water. Deducting this from the total of 3.08 in. as found above leaves 2.3 in. to be produced by the new chimney. For an average temperature of 350° in the chimney this would require a height of 515 ft. A height of 500 ft. was therefore decided upon, but the actual height of the chimney as contracted for and built was 506 ft. A question that is sometimes asked is why the extra 6 ft. was added. In answer to this it may be explained that when specifications were made covering the construction of this chimney, only preliminary plans for the flue system had been made. These plans provided for an excavation 6 ft. in depth for the flues, at which elevation the brickwork for chimney was to begin, and, since it was desired to have the chimney 500 ft. above the surface of ground, a height of 506 ft. was specified. Later, these plans were changed so that the bottom of flues as well as base of chimney was at or slightly above the ground line, but the chimney was built 506 ft. high as per original specification.

The location of the chimney near the site of the old one was practically fixed by the topography of the country, while the location and length of dust chamber was largely determined, as already stated, by its accessibility and by the location of other structures and equipment.

Plate XIII[1] shows a general plan and arrangement of the new system. It should be explained at this point that the smeltery end of this plan will soon be a matter of history, since the number, size and location of furnaces will be entirely changed when reconstruction of the plant, which is now well under way, is completed. It does represent, however, the arrangement and conditions under which the new flue system has been working since it was put into operation on June 12, 1909.

In addition to the parts of the system already described, new department flues were built for the blast furnaces and for the MacDougall furnaces. The old converter flue joins the MacDougall furnace flue near its entrance to the main uptake. The system is arranged so that the reverberatory furnace gases can be bypassed around the main dust chamber, or be passed through it as desired.

All parts of the new system are equipped with hopper bottoms up to the dampers at upper end of the main dust chamber, so that the dust can be completely cleaned out while the flues are in operation.

In Plate XIII[1] is shown a cross-section through the blast-furnace department. Each furnace is connected to the flue by means of a steel pipe goose-neck, as shown. The end entering the flue is equipped with

[1] This is given as Fig. 3 of the paper of Corwin and Rodgers, Increasing the Efficiency of MacDougall Roasters, p. 1396, July *Bulletin*.

a hinged damper which is closed when the furnace is not in operation. The tracks under the hoppers in this flue are on the reverberatory-furnace charge-floor level, so that the regular calcine trammer's crew draws and trams this dust to the reverberatory furnaces without extra expense for labor. A similar arrangement is obtained with the MacDougall furnace flue, a cross-section of which is also shown.

The uptake, which carries the combined converter, MacDougall and blast-furnace flue gases, is rectangular in section, being 32 ft. 10 in. long

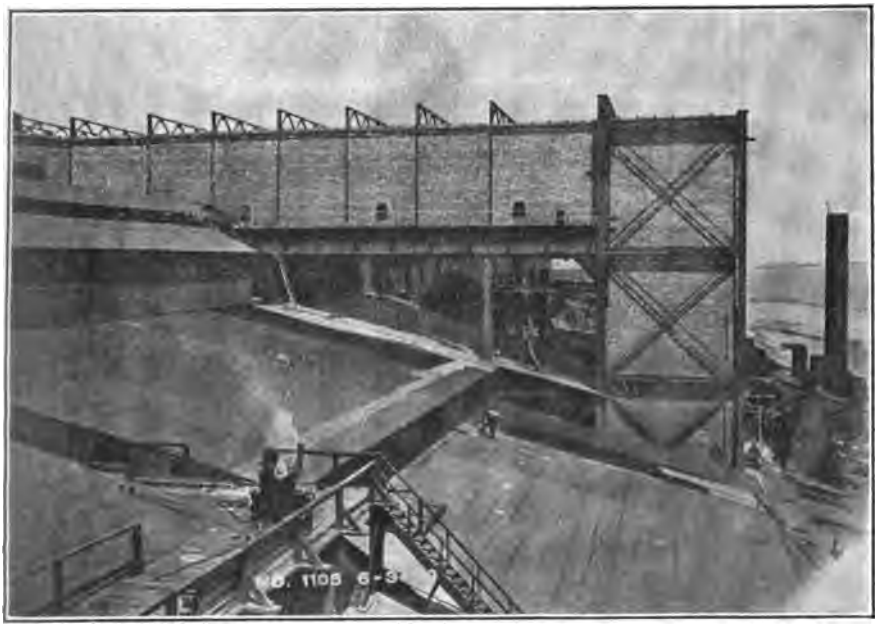


PLATE XIV.—UPTAKE AND CROSSTAKE.

by 20 ft. wide, inside dimensions, by 93 ft. 9 in. high from top of hoppers. The blast and MacDougall furnace gases enter from opposite sides of this uptake, the bottom of the MacDougall flue being about 9 ft. above the top of the blast-furnace flue. In order to prevent any baffling action of one upon the other and at the same time to help the two gases to mix more readily and thus prevent their being carried along their respective sides of the flue and dust chamber, a steel plate diaphragm was hung in the uptake, which has the effect of causing a layer of blast-furnace gases to travel along the bottom of the flue instead of one side, while the MacDougall furnace gases will travel along the top.

A section D-D of the cross-take flue which connects the uptake with the main dust chamber is shown in the plate referred to above. The entire bottom of this flue is equipped with hoppers, from which the dust

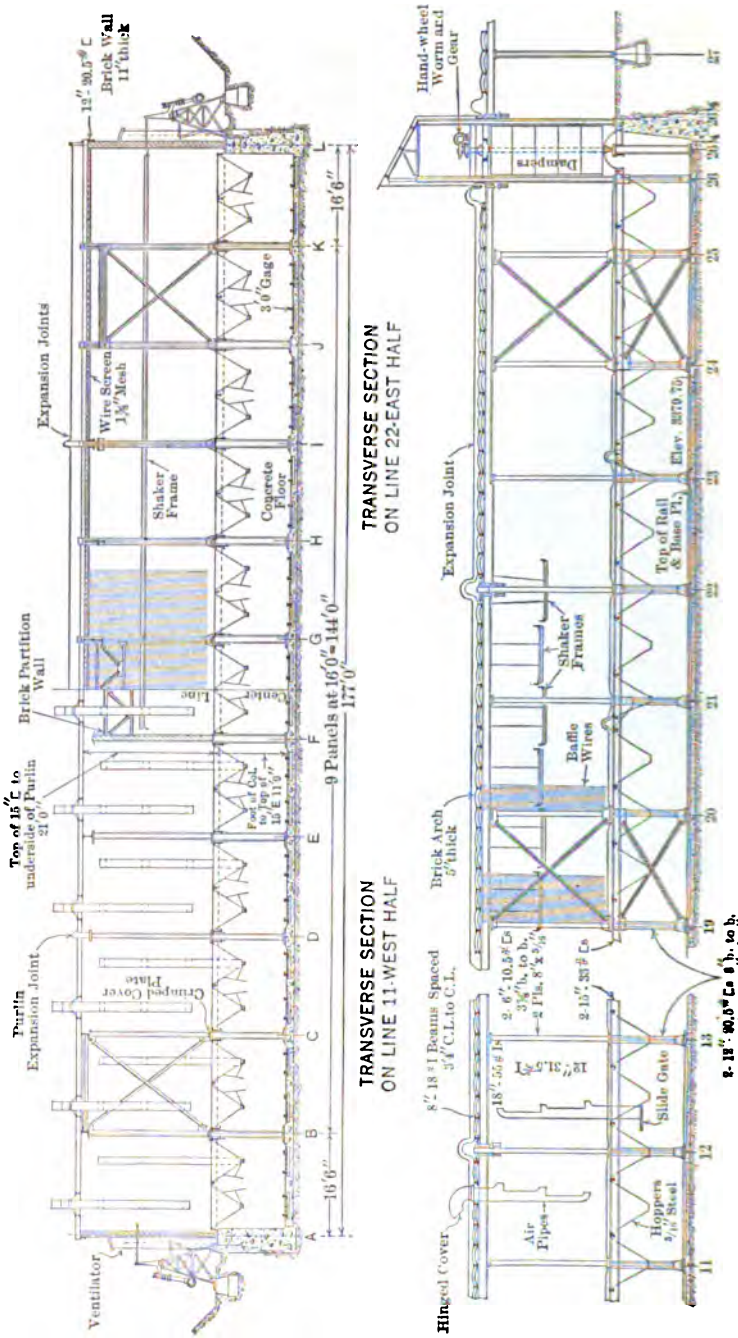



PLATE XV.—MAIN DUST CHAMBER.

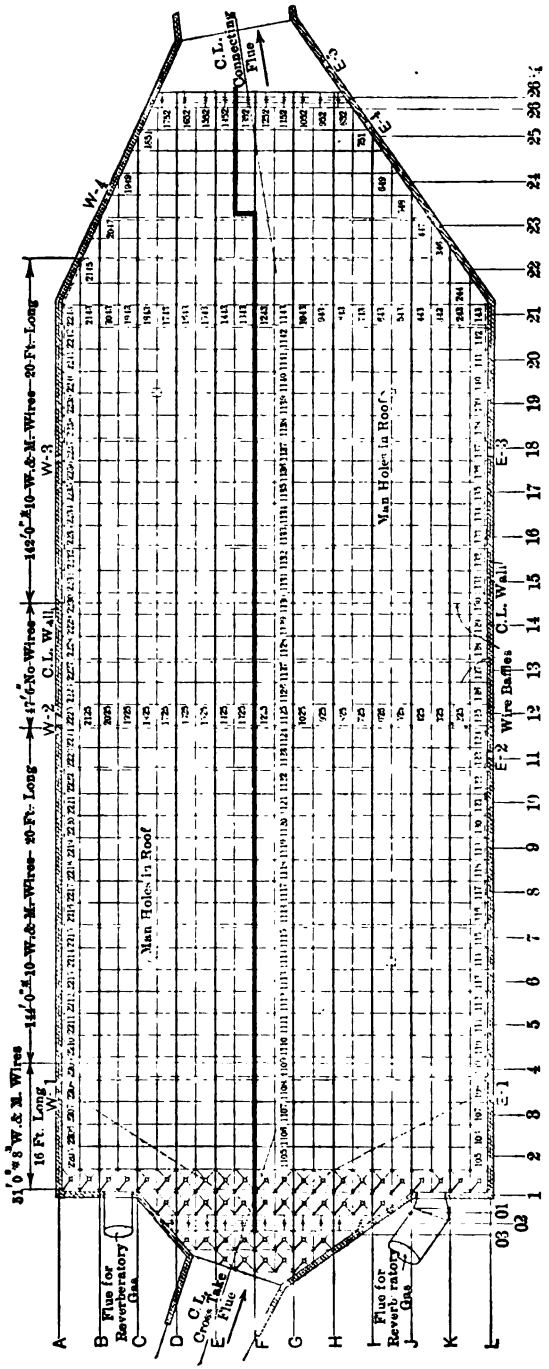



PLATE XVI.—HOPPER PLAN OF DUST CHAMBER.

is drawn into hoppers on a crane and is thereby transferred to chutes leading to the bottom of the uptake flue, where it is drawn out and handled in the same manner as described above. Plate XIV gives a general view of the uptake and cross-take.

In the flue connecting the main dust chamber to the chimney, no provision was made for removing the accumulated dust, the bottom being simply a level dirt floor, as shown in section E-E. Openings 3 ft. 3 in. by 5 ft., provided with hinged cast-iron doors, are located about every 110 ft. on the east side and about every 240 ft. on the west side. It will be noted from the general plan that this flue branches near its upper end, so as to enter the chimney at two points 90° apart. The openings in the base of the chimney are each 36 ft. high by 15 ft. wide. From the point where the flue branches, each leg grows narrower and higher as so to maintain a practically uniform sectional area. Section F-F shows a view of one branch, at a point near the chimney entrance.

## VI. Main Dust Chamber.

A cross-section and partial longitudinal section of the main dust chamber is shown in Plate XV. These sections are taken near the upper end of the chamber, in order to show special features of construction such as wire-shaking arrangement, cold-air pipes, etc. Other details of construction are the same throughout. Plate XVI shows a hopper plan of the dust chamber. The part within the dotted lines is hung with steel wires spaced at about 2.3 in. center to center, there being a total of about 1,200,000 wires, each of which weighs about 1 lb. As indicated on the plan, these wires are hung in two groups or divisions, there being about 47 ft. of clear flue between them. It will be noted that a clear space is left at the inlet end, affording an unobstructed passage for the gases to distribute themselves equally over the full width of the chamber. The total average length of flue filled with wires is approximately 317 ft. There are two sizes of wire used in the first group or division. For a distance of about 51 ft. from the inlet end, No. 8 W. & M. wires, 16 ft. long are used, the rest being No. 10 wires, 20 ft. long. The second division is hung with No. 10 W. & M. wire 20 ft. long.

At the upper end of the first division of wires, there are 22 pipes for the admission of air in case it should be desired to lower the temperature of the flue gases, with a view to condensing fume.

These pipes are shown on Plate XV and by photograph on Plate XVII, which plate also shows other structural details.

The open space between the two groups of wires is provided in order that a thorough intermingling of the cold air and gases can take place before entering the second division.

It will be noted that 11 of these air pipes enter from the top, or roof, of the flue and 11 through the bottom.

A brick partition wall, on the F line of columns, divides the chamber into two parts, there being six panels on the east and five on the west side. At each end of the chamber a line of butterfly dampers is provided

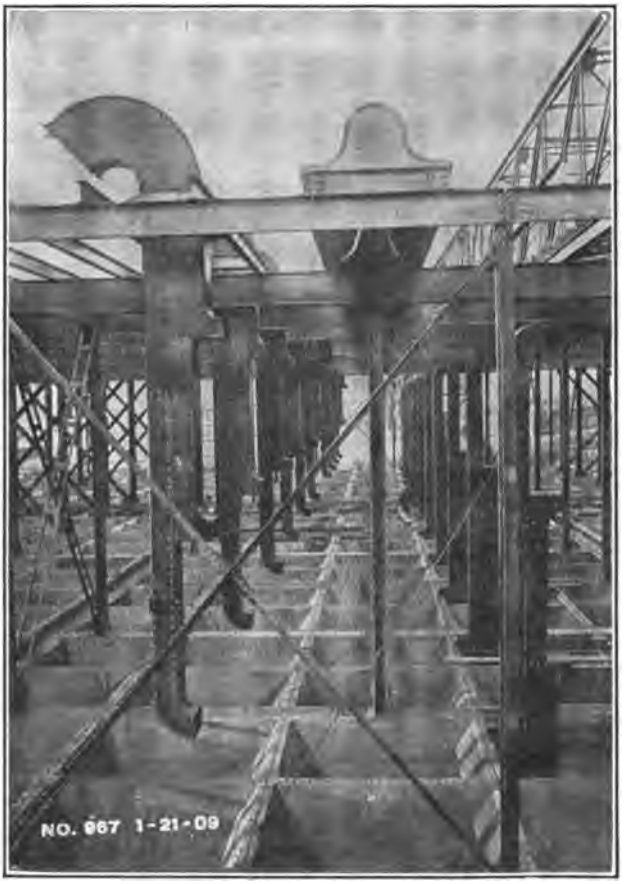



PLATE XVII.—VIEW OF COLD-AIR PIPES, EXPANSION JOINTS, AND HOPPERS.

so that either side, or the whole, of the chamber may be closed to the passage of gases. Plate XVIII shows these dampers partially erected. They are made of heavy cast-iron plates which are carried by a vertical shaft. The step bearing at the bottom is below the floor plate, where it is free from dust and can be oiled and inspected. The upper end of the shaft passes out through the roof of dust chamber, each one being equipped with a worm wheel, worm, and hand wheel for operating the damper.

Guided by the observations made in the experimental flue, which showed that the dust and condensed fume collected on the wires at temperatures about 400° and below, the upper division of wires in the new chamber was equipped with a device for shaking them in order to remove the condensed fume and dust. We estimated that the temperature in the first or lower division of wires would be above the point where the fume would condense and collect on the wires, but in this we were in error. The improved draft conditions provided by the new system resulted in the entrance of so much more cold air at the various furnace

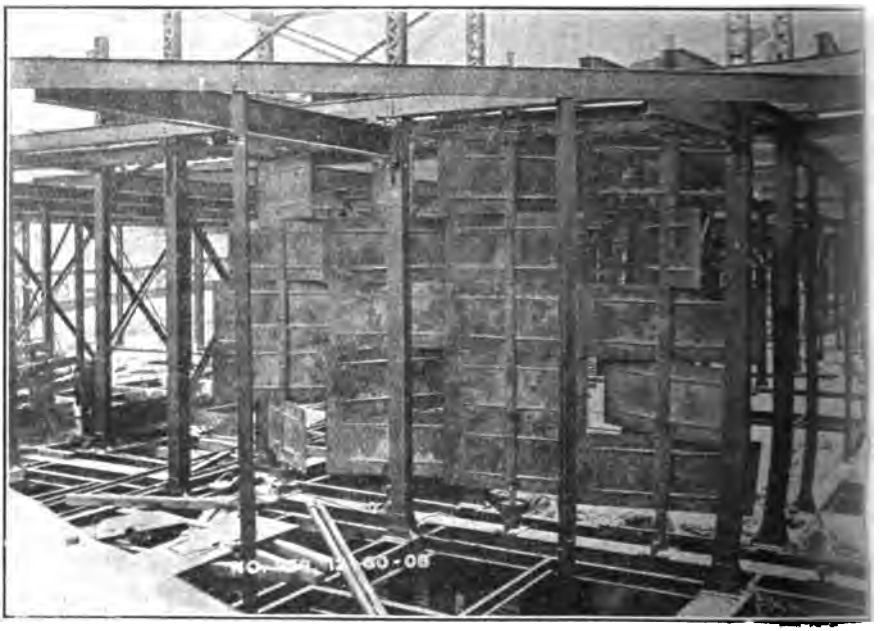


PLATE XVIII.—VIEW OF DAMPERS AT INLET END OF DUST CHAMBER (Partly erected

and flue openings than under the old system, that a temperature of under 400° F. exists in the first division of wires with the result that there is an accumulation of solid matter upon them. In order to remove this dust connections were made to the old stone dust chamber which carries the hot reverberatory gases that are normally passed around the new chamber. At intervals the lower dampers on one side at a time of the new chamber are closed and these gases are turned into that side. This results in volatilizing and disintegrating the material on the wires so that it breaks up and the greater part of it falls off.

Plate XIX shows the manner in which the wires are suspended. Crimped iron screen cloth 1⅝-in. mesh is stretched along the bottom of the roof purlins. Flat bars 2.5 in. by 5/16 in., bolted to the purlins

hold the screen in place. The warp or longitudinal wires are No. 8 W. & M. gauge, while the woof or cross wires are No. 10 W. & M. gauge. The baffle wires are provided with a hook in the form of a shepherd's crook on one end and are suspended from alternate intersecting points on the screen. This arrangement has the effect of staggering the wires which are about 2.25 in. apart, making the baffling action more effective.

The device for shaking the upper division of wires consists of angle-iron

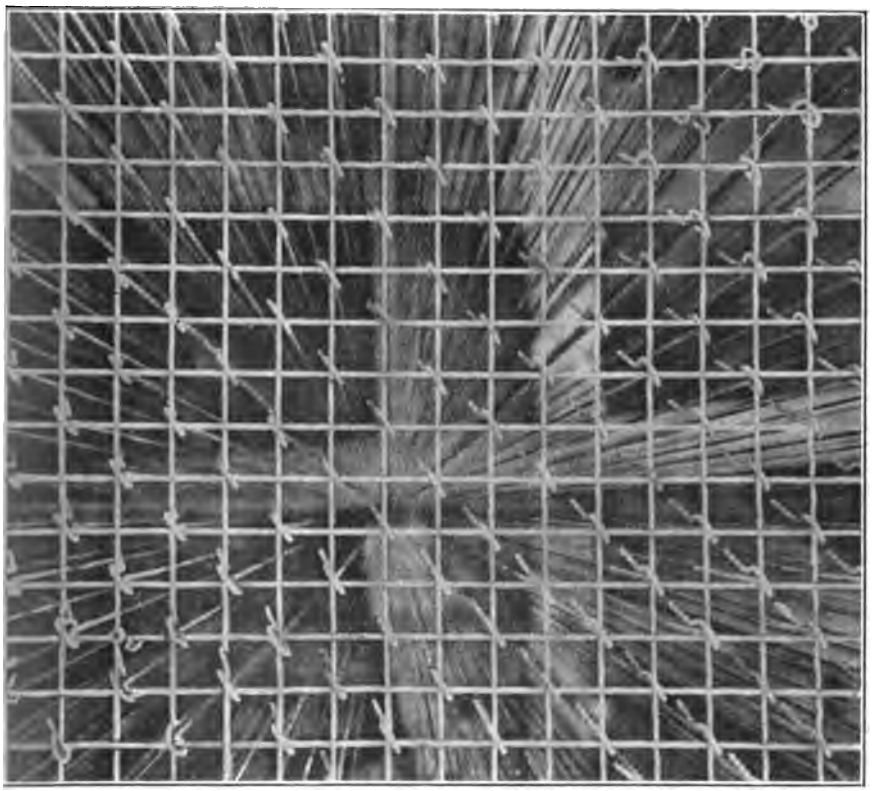


PLATE XIX.—METHOD OF SUPPORTING BAFFLE WIRES.

frames suspended from the roof beams by hangers about 10 ft. in length, thus bringing the frames at about the middle point of the baffle wires. On these frames a wire netting having a 4 by 7 in. mesh is stretched, through which the baffle wires pass. The frames are 10 ft. wide and extend from the side walls to near the partition wall. A connecting rod attached to each frame passes out through the flue wall and is connected to a bell-crank lever which is actuated by an eccentric, giving the frame a stroke of 9.5 in. The frames make about 60 strokes per minute.

Line shafts, carrying the eccentrics on each side of the flue, are operated

by electric motors. The eccentrics are set at different angles in order to prevent vibrations which might be produced in the structure if the complete group of wires was moving in unison.

The wires are shaken for a period of about 30 min. at intervals of from 60 to 90 days as may be required. While the shaking device on one side of the chamber is in operation all gases are passed through the other side. The upper dampers are closed on the side where the wires are being shaken, thus guarding against the loss of any dust removed from the wires.

Plates XX and XXI show the wire-shaking device during construction.

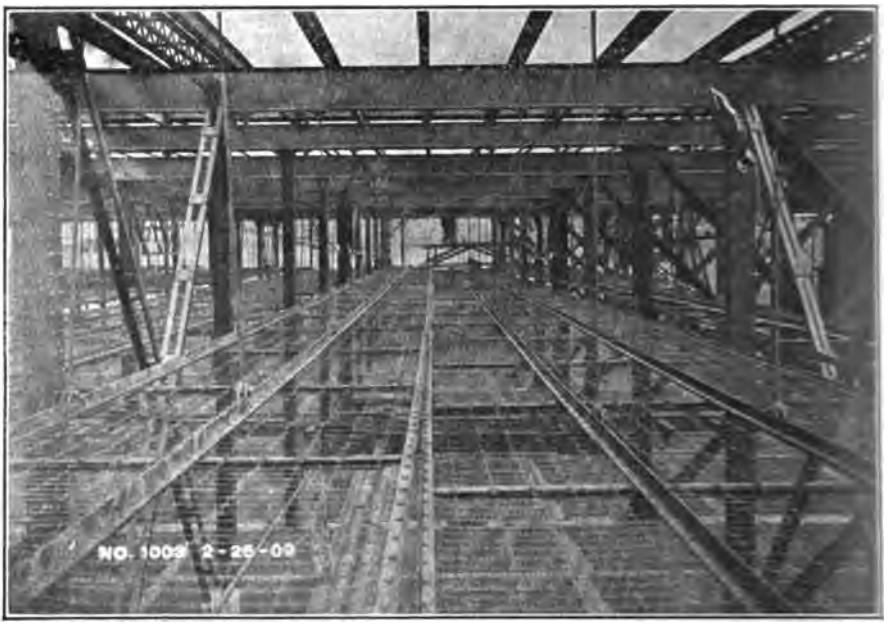



PLATE XX.—DEVICE FOR SHAKING WIRES. (Baffle wires not in place.)

## VII. SOME DETAILS OF CONSTRUCTION.

Throughout the flue system a structural steel framework is used. The inner columns are of box section, being built up from plates and channels, while the columns in the outside wall are 12-in. I-beams, between which a brick curtain wall 11 in. thick is laid. The main roof beams resting on top of columns are chiefly I-beams, channels being used only at the expansion joints.

The roof consists of steel I-beam purlins between which brick arches 5 in. thick are sprung. Special perforated bricks were made for this purpose, the skewback brick being molded to fit the I-beams, while all others were made radial to suit the arches. The valleys between the skewback brick and top of I-beams were filled with concrete (see Fig.

1, Plate XXII) and, when the roof was completed, it was given two coats of a one-to-one concrete wash, in order to fill up all cracks and prevent leakage of air or water.

For the walls of the flue, which are 11 in. thick, special perforated brick were also made. These brick were 11 in. long by 5⅜ in. wide by 4⅜ in. thick. They are laid up to break joints, one row of headers to

PLATE XXI.—WIRE-SHAKING MACHINERY AND GENERAL VIEW OF FLUE SYSTEM.

four of stretchers, and were all made at the plant which was constructed for the manufacture of brick for the chimney.

Each column in the dust chamber, for a distance of about 180 ft. from the inlet end, was provided with a hole in the base and another near the top, so that it could be cooled by allowing cold air to blow through it into the flue, in case the temperature should become high enough to endanger its strength. We have never found it necessary to use these openings.

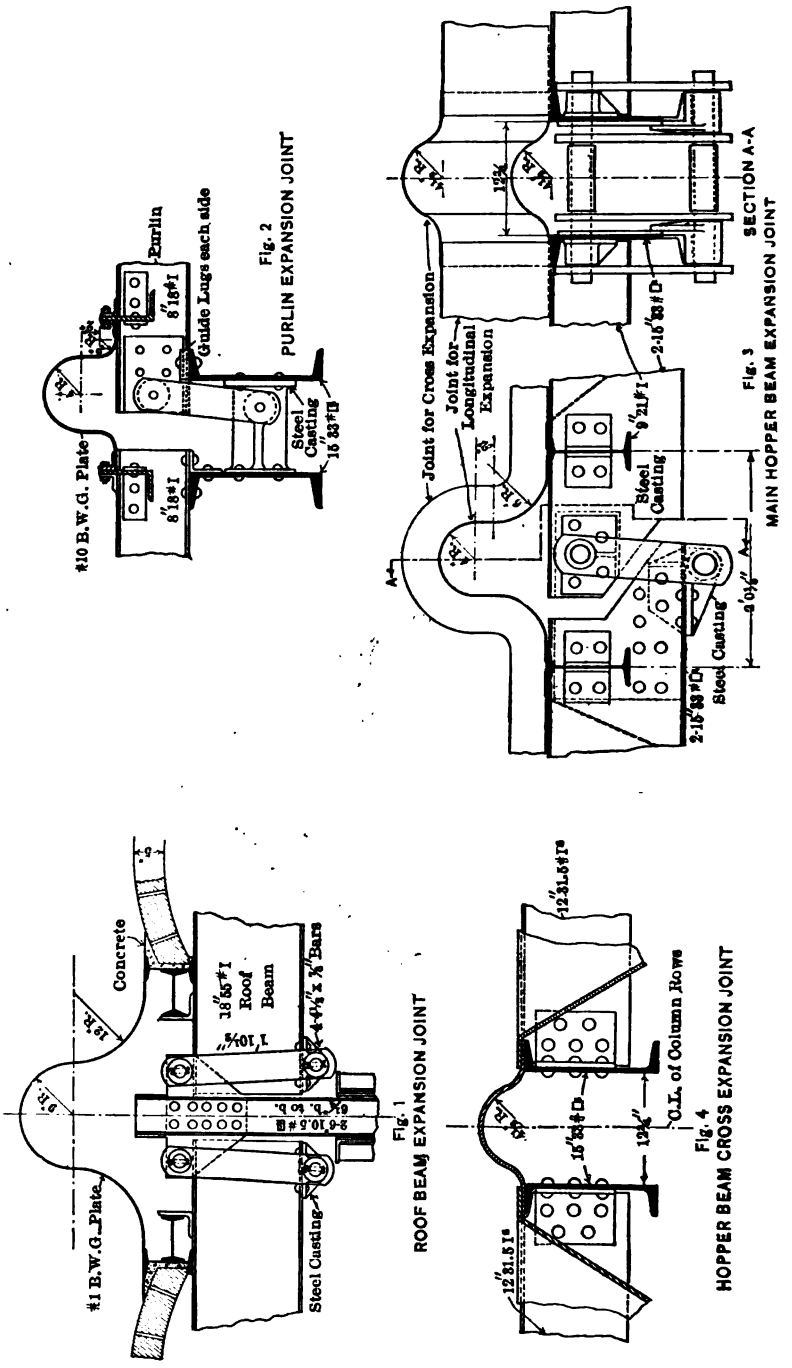



Fig. 2 PURLIN EXPANSION JOINT

Fig. 3

Fig. 1 ROOF BEAM EXPANSION JOINT

Fig. 4 HOPPER BEAM CROSS EXPANSION JOINT

PLATE XXII.—DETAILS OF EXPANSION JOINTS.

In a structure of this size and character, where a goodly part of the steel is exposed to high and varying temperatures, we considered it very essential to the life and maintenance of the structure to make ample provision for the expansion and contraction of the steel. We were also advised by our friends in Anaconda, who had experienced some misfortunes with similar structures, to make all parts of good heavy construction. We believe that it is to the exercise of these precautions that we

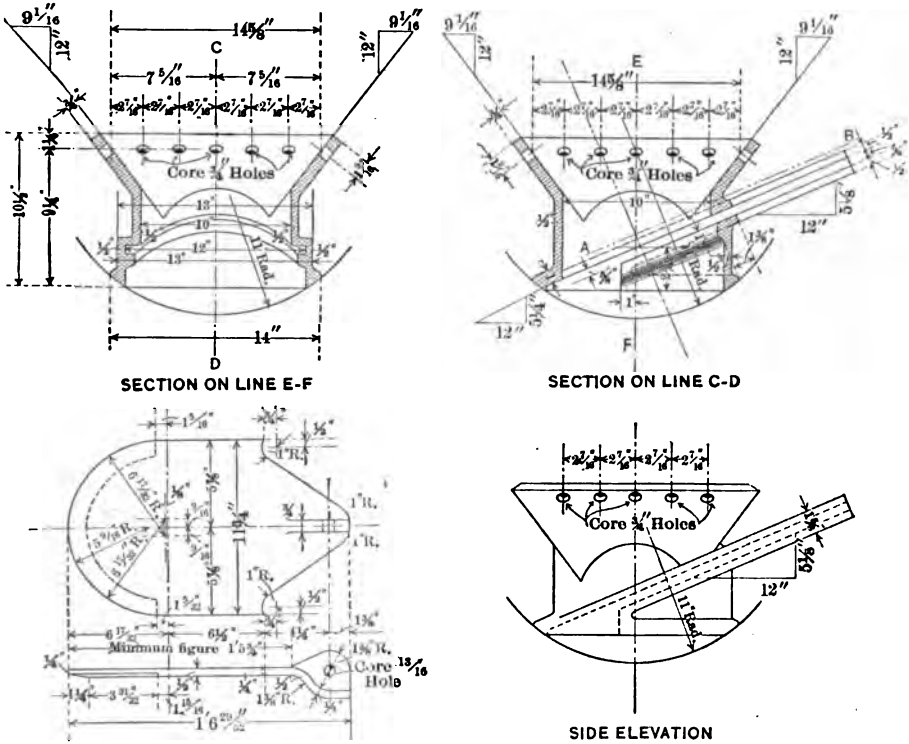



PLATE XXIII.—HOPPER GATE AND VALVE.

are able to say at this time, after the flue has seen four years of continuous service, that it stands as true and plumb as upon the day it was erected.

To provide for this expansion the framework is built in units of from three to seven panels each. The middle panel is securely braced, which holds it plumb, the expansion taking place in both directions from this central point. The roof beams are riveted to the tops of all columns except the one midway between the braced panels, where they are suspended from the columns by links as shown in Plate XXII, Fig. 1. The roof between the purlins on each side of this expansion joint is made of No. 10 steel, bent as shown in the figure. These joints extend across the full width of the flue.

To take up the expansion in the purlins which extend across the flue two expansion joints are provided running the full length of the dust chamber, thus dividing the roof into three sections as indicated in the sectional elevation Plate XV. The middle panel in each of these sections is securely braced, so that the expansion is taken up at the joint, a detail of which is shown in Fig. 2, Plate XXII.

The main-hopper beams are 15-in. channels, which are riveted to the sides of the lower columns. A detail of the expansion joints in these beams is shown in Fig. 3, Plate XXII.

In the longitudinal section shown in Plate XV it will be seen that

PLATE XXIV.—HOPPERS UNDER DUST CHAMBER.

there are two hoppers to each longitudinal panel, the center line of the hoppers coinciding with the center line of the panel and of the columns.

The object of this arrangement is to allow the expansion of the cross-hopper beams to be taken up by the spring in the main longitudinal beams. A crimped plate covers the space between the main channel beams as shown in Fig. 4, Plate XXII.

A part of the structure which might seem of minor importance, but which is operated more frequently than any other single element, is that of the gate on the bottom of the hoppers. When it is considered that there are 1,042 hoppers under the main dust chamber, 100 under the cross-take flue, 74 under the blast-furnace flue, 12 under the MacDougall flue

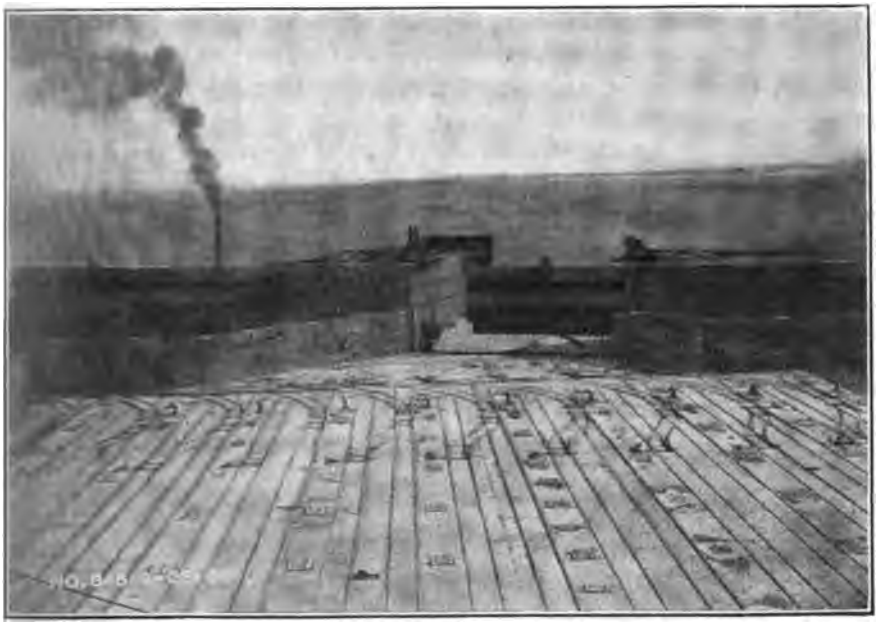


PLATE XXV.—TRACKAGE UNDER DUST CHAMBER.

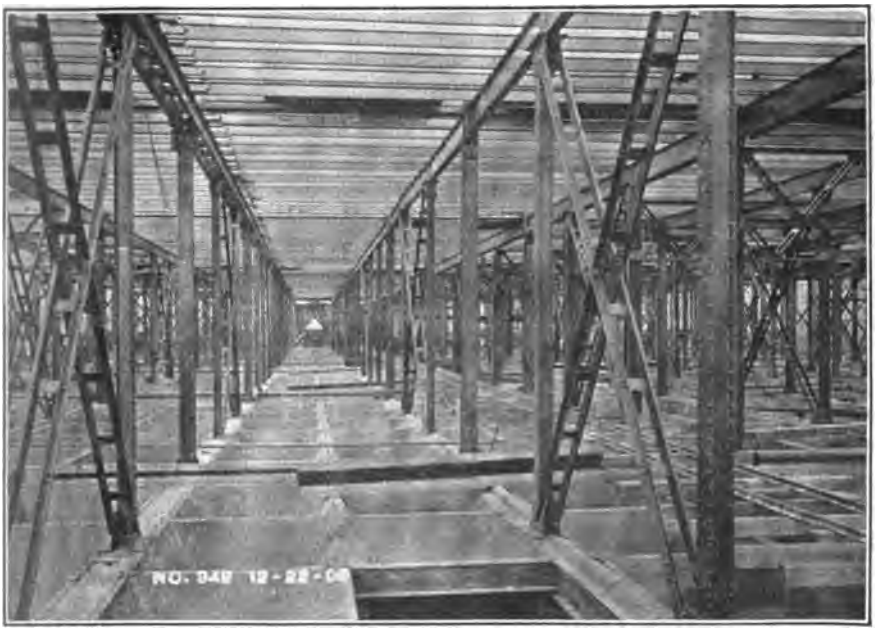


PLATE XXVI.—INTERIOR OF DUST CHAMBER DURING ERECTION.

and eight under the uptake, some of which are being operated every day, the desirability of an efficient and easily operated hopper gate is appreciated. Plate XXIII shows the gate which was designed for this work. It has given extremely good satisfaction. It is made of cast iron, all holes being cored so that it required no machine work of any kind. The essential feature of this gate is that there are no closed slots to fill up with dust and cause the slide to stick or to prevent it from closing.

The hoppers and tracks under the dust chamber are shown in Plate

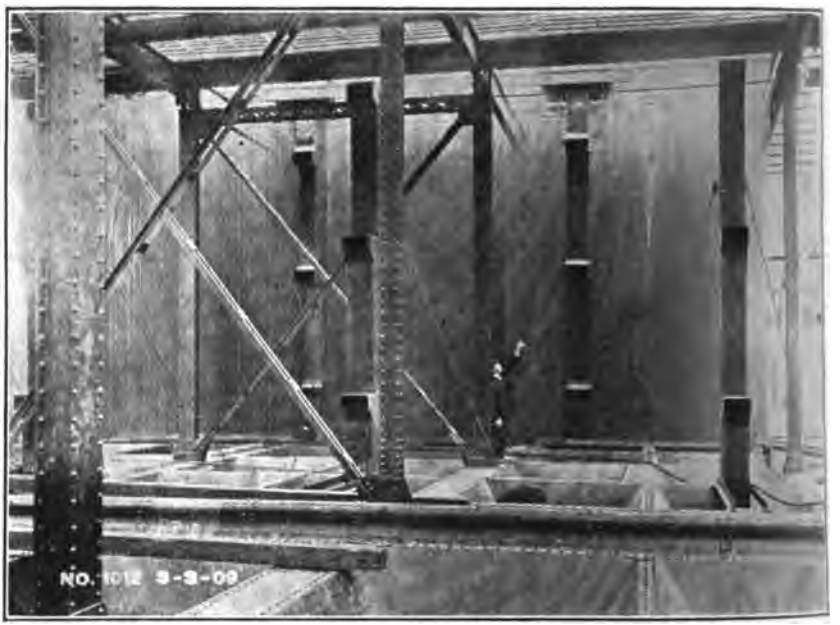



PLATE XXVII.—INTERIOR OF DUST CHAMBER, SHOWING WIRES AND COLD-AIR PIPES.

XXIV, and Plate XXV shows a portion of the track system before any steel was erected. The total length of trackage under the dust chamber is about 8,900 ft.

Dust is transferred from the main dust chamber to the furnaces in hopper-bottom cars of about 80 cu. ft. capacity. The over-all length of each car is just equal to the span on one hopper, so that one spotting only of the train is necessary for each loading.

Some additional details of dust-chamber construction are shown on Plates XXVI and XXVII.

## VIII. The Chimney.[1]

Having decided upon the construction of a chimney 506 ft. high and 50 ft. inside diameter at the top, the contract for furnishing the brick and building the chimney was let to the Alphons Custodis Chimney Construction Co., of New York, who designed and erected the structure. The foundation was put in by the owners, the design and construction having the approval of the chimney contractor, who is responsible for the stability of the whole structure.

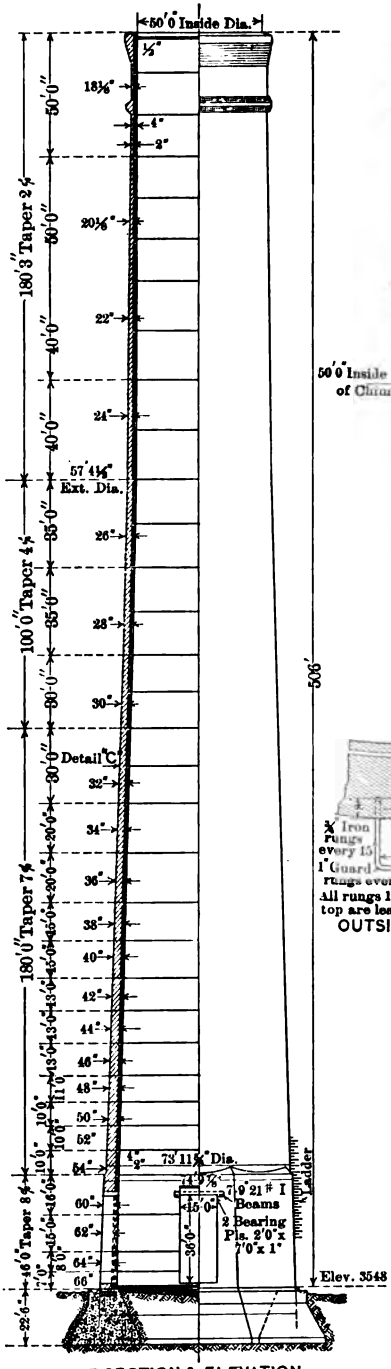
The chimney is designed to withstand a wind velocity equivalent to 33 1/3 lb. per square foot diametral projection. The total estimated weight of brickwork in the completed chimney is 17,000 tons. The thickness of the shell is so proportioned that the combined dead and wind load do not exceed 21 tons per square foot at any point. There is no tension at any point on the windward side. Specifications for the brick of which the chimney is constructed required that they withstand a test pressure of not less than 3,750 lb. per square inch of total sectional area, including perforations. General drawings of the chimney and of some details of construction are shown on Plate XXVIII, while Plate XXIX. is a view of the completed structure.

The whole of the interior is circular in section. The exterior is octagonal for a height of 46 ft. above the base, the remainder being circular.

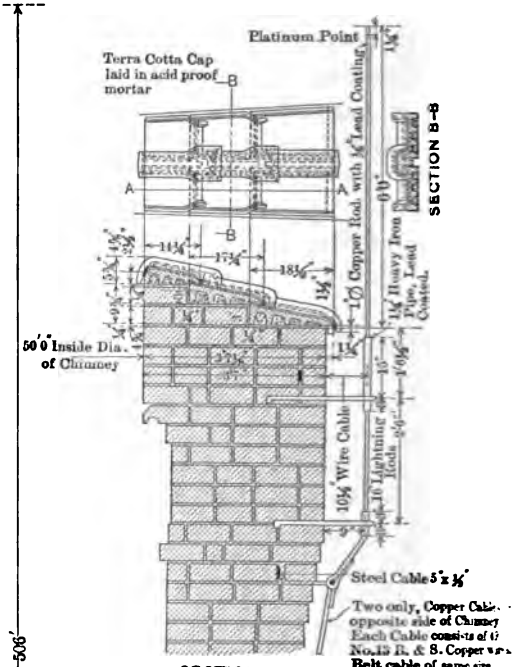
The profile of the chimney presents four separate tapers as follows: The upper 180 ft. is 2 per cent. taper (1 per cent. batter); the next lower 100 ft. is 4 per cent. taper (2 per cent. batter); the next lower 180 ft. is 7 per cent. taper (3.5 per cent. batter); and the octagonal base 46 ft. in height is 8 per cent. taper (4 per cent. batter). The wall thickness varies by steps of 2 in., the sections of constant thickness being of variable heights, ranging from 10 ft. at the bottom of the circular section, where the wall is 54 in. thick, exclusive of lining, to 50 ft. at the top, where the wall is 18⅛ in. thick.

The chimney is lined throughout with a 4-in. course of hard-burned brick laid in acidproof mortar, the joints being made as thin as possible. This lining is what is known as sectional construction, each section resting on a corbel ring built out from the inner wall of the main chimney shell. No section of lining exceeds 20 ft. in height. An air space of 2 in. is left between lining and main chimney shell. Fig. 1, Plate XXVIII, shows a detail of corbel and lining. The top brick of the corbel is molded with a lip for the purpose of forming a drip outside of the lining. In order to

[1] New 506-ft. Chimney at Great Falls Smelter, *Engineering and Mining Journal*, vol. LXXXVII, p. 156. The World's Largest Chimney, *Engineering News*, vol. LX, p. 583. The Chimney of the B. & M. C. C. & S. M. Co., *Engineering Record*, vol. LVIII, p. 600. The World's Largest Chimney, *Mines and Minerals*, vol. XXX, p. 257.




HALF SECTION & ELEVATION OF CHIMNEY & FOUNDATION

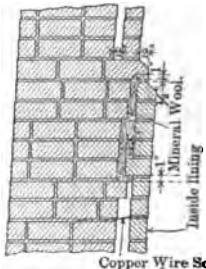



SECTION A-A

Fig. 2

All rungs 100 ft. down from top are lead covered.

OUTSIDE LADDER




DETAIL "C"

SHOWING CORBEL AND METHOD OF SUPPORTING LINING

Fig. 1

PLATE XXVIII.—DETAILS OF 506-FT. CHIMNEY.

prevent any dust or condensed fume from settling in the space between lining and chimney wall, this space was filled with mineral wool for a depth of about 2 ft. at the top of each section. The acidproof mortar, which is composed of silicate of soda, asbestos wool and other ingredients, is a very slow-setting material and, when fresh, acts as somewhat of a

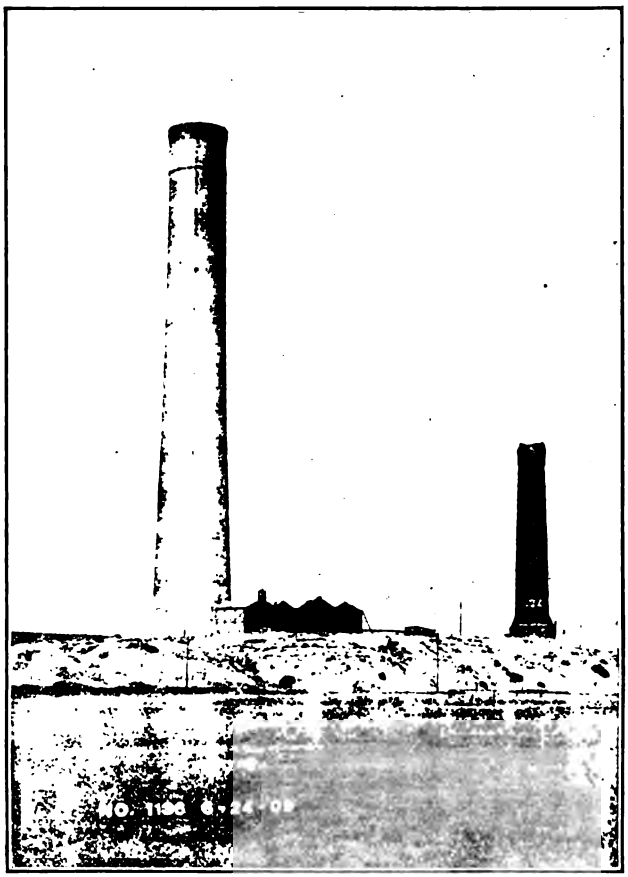


PLATE XXIX.—GREAT FALLS CHIMNEY, 506 FT. HIGH.

lubricant between adjacent courses of brick. It is, however, extremely hard when thoroughly set and dry. In order to hold the lining course in place until the mortar had set, small sections of copper wire netting were built into the main shell and lining at intervals of 7 or 8 ft.

The mortar used in the main part of the chimney was a 1:2:5 cement-lime-sand mixture. For a distance of 50 ft. down from the top, all outside joints in brickwork were pointed up with acidproof mortar and the top 3 ft. of the main wall was laid with this mortar throughout.

The top of the chimney is corbeled out in steps of 0.75 in. to a thickness of 42 in. The cap is composed of specially shaped radial interlocking terra-cotta tiles laid in acidproof mortar. Fig. 2, Plate XXVIII, shows the general design and arrangement of this cap.

For protection against lightning, 16 rods are provided, each of which extends 5 ft. above the chimney top. They are made of 1-in. round copper rod, lead coated for protection against acid, and tipped with a platinum point 1.25 in. high. These rods are all connected to a copper cable which encircles the chimney a few feet below the top. From this encircling cable two ⅝-in. copper cables lead to the ground on opposite sides of the chimney. The lower end of each cable is fastened to a copper plate about 6 sq. ft. in area, which is buried several feet under ground and at a distance from the foundation, at a point where moisture is usually present. The cable band, as well as the vertical cables for a distance of about 100 ft. down from the top, are lead covered. The cables are composed of 49 No. 13 B. & S. gauge copper wires. Fig. 2, Plate XXVIII, shows the general arrangement and method of supporting the lightning rods.

An outside ladder built into the brickwork extends from the base to the top. It consists of one-piece rungs made of 0.75-in. round iron, which stand out 6 in. clear from the chimney wall. The hooked ends of these rungs extend 6 in. into the chimney wall. They are placed at every third joint, or about 15 in. apart. At every second rung there is a guard rung or loop made of 1-in. round iron. The hooked ends extend 8 in. into the brickwork, and the loop is 16 in. outside of the ladder rung and is 28 in. wide inside. For a distance of 100 ft. down from the top, the ladder and guard rungs are lead covered.

At each corbel point on the circular portion of the chimney, a flat steel cable about 5 in. wide by 0.5 in. thick is laid in the brickwork. These are for the purpose of reinforcing the shell at the corbel points, where a small portion of it is exposed to the heat of the gases. The cables used were old hoisting cables that had been discarded at the mines.

There are four openings in the base of the chimney, each of which is 36 ft. high by 15 ft. wide. The brickwork across the top of each of these openings is carried by seven 9-in. 21-lb. I-beams, which rest on 1-in. plates at each end, and are protected from the acid fumes by special brick fitted over the flanges. Only two of these openings are in use at the present time, the other two being bricked up with double curtain walls which are flush with the inside and outside of the chimney wall. The outside, or facing, course of brick in all flue openings is laid in acid-proof mortar, and a course of brick 1 ft. thick, laid in the same material, covers the floor of the flue openings and protects the concrete base at this point. Plate XXX is an interior view at the base of the chimney.

PLATE XXX.—CHIMNEY BASE INTERIOR.

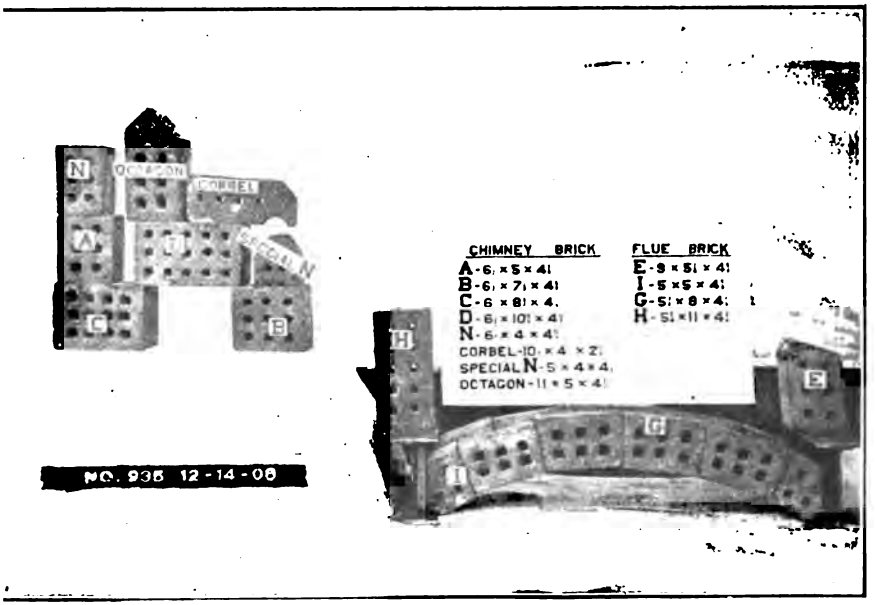



PLATE XXXI.—BRICK USED IN CHIMNEY AND FLUES.

showing the flue entrances. It also shows several corbel rings and a portion of the paved floor which is described later.

In order to break joints and also to obtain the required variations in the thickness of the walls, five sizes of brick were used in the main chimney shell. These are known as perforated radial brick. The holes are about 1 in. square and are in a vertical position when the brick are laid. Plate XXXI shows the various sizes and shapes of brick used in the construction of the chimney and flues.

To supply these brick, a plant was erected near the chimney site, equipped with the necessary grinding pans, etc., an auger machine of over 100 tons capacity in 8 hr., a drier which is arranged to use either direct or waste heat, and eight circular beehive kilns, independently fired. Altogether, the plant has a capacity of 100 tons per day and is being operated at the present time for the manufacture of such products as paving brick, fire brick, hollow tile, drain tile, etc., as well as common and perforated building brick. The shale of which the brick was made was quarried with steam shovel, near the plant. An analysis of this shale is as follows:

| Loss on Ignition | Insoluble | $SiO_2$ | $Al_2O_3$ | FeO | CaO | MgO | $SO_3$ |
|---|---|---|---|---|---|---|---|
| 8.02 | 77.8 | 58.3 | 19.0 | 6.3 | 1.2 | 2.3 | 0.74 |

Crushing tests were made on several samples of brick, with results as shown in the following table. In all cases pressure was applied parallel to the perforations. In calculating the crushing weights, the gross area of the brick was used, no deduction being made for the area of holes.

| | Pressure in Lb. per Sq. In. | |
|---|---|---|
| | First Crack | Ultimate |
| Hard burned | 3,033 | 7,173 |
| Medium burned | 3,064 | 4,255 |
| Soft burned | 2,766 | 3,949 |

*Chimney Foundation.*

The chimney foundation is shown on Plate XXXII. It is composed of concrete, and is annular in shape, the inside being circular, and the outside octagonal.

It contains 4,300 cu. yd. of concrete composed of a 1:2.3:4.5 mixture of cement, tailing sand from the concentrator, and broken slag from the smelter. A drainage gutter near the top having a slope of ⅛ in. per foot

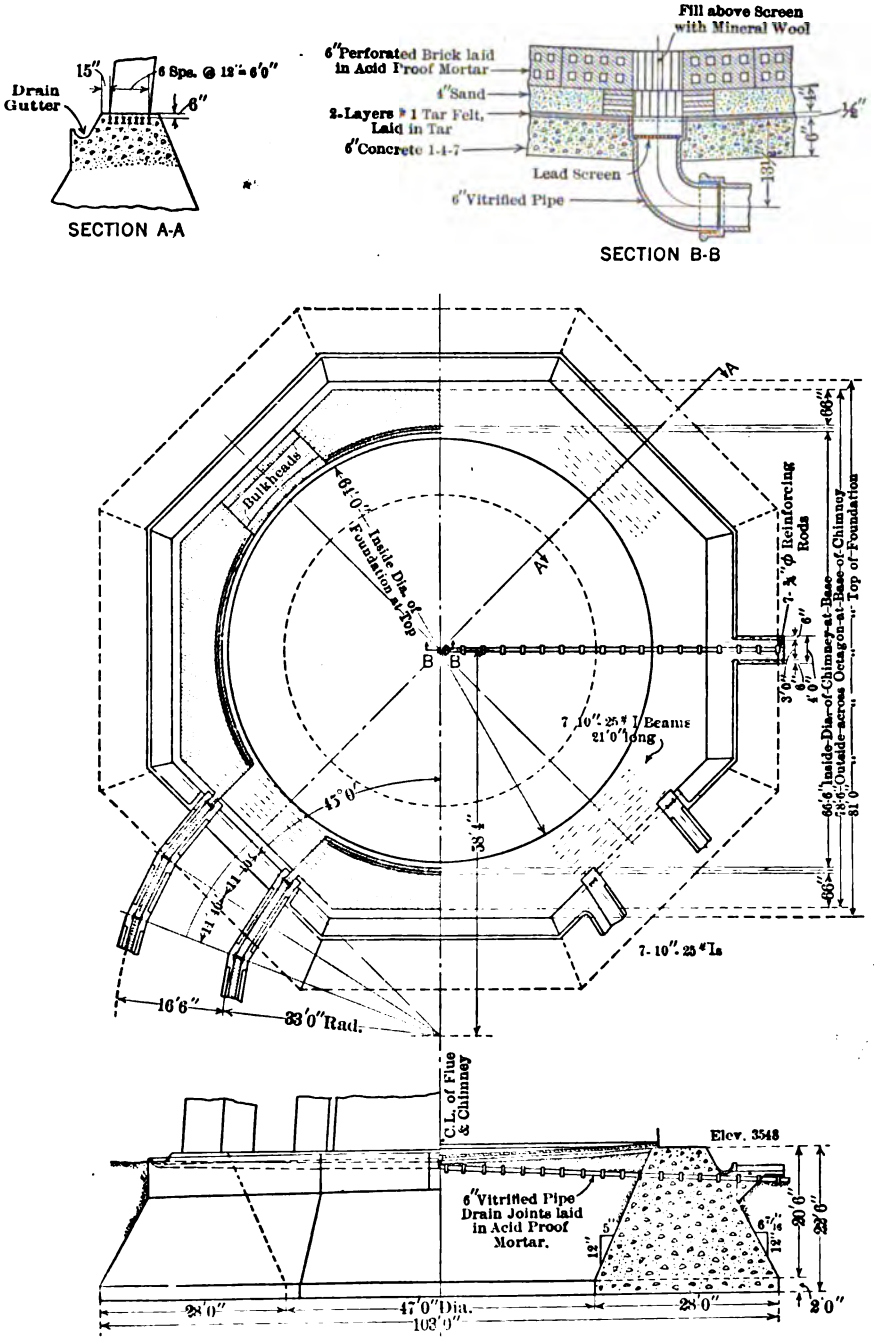



PLATE XXXII.—FOUNDATION FOR 506-FT. CHIMNEY.

was built as a part of the foundation. The footing on which the foundation rests is a hard shale. A loading test was made on the bottom of the excavation to determine whether it would withstand the required maximum pressure which it was estimated would be 5.25 tons per square foot. Four cast-iron plates 2 ft. square by 3 in. thick were set in a thin

*Flue System—Construction Data.*

| | Cu. Yd. |
|---|---|
| **Excavation:** | |
| Main dust chamber | 90,354 |
| Connecting flue | 14,800 |
| Chimney | 5,500 |
| Track to smelter | 10,000 |
| Total | 120,654 |
| **Concrete:** | |
| Chimney | 4,300 |
| Flue system | 7,933 |
| Total | 12,233 |

| | Tons |
|---|---|
| **Steel and Iron Work:** | |
| Structural steel | 3,412 |
| Wire baffles (1,200,000) | 600 |
| Wire netting (89,400 sq. ft.) | 35 |
| Castings | 203 |
| Total | 4,250 |

| | | Cu. Ft. | Tons | Equivalent in Common Brick |
|---|---|---|---|---|
| **Brickwork:** | | | | |
| Blast-furnace dust chamber | | 15,968 | 926 | 351,000 |
| MacDougall dust chamber | | 6,505 | 377 | 143,000 |
| Uptake | | 13,100 | 760 | 288,000 |
| Cross-take | | 7,977 | 463 | 176,000 |
| Main dust chamber | | 63,356 | 3,675 | 1,394,000 |
| Connecting flue | | 91,866 | 5,328 | 2,021,000 |
| Chimney | | 293,104 | 17,000 | 6,441,000 |
| Shell | 261,429 | | | |
| Lining | 29,948 | | | |
| Floor | 1,727 | | | |
| Totals | | 491,876 | 28,529 | 10,814,000 |

| | Ft. |
|---|---|
| **Trackage:** | |
| Under dust chamber (40-lb. rail) | 8,900 |
| Dust chamber to smelter (55-lb. rail) | 4,300 |
| Under blast furnace flue (40-lb. rail) | 700 |
| Total | 13,900 |

cement grout and at equal elevations. These plates were at the corners of a rectangle 6 ft. 6 in. by 16 ft. 6 in. Plate girders were then symmetrically placed on these plates, and the whole loaded with rails sufficient to produce a load 6.5 tons per square foot of bearing area of the cast-iron plates, being about 24 per cent. in excess of the maximum calculated pressure. The test was continued for a period of 20 days, during which time

heavy rains flooded the excavation to a depth of several inches over the bearing plates, making the test an especially severe one. The total settlement of the bearing plates was 0.058, 0.058, 0.075, and 0.088 ft. respectively, from which it was decided that the footing material was safe for the required loading.

To prevent possible damage to the concrete foundation by the action of leachings from dust on the floor or sides of the chimney, in case the plant should be shut down for an extended period, the floor was given a slope of 0.5 in. per foot to the center, from which a vitrified pipe laid in acidproof cement leads to the outside of the foundation. The floor itself is constructed from the bottom up as follows: 6 in. of concrete footing; two layers of No. 1 felt with a good coating of tar under, between, and on top of same; 4 in. of sand to act as a bed for brick flooring and also to prevent the heat of gases from damaging the tar and felt; and 6 in. of good quality perforated brick laid in acidproof mortar. The details of this construction are shown on Plate XXXII.

## IX.—Operation of New Flue System.

The new flue system was put into operation on June 12, 1909. It has been in service continuously ever since, with the exception of a period of about 40 days in December, 1909, and January, 1910, when the plant was shut down on account of a strike among employees of the railway company, which stopped the delivery of ore and other necessary material.

From June 12, 1909, to Jan. 1, 1913, the total amount of dust collected in the various parts of the system, as well as the average analysis of the dust covering the whole period of 41 months, is shown in Table VIII.

Table VIII.—*Quantity and Analyses of Flue Dust.*

| Name of Flue | Tons Dust | | Cu Per Cent. | Ag Oz. per Ton | Au Oz. per Ton | Insol. | $SiO_2$ | FeO | $Al_2O_3$ | CaO | S |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | Total for 41 Mo. | Average per Mo. | | | | | | | | | |
| Blast furnace | 87,020 | 2,122 | 8.08 | 2.7 | 0.019 | 34.7 | 26.8 | 33.0 | 7.1 | 1.7 | 16.1 |
| MacDougall furnace | 18,741 | 457 | 10.22 | 3.6 | 0.023 | 37.6 | 29.0 | 27.1 | 7.5 | 0.3 | 21.3 |
| Uptake and crosstake | 17,360 | 423 | 12.59 | 4.1 | 0.026 | 34.8 | 26.0 | 27.2 | 7.2 | 0.7 | 19.5 |
| Main dust chamber | 64,048 | 1,562 | 8.61 | 3.3 | 0.020 | 33.3 | 23.6 | 14.5 | 8.0 | 0.7 | 11.8 |
| Connecting flue[a] | 4,000 | 98 | 3.09 | 3.1 | 0.012 | 12.6 | 8.5 | 5.4 | 4.0 | 0.1 | 10.6 |
| Total | 191,169 | 4,662 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |

[a] Weight estimated. Average analysis is from sample taken in June, 1912, at different points from dust chamber to chimney.

A comparison of the dust recoveries from the old and new flue systems, with relation to the amount of original cupreous material treated, may be of interest, and is shown in Table IX.

TABLE IX.—*Comparison of Dust Recoveries from Old and New Flue Systems.*

| | Old Flue System | New Flue System |
|---|---|---|
| | Oct. 25, 1903, to June 12, 1909, 68 Months | June 12, 1909 to Jan. 1, 1913, 41 Months[a] |
| Tons new cupreous material treated at MacDougall furnaces | 1,213,890 | 579,545 |
| Tons new cupreous material treated at blast furnaces | 1,663,498 | 898,481 |
| Total tons new cupreous material treated at MacDougall and blast furnaces | 2,877,388 | 1,478,026 |
| Tons dust recovered from blast furnace dust chamber | 50,604 | 87,020 |
| Tons flue dust recovered from entire system | 104,239 | 191,169 |
| Dust recovered from blast furnace dust chamber in per cent. of total new cupreous material to blast furnaces | 3.04 | 9.69 |
| Total dust recovered from entire system in per cent. of total new cupreous material treated | 3.62 | 12.93 |

[a] Plant not operating during December, 1909, and part of January, 1910.

The experimental brick-flue tests indicated that 3.7 per cent. of the new cupreous material charged to the MacDougall roasters was being lost through the old chimney and flue system. Also, that 2.57 per cent. of the new cupreous material charged to blast furnaces was lost in the same way. Applying these percentages to the respective tonnages of new cupreous material treated by the two departments, when operating with the old flue system, and adding to these results the tonnage of dust recovered, we find that 6.66 per cent. of the new cupreous material charged to the furnaces was carried into the flues as dust. Assuming that at present a practically complete recovery of all dust is made, it will be seen that with the new conditions the amount of dust produced is about twice as much as under the old system.

There are several reasons why this is so, two of which are well defined: First, as will be shown later, there is a greatly increased volume of gas passing through the new system per ton of material treated. This, of course, is the result of a much stronger draft, together with improvements in flue construction which have largely eliminated draft hindrances. In this connection it may be of interest to note that the top of the new chimney is 751 ft. above the blast-furnace charging floor, while the top of the old chimney was only 419.5 ft. above the same floor. Second, since the new system has been in operation, a considerable tonnage of fine concentrate has been charged to the blast furnaces, a practice which was not in vogue under the old system.

The curve on Plate XXXIII may be designated as a cumulative plot of the dust collected and drawn from the wire-baffled dust chamber.

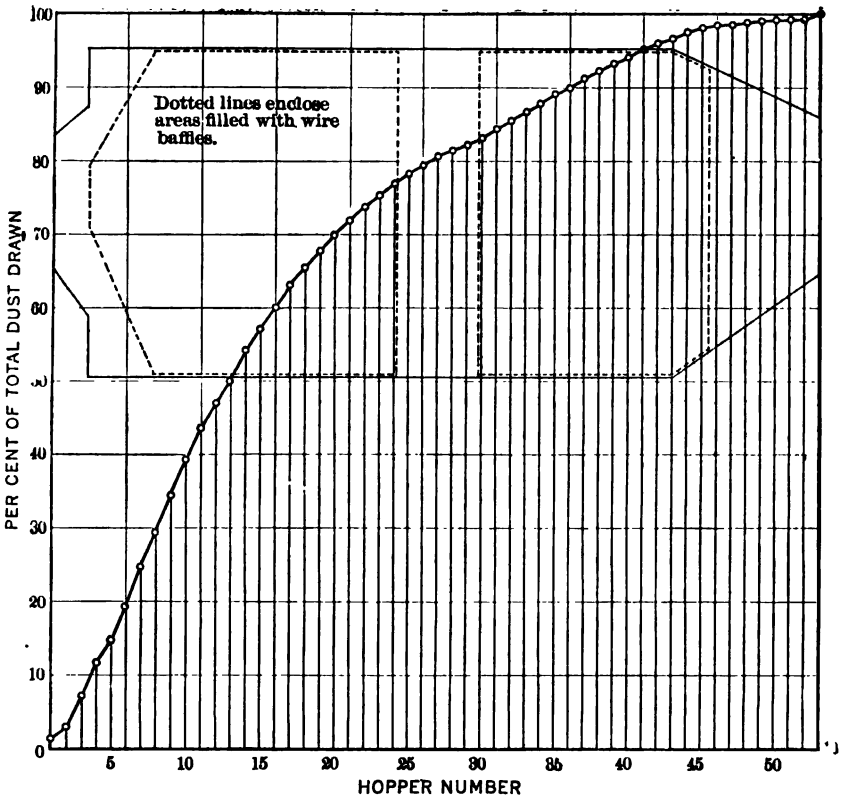



PLATE XXXIII.—CUMULATIVE PLOT OF DUST DRAWN FROM DUST CHAMBER.

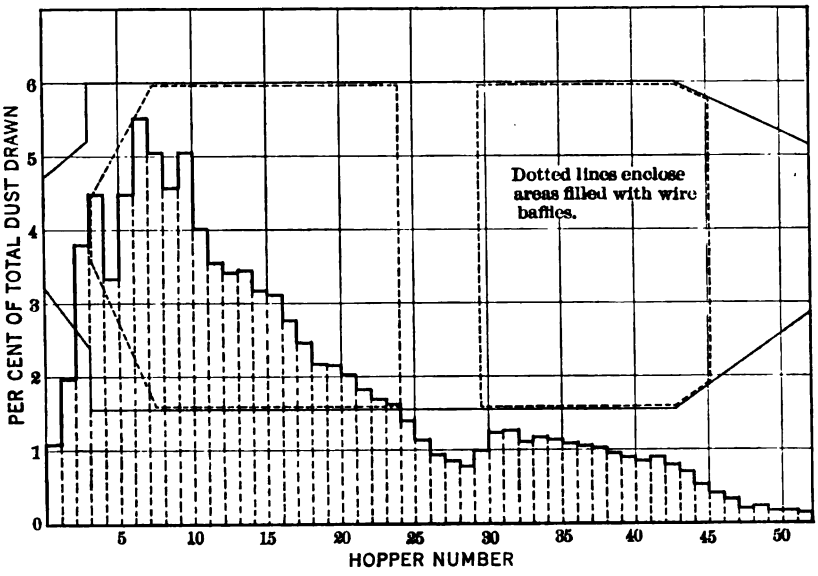



PLATE XXXIV.—PLOT SHOWING RELATIVE DEPOSITION OF DUST THROUGH DUST CHAMBER.

That is, it shows the dust collected from the inlet end up to any given point, in per cent. of the total dust collected in the chamber. The superimposed outline sketch of the dust chamber, showing the wire-baffled areas within the dotted lines, makes clear the cause of changes in the slope of the curve at various points.

The curve or plot on Plate XXXIV shows the relative actual total amounts of dust drawn from the different cross rows of hoppers in the dust chamber. The outline sketch of the dust chamber is again superimposed on this plate to show the location of the maximum dust settlement.

From these two plates it will be noted that about 77.5 per cent. of the total dust deposited is obtained before reaching the upper end of the first section of wires. On account of the low percentage of copper in the dust from the upper part of the chamber, it is evident that the proportion of values collected in the first section of wires is a greater percentage of the total than is indicated by the amount of dust.

Table X shows how the analysis of the dust changes from the inlet end of the dust chamber up to the base of the chimney.

TABLE X.—*Analyses of Dust from Inlet End to Base of Chimney.*

*Dust Chamber*

| | Cu | Insol. | $SiO_2$ | FeO | $Al_2O_3$ | CaO | [illegible] |
|---|---|---|---|---|---|---|---|
| Hopper Nos. | | | | | | | |
| 1–7 | 10.05 | 32.8 | 23.7 | 17.4 | 7.8 | 0.3 | [illegible] |
| 8–12 | 9.17 | 32.1 | 22.6 | 15.2 | 7.7 | 0.4 | [illegible] |
| 13–17 | 8.72 | 31.6 | 23.35 | 17.2 | 7.5 | 1.1 | [illegible] |
| 18–24 | 7.43 | 28.15 | 20.6 | 13.45 | 7.2 | 1.1 | [illegible] |
| 25–29 | 6.83 | 27.1 | 19.3 | 12.0 | 7.4 | 1.2 | [illegible] |
| 30–38 | 5.83 | 22.7 | 15.7 | 9.7 | 6.5 | 1.1 | [illegible] |
| 39–45 | 5.17 | 21.0 | 14.4 | 8.7 | 6.7 | 1.0 | [illegible] |
| 46–52 | 3.73 | 16.3 | 10.85 | 6.5 | 4.6 | 1.1 | [illegible] |

*Connecting Flue.*

| | | | | | | | |
|---|---|---|---|---|---|---|---|
| Door No. | | | | | | | |
| 7–E | 4.13 | 17.4 | 11.5 | 7.2 | 5.6 | 0.1 | [illegible] |
| 9–E | 3.13 | 12.9 | 8.8 | 5.4 | 3.6 | 0.1 | [illegible] |
| 10–E | 3.64 | 15.0 | 10.5 | 6.4 | 4.1 | 0.1 | [illegible] |
| 12–E | 2.36 | 9.2 | 6.1 | 4.1 | 3.8 | 0.1 | [illegible] |
| Base of chimney | 2.21 | 8.4 | 5.4 | 3.8 | 2.8 | 0.1 | [illegible] |

Plate XXXV shows the relative deposition of dust across the chamber. In making this plot the total weight of dust from hoppers 4 to 43 inclusive was taken, since all other hoppers are in converging portions of the chamber. The view is taken facing the chimney.

shows that a fairly uniform rate of settlement is effected over the full width of the chamber. The marked irregularity in adjacent rows of hoppers in this as well as in Plate XXXIV is due chiefly to the manner in which the dust is drawn. The arrangement of tracks under the hoppers is such as would lead to a result of this kind unless strict supervision is exercised over the operators.

Plate XXXVI shows the distribution of dust collected in the blast-furnace flue, with relation to the furnace locations and furnace days in

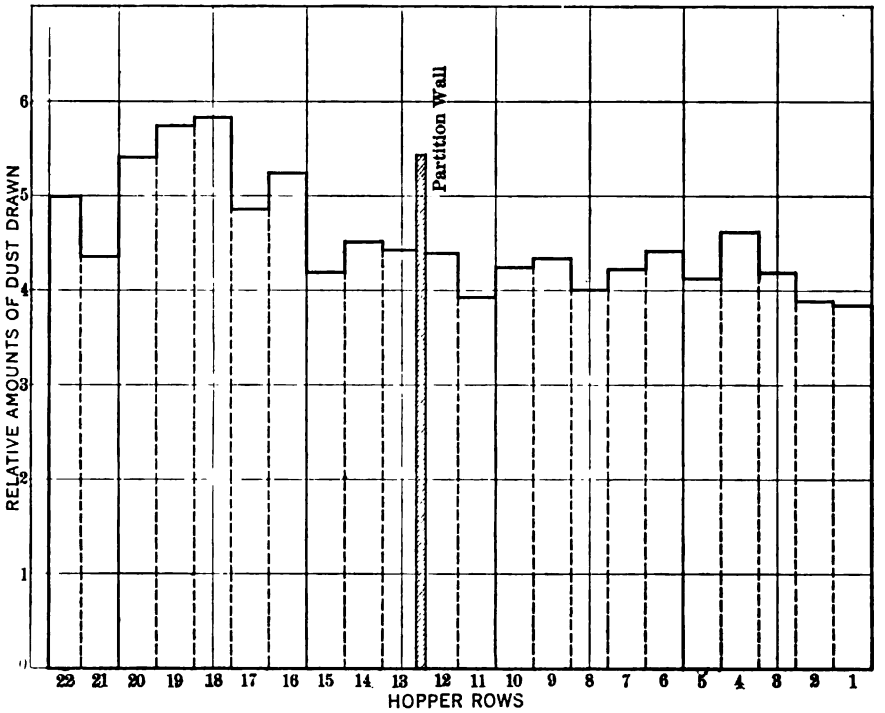



PLATE XXXV.—RELATIVE DEPOSITION OF DUST ACROSS DUST CHAMBER.

operation, from June 12, 1909, to Jan. 1, 1913. The increase in the amount of dust from hopper No. 1 is doubtless due to the flow of dust from the uptake when it is partially filled up from dumping cross-take dust into it.

Recent observations made by drawing out several wires from the central portion of both the upper and lower groups of wires show that in the first or lower group there is a light accumulation of dust on each wire which readily falls off when gently rapped. A cross-section through this accumulated dust is wedge shaped, the point of the wedge facing the current of gas being almost as thin as a knife edge. The depth of the

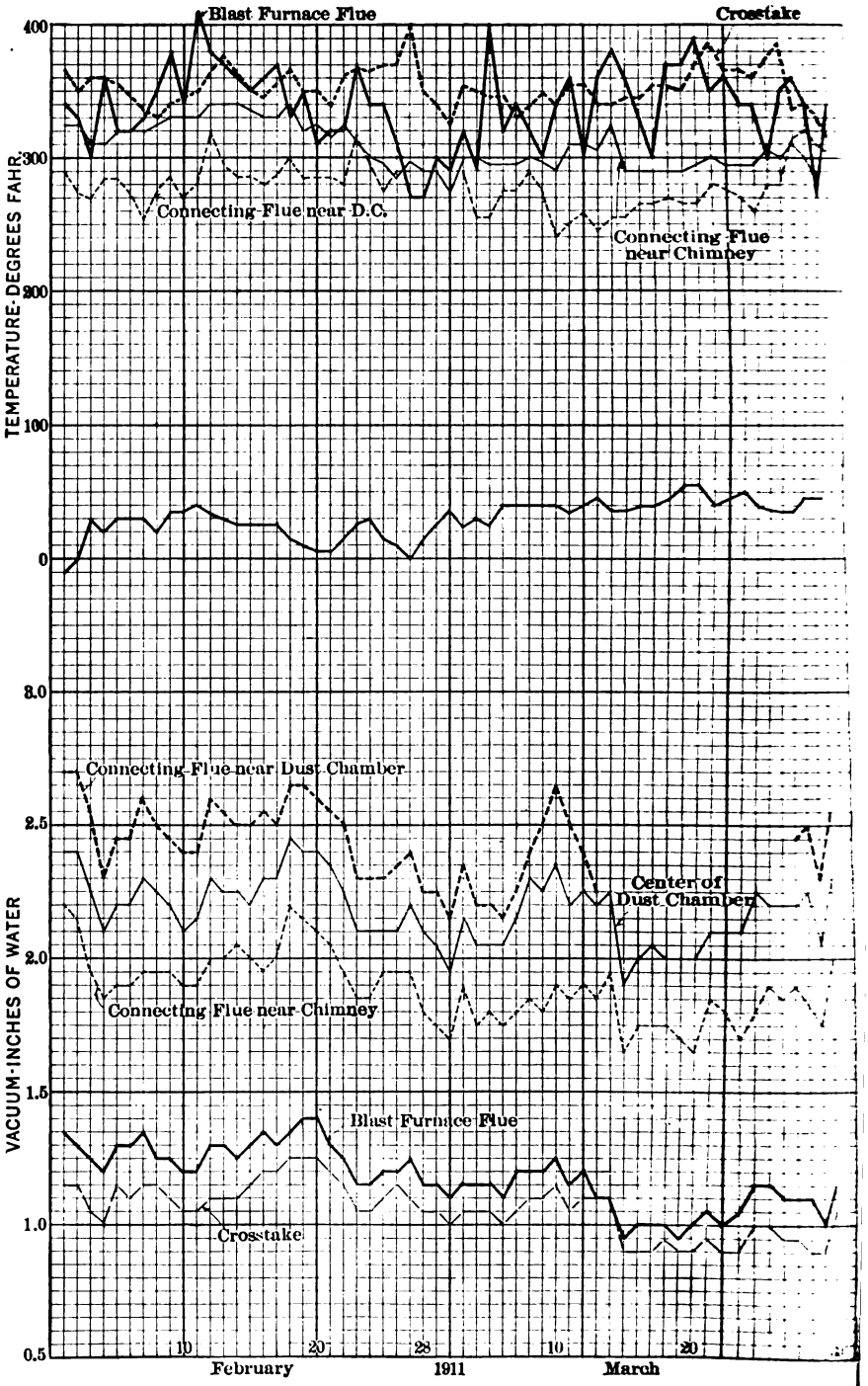



PLATE XXXVII.—DRAFT AND TEMPERATURE CURVES.

MacDougalls, three 12-ft. converters and three reverberatories in operation. Gases from two of the reverberatories were being bypassed around the main dust chamber. In taking these readings, Pitot-tube elements were used, the impact- and static-pressure ends being read independently.

Using these observations, the total draft energy on the MacDougall furnaces is calculated, and also the loss in draft through the various sections of the flue system.

The lower hearth of the MacDougall furnaces is at an elevation of 3,313, while the top of chimney is at an elevation of 4,054 ft.

| | Difference in Elevation Ft. | Average Temperature F.° | Draft Loss In. Water |
|---|---|---|---|
| Lower hearth of MacDougalls to cross-take..... | 100 | 460 | 1.40 |
| Cross-take through wire baffles to connecting flue. | 0 | 329 | 1.26 |
| Connecting flue near dust chamber to near chimney | 157 | 311 | 0.25 |
| Connecting flue near chimney to top of chimney.. | 484 | 300 | 0.06 |
| Totals..... | 741 | ..... | 2.97 |

It should be explained that the loss through the wire-baffled dust chamber shown in the above tabulation is higher than normal, because of the fact that the upper section of wires had not been shaken for about four months and also that there was an accumulation of several feet of dust in the lower group of wires.

A comparison of draft readings at similar points on the old and new systems may be of interest. Readings, of course, vary rather widely, depending upon furnace and other conditions, but in general are about as follows:

| | Draft Readings—In. Water | | | |
|---|---|---|---|---|
| | Blast Furnace Flue | MacDougall Flue | Reverb. Flue Near Furnaces | On Flue Near Chimney |
| Old system..... | 0.50 | 0.25 | 0.90 | 0.60 |
| New system..... | 1.15 | 0.90 | 1.50 | 1.85 |

## XI. Velocities and Volumes.

During March and April, 1911, tests were made to determine the velocity and volume of gases passing through the various parts of the new flue system. From these tests also, the amount of gas per furnace and per ton of cupreous material treated has been calculated.

Pitot tubes were used in making velocity determinations, nine impact

points being inserted into the flue, representing as many equal rectangular areas. Bristol pyrometers were used for obtaining temperatures, and other instruments and methods were the same as have been described earlier in this paper.

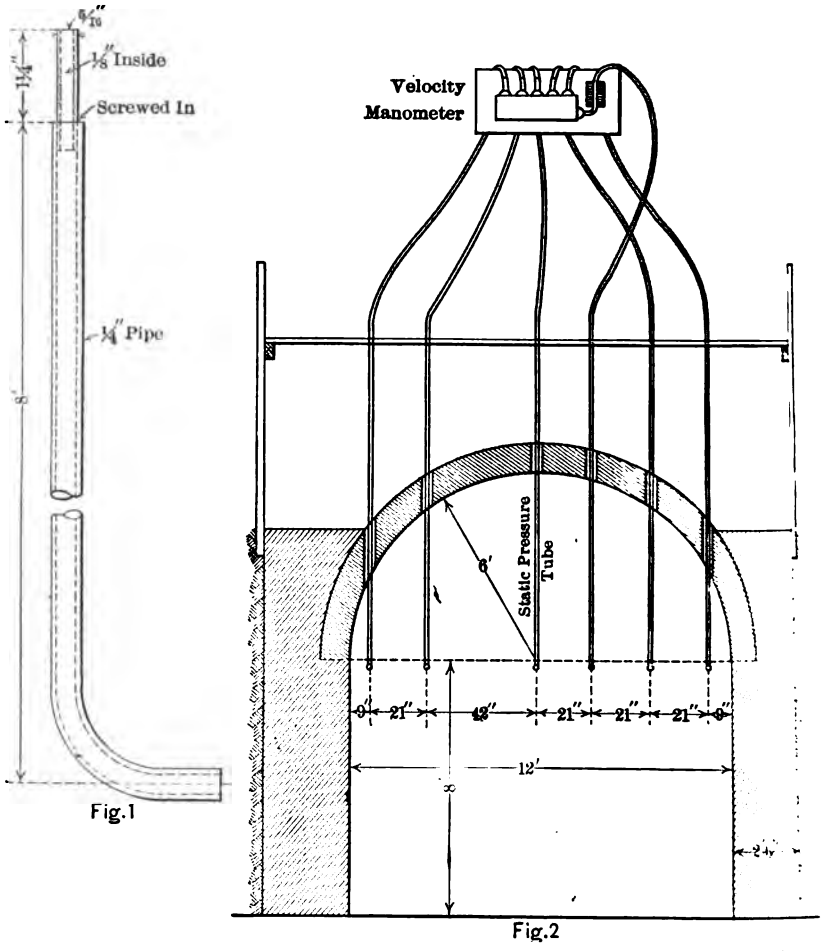



PLATE XXXVIII.—CONSTRUCTION AND ARRANGEMENT OF PITOT TUBES IN REVERBERATORY FLUE.

During this test the reverberatory gases were bypassed around the dust chamber through the old No. 3 flue, in which velocity readings were taken with five Pitot points as indicated on Plate XXXVIII.

The furnaces referred to in the tables below are of the following sizes:

Blast furnaces, 56 by 180 in. at tuyeres.
MacDougall furnaces, 14 ft. 6 in. inside of lining; six hearths.
Reverberatories, gas fired, 42 ft. 4 in. by 15 ft. 9 in.
Converters, 12 ft. upright.

TABLE XII.—*Velocities, Volumes, Average Temperatures and Weights of Gases.*

| Location See Plate XIII a | DATE, 1911 | No. and Kind of Furnaces | Average Temperature F.° | Clear Area of Flue Sq. Ft. | Vel. in Ft. per Sec. | Vol. at Observed Temperature Cu. Ft. per Min. | Pounds Gas per Min. | Pounds Gas per Furnace per Min. |
|---|---|---|---|---|---|---|---|---|
| A | Apr. 6–8..... | 4 B. F. | 345 | 401.6 | 17.26 | 415,960 | 18,510 | 4,630 |
| B | Apr. 6–8..... | 10 MacD. | 352 | 169.0 | 17.50 | 253,500 | 10,980 | 1,100 |
| C | Apr. 6–8..... | 5 Conv. | 303 | 78.5 | 50.37 | 213,630 | 9,840 | 1,970 |
| G | Mch. 21–24.... | 2 Rev. | 496 | 152.5 | 43.34 | 396,700 | 14,530 | 7,270 |
| $D_1$ | Apr. 6–8.... | 4 B. F.<br>10 MacD.<br>5 Conv. | 331 | 636 | 21.80 | 859,300 | 38,190 | ..... |
| $E_1$ | Mch. 21–24.. | 4 B. F.<br>6 MacD.<br>5 Conv. | 286 | 977 | 15.51 | 901,400 | 42,280 | ..... |
| F | Mch. 21–24.. | 4 B. F.<br>6 MacD.<br>5 Conv.<br>2 Rev. | 311 | 977 | 21.03 | 1,234,900 | 56,110 | ..... |
| $E_2$ | Apr. 10..... | 4 B. F.<br>10 MacD.<br>5 Conv. | 290 | 977 | 15.54 | 910,950 | 42,270 | ..... |
| $D_2$ | Apr. 10..... | do. | 322 | 636 | 22.74 | 867,760 | 38,700 | ..... |

[a] Fig. 3 of paper by Corwin and Rodgers on Increasing the Efficiency of MacDougall Roasters, July *Bulletin*.

Tests A, B, and C on the individual department flues show a total of 39,330 lb. of gas per minute. The gases from these flues unite in the cross-take on which a simultaneous test was made showing 38,190 lb. per min., thus checking within about 3 per cent.

Tests $D_1$ and $E_1$ taken at the inlet and outlet end respectively of the main dust chamber cannot be compared directly, since they were made on different days, and a different number of furnaces were in operation. Tests $D_2$ and $E_2$ may be compared since they were taken on the same day and under the same conditions.

| | Lb. |
|---|---|
| $D_2$—Gas entering main dust chamber through cross-take flue. | 38,700 |
| $E_2$—Gas leaving main dust chamber through connecting flue. | 42,270 |
| Leakage of air into main dust chamber..... | 3,570 |
| Leakage in per cent. of gas entering main dust chamber..... | 9.2 per cent. |

The temperature of the atmosphere on the day of the test was 54° F. Calculating the temperature which would result from mixing 3,750 lb. of air at 54° F. with 38,700 lb. of gas (considered as air) at 322° F., we obtain a temperature of 299°. Since the air leaving the dust chamber was at a temperature 290°, the loss due to radiation was 9° F. Summarizing we have:

| | |
|---|---|
| Temperature of gases entering chamber through cross-take flue. | 322° F. |
| Temperature of gases leaving chamber through connecting flue. | 290° F. |
| Temperature of outside air..... | 54° F. |
| Drop in temperature of gases in passing through chamber..... | 32° F. |
| Drop in temperature of gases caused by leakage of cold air..... | 23° F. |
| Drop in temperature of gases caused by radiation..... | 9° F. |
| Per cent. of total drop due to leakage of cold air..... | 71.9 |
| Per cent. of total drop due to radiation..... | 28.1 |

A further check on the accuracy of the tests may be made by comparing the sum of tests $E_1$ and G with test F. Test $E_1$, which includes blast furnace, MacDougall and converter gases plus test G, which covers reverberatory gases only, show a total of 56,810 lb. per minute. Test F on the connecting flue near the chimney, where all furnace gases were combined, shows a total of 56,110 lb. gas per minute, being a difference of about 1.25 per cent.

Based on the amounts of gas for the various department flues, as presented in Table XII, Table XIII is presented to show the volume and weight of gas for one furnace of each kind when operating on the new flue system.

TABLE XIII.—*New Flue System.*

*Amount of Gas per Furnace and per Ton of Charge.*

| Kind of Furnace | Observed Temperature Corresponding to Given Volume F.° | Rate per Min. | | Rate per 24 Hr. | | Average Tons Charged | Per Ton of Charge | |
|---|---|---|---|---|---|---|---|---|
| | | Cu. Ft. | Lb. | Cu. Ft. | Lb. | | Cu. Ft. | Lb. |
| Blast..... | 345 | 103,990 | 4,630 | 149,745,600 | 6,667,200 | [a]391.6 | 384,900 | 17.00 |
| MacDougall... | 352 | 25,350 | 1,100 | 36,504,000 | 1,584,000 | [b]70.7 | 516,300 | 22.40 |
| Converter..... | 303 | 42,730 | 1,970 | 61,531,200 | 2,836,800 | 32.0 | 1,922,900 | 88.60 |
| Reverberatory.. | 496 | 198,350 | 7,270 | 285,624,000 | 10,468,800 | 188.5 | 1,515,200 | 55.50 |

[a] Includes flux but does not include fuel.
[b] Tons copper produced per converter day.

Similar values for the old flue system are shown in Table XIV.

TABLE XIV.—*Old Flue System. Amount of Gas per Furnace and Per Ton of Charge.*

| Kind of Furnace | Observed Temperature Corresponding to Given Volume F.° | Rate per Min. | | Rate per 24 Hr. | | Average Tons Charged | Per Ton of Charge | |
|---|---|---|---|---|---|---|---|---|
| | | Cu. Ft. | Lb. | Cu. Ft. | Lb. | | Cu. Ft. | Lb. |
| Blast..... | 600 | 89,400 | 3,030 | 128,736,000 | 4,363,200 | *a*417.0 | 308,720 | 10,460 |
| MacDougall... | 562 | 9,140 | 320 | 13,161,600 | 460,800 | *b*38.2 | 344,540 | 12,060 |
| Converter..... | 338 | 24,900 | 1,120 | 35,856,000 | 1,612,800 | 30.9 | 1,160,390 | 52,190 |
| Reverberatory.. | 1,000 | 78,680 | 1,930 | 113,299,200 | 2,779,200 | 207.3 | 546,550 | 13,410 |

*a* Includes flux but does not include fuel.
*b* Tons copper produced per converter-day.

A parallel comparison of the old and new systems is shown in Table XV.

TABLE XV. *Comparison of Old and New Systems.*

| Kind of Furnace | Observed Temp. of Gas F.° | | Pounds per Minute | | | Pounds per Ton of Charge | | |
|---|---|---|---|---|---|---|---|---|
| | Old | New | Old | New | Excess of New Per Cent. | Old | New | Excess of New Per Cent. |
| Blast..... | 600 | 345 | 3,030 | 4,630 | 52.8 | 10,460 | 17,000 | 62.5 |
| MacDougall..... | 562 | 352 | 320 | 1,100 | 243.0 | 12,060 | 22,400 | 85.7 |
| Converter..... | 338 | 303 | 1,120 | 1,970 | 75.9 | 52,190 | 88,600 | 69.8 |
| Reverberatory... | 1,000 | 496 | 1,930 | 7,270 | 276.0 | 13,410 | 55,500 | 314.0 |

The points of determination of the gas volumes for the two systems were as follows:

Blast Furnaces:

Old system—On old flues above skyline, four furnaces in operation.

New system—On blast-furnace flue between No. 1 furnace and uptake, four furnaces in operation.

MacDougall Furnaces:

Old system—Directly over No. 2 furnace, test on one furnace only.

New system—On MacDougall flue near furnaces, gas measured from seven furnaces.

Converters:

Old and new systems measured at same point on steel flue over roof.

Reverberatory Furnaces:

Old system—On uptake east of old stone dust chamber, two furnaces in operation.

New system—On old No. 3 flue south of skyline tracks, two furnaces in operation.

The large excess of gas from the MacDougall furnaces under present conditions is due not only to a better draft, but to a very much improved arrangement of flues, which greatly reduces the draft hindrances, as already pointed out.

The cause of the large increase in the weight of the waste gas from the reverberatory furnaces is not so easily explained, but, in addition to better draft conditions, it is believed that the quantities given include a big leakage of cold air which came in through the old stone chamber. This belief is strengthened by the results of some temperature readings, which were taken on the reverberatory waste gases at a time when the gases from three furnaces were passing through the old stone dust chamber. These readings gave an average temperature of 1,057° F. just before entering the stone chamber and a temperature of 605° F. after leaving the chamber.

The measurement on volume of reverberatory gas on the old system was made on the uptake from the underground flue before it entered the stone dust chamber, and for that reason includes no leakage of cold air except what comes in at the furnace, and consequently is less in volume and of a higher temperature.

Based on the quantities of gas given in the tables above, the approximate velocities through the various parts of the new system as it is being operated at present are as follows:

| | *Ft. per Sec* |
|---|---|
| Blast-furnace flue | 17.3 |
| MacDougall furnace flue | 17.5 |
| Cross-take flue | 21.8 |
| Main dust chamber | 4.0 |
| Connecting flue near chimney | 21.0 |
| In chimney | 10.0 |

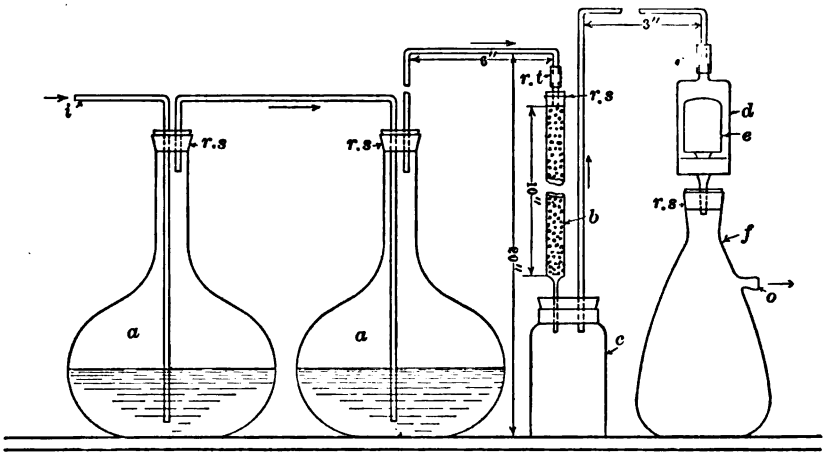
The velocity of 4 ft. per second in the main dust chamber is based on the full area, clear of dust and wires. The actual velocity through the wires varies with the amount of dust in the flue but will probably average from 20 per cent. to 25 per cent. higher.

## XII. Tests on the New Flue System.

In September, 1911, a series of tests was started to determine the volume and solid contents of gases passing through the new flue system. These tests were carried out under the supervision of G. N. Libby, to whom we are indebted for the following description of apparatus and methods employed in taking the samples:

"The apparatus used in the determination of the suspended and condensable material in the flue gases is shown in Plate XXXIX. Samples were taken in each case simultaneously at opposite points in the flue 8 ft. from the east and west sides, by means of a 10 ft. by 0.5 in. glass tube inserted horizontally through openings in the flue wall and supported by angle irons fastened in these openings. The inner end of the glass tube was bent at right angles and a platinum tip 5 mm. internal diameter fused in the same. This 5-mm. opening was placed in the flue facing the

flow of gases. Connected to the outer end of the tube in series were (1) two 1-liter wash bottles each containing about 250 c.c. of distilled water, (2) a glass tube 10 in. by 0.5 in. filled with glass beads and (3) finally, a 'dust collector' with a paper filter. This latter was kept warm by an electric light bulb placed near it, in order to keep condensing water from clogging it up. Only the slight heat necessary to keep the




*a, a.* 250 = cc. Florence flasks, about half full of distilled water.
*b.* Filter tube with glass beads.
*c.* 8-oz. wide-mouthed bottle.
*d.* Glass shell for filter paper cartridge. (Shell was inclosed in pasteboard box with one 60-watt lamp to avoid condensation.)
*e.* Extraction cartridge (packed in cotton).
*f.* Filter flask.
*i.* Inlet to system.
*o.* Outlet of system.
*r.s.* Rubber stoppers.
*r.t.* Rubber tubing.
Arrows indicate direction of flow of gases through filter system.
All connections made air tight.
Glass tubing is ⅛ in. diameter.

PLATE XXXIX.—GAS SAMPLING APPARATUS.

gases above the dew point was used. It was regulated by insulating the lamp with varying amounts of glass wool.

"Aspiration was effected through this system by the displacement of water in two large ammonia drums of about 10 cu. ft. capacity. One was being filled while the other was emptying and aspirating, thus giving a continuous suction. Glass gauge tubes were fitted in the side of each drum and the drum calibrated by placing it on a tested pair of scales, filling it with water to the zero mark on the gauge and allowing the water to run from the drum 0.10 cu. ft. at a time, by weight, and marking the changed height of water on the gauge glass.

"Endeavor was made to keep the rate of aspiration such that the gases would be drawn through the 5-mm. opening at the same speed as the gases passing. This, of course, could only be approximated, but was close enough for all practical purposes.

"The volume of the sample was determined by reading the number of cubic feet of water displaced and correcting to standard conditions. A thermometer placed in the drum gave the temperature, and a manometer just after the filter gave the pressure. These were read after the aspiration of each drum.

"A test of the gas after passing through the apparatus was made, and it was found that only a trace of $As_2O_3$ remained.

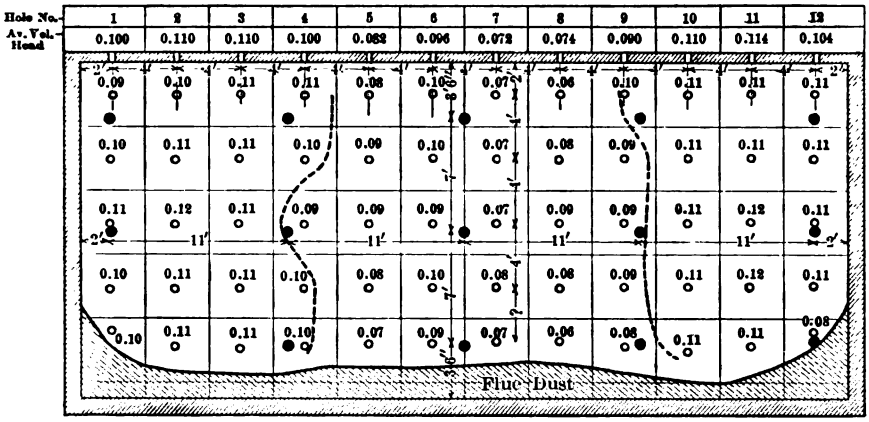
"After the sampling was finished the apparatus was disconnected from the aspirators, the flue tube washed off on the outside, and the whole taken to the laboratory where the flue tube, the wash bottles, the bead tube, and beads were washed thoroughly and the matter obtained, together with the water-soluble content of the filter, placed in a large evaporating dish and evaporated slowly to dryness on the steam bath at about 90° C. This matter was then all carefully scraped up, placed in a small weighing bottle, dried at 95–97° C. for 24 hr. and weighed. This weight, plus the difference in weight of the filter before and after aspiration dried at 95–97° C., was called total collected matter. On account of the $H_2SO_4$ content of the sample it was, of course, impossible to drive off all moisture at the temperature used in drying, and this probably accounts for most of the difference which exists between the total collected matter weighed and that found by chemical analysis when calculated with its maximum possible oxygen content."

Samples were taken at two places on the connecting flue. The first set was obtained at the top of the first 30 per cent. grade north of the main dust chamber. The reverberatory gases enter the connecting flue through a bypass further up and were therefore not included in these samples.

The second set of samples taken from Oct. 5 to 10, 1911, was obtained from each of the two branches just before entering the chimney. The results of this latter set only will be presented, for the reason that all smelter gases were combined before passing the point where the samples were taken.

Observations for velocity and volume of gases were made simultaneously with taking the samples. In addition to the regular velocity readings taken during the time covered by this series of tests, several determinations were made on the average velocity of the gases over the whole flue area, by taking observations at a large number of points in the cross-section. A chart showing the results of one of these determinations is presented in Plate XL. This chart also shows the depth of dust in the

flue, which was determined by sounding from the roof, and the location of the Pitot points used for regular velocity determinations. It will be noted that the velocity in the central portion of the flue is lower than at the sides. This may be due to the proximity of the dividing point where




1. Av. V. H. for West 16 ft. = 0.103.
   Av. V. H. for Middle 16 ft. = 0.078.
   Av. V. H. for East 16 ft. = 0.107.
   Av. V. H. for Entire Section = 0.096.
2. Av. V. H. for West 16 ft. = 0.105.
   Av. V. H. for Middle 16 ft. = 0.084.
   Av. V. H. for East 16 ft. = 0.105.
   Av. V. H. for Entire Section = 0.098.
3. Av. V. H. for West 16 ft. = 0.108.
   Av. V. H. for Middle 16 ft. = 0.084.
   Av. V. H. for East 16 ft. = 0.108.
   Av. V. H. for Entire Section = 0.100.
4. Av. V. H. for West 16 ft. = 0.105.
   Av. V. H. for Middle 16 ft. = 0.084.
   Av. V. H. for East 16 ft. = 0.108.
   Av. V. H. for Entire Section = 0.099.
   'v. V. H. for West 16 ft. = 0.105.
   V. H. for Middle 16 ft. = 0.073.
   V. H. for East 16 ft. = 0.095.
   H. for Entire Section = 0.091.
   . for Entire Flue = 0.097.
   .. H. for West 16 ft. of Flue = 0.105.
   .v. V. H. for Middle 16 ft. of Flue = 0.081.
   Av. V. H. for East 16 ft. of Flue = 0.105.
   Broken Line Connects Approximate Points of Average Velocity.
   All Velocity Head Readings Given in Inches $H_2O$.
   Indicates Points Where Regular Readings Were Taken.

PLATE XL.—CROSS-SECTION OF FLUE OF STATION 42.5 FT. SOUTH OF "Y," SHOWING VELOCITY HEAD READINGS.

the flue branches into the two chimney openings. It was found that the average velocity as determined by this special test checked closely with the results of the regular velocity observations.

As explained by Mr. Libby, the samples were aspirated from the flue through glass tubes inserted through the side walls, and extending approximately to the central point of the flue. The total amount of the dust sample with this type of apparatus is necessarily small, but the results of the various tests are very uniform.

Samples were taken at two points in each branch of the flue, the upper point being 9 ft. below the roof and the lower point 8 ft. 6 in. above the floor, the two branches of the flue being designated as East and West respectively. Three samples were taken at each point, making 12 in all.

During these tests the following furnaces were in operation: two blast furnaces; two reverberatory furnaces; eight MacDougall furnaces; and 2.6 converters (average). The average results of the three samples taken at any given point, together with the general average for the whole series are shown in Table XVI.

TABLE XVI.

| Sample Point | Main Flue Gases | | Average Cu. Ft. of Sample Stand. Cond. | Average Solids Collected Grams | Calculated Solid Contents of All Flue Gases per 24 Hours | | | |
|---|---|---|---|---|---|---|---|---|
| | Velocity Ft. per Second | Cu. Ft. per 24 Hours Stand. Cond. | | | Solid Matter Lb. | Cu Lb. | *a*Ag Oz. | *a*Au Oz. |
| Upper E | 18.63 | 901,816,800 | 180.4 | 4.566 | 51,220 | 311 | ..... | ..... |
| Lower E | 19.34 | 960,591,840 | 184.4 | 4.225 | 48,630 | 347 | ..... | ..... |
| Upper W | 18.63 | 901,816,800 | 170.8 | 4.672 | 54,900 | 358 | ..... | ..... |
| Lower W | 19.34 | 960,591,840 | 183.6 | 4.174 | 48,010 | 373 | ..... | ..... |
| Average | 18.98 | 931,204,320 | 179.8 | 4.409 | 50,690 | 347 | 28 51 | 0.202 |

*a* These values obtained by proportion, the sample not being of sufficient weight to make direct determinations.

The average amount of new cupreous material charged to furnaces per 24 hr. during this period contained 181,220 lb. of copper.

The loss of copper in escaping gases in per cent. of original copper charged to furnaces was therefore 0.19 per cent.

An average complete analysis of the suspended and condensable material contained in gases sampled is presented in Table XVII.

Tests on the old flue gases by the use of wire baffles in the experimental brick flue, indicated that 89.7 per cent. of all copper entering the flue with blast-furnace gases, and 96.2 per cent. of all copper entering with MacDougall gases, was recovered in the flue.

TABLE XVII.—*Complete Analysis of Suspended and Condensable Matter in Gases.*

| *Constituent.* | *Per Cent. of Total.* |
|---|---|
| Free $H_2SO_4$ | 22.23 |
| Silica | 2.30 |
| Copper | 0.70 |
| $Fe_2O_3$, $Al_2O_3$ | 7.08 |
| Sulphur | 6.67 |
| $Sb_2O_3$ | 1.47 |
| $Bi_2O_3$ | 0.81 |
| PbO | 0.49 |
| CaO | 0.18 |
| ZnO | 3.31 |
| O to combine with sulphur (calculated) | 10.23 |

Based on recoveries from the new wire-baffled dust chamber, and estimates on the amount of copper in the dust contained in the connecting flue, and the amount escaping from the chimney, we find that 94.2 per cent. of all copper entering the chamber is recovered from it, showing a remarkably close confirmation of test results.

Calculations based on the amounts of material in the flue system, as determined by weights drawn, and by estimates on dust and fume in the connecting flue and escaping gases, are presented in Table XVIII.

TABLE XVIII.—*Percentage Distribution of Material in Flue System.*

| | Weight | Copper | $SiO_2$ |
|---|---|---|---|
| Blast-furnace flue | 39.3 | 41.4 | 47.3 |
| MacDougall furnace flue | 8.5 | 11.3 | 11.0 |
| Cross-take flue | 7.8 | 12.9 | 8.8 |
| Main dust chamber | 28.9 | 32.5 | 30.7 |
| Connecting flue | 1.8 | 0.7 | 0.7 |
| Stack discharge | 13.7 | 1.2 | 1.5 |
| TOTALS | 100.0 | 100.0 | 100.0 |

*Production of Arsenic.*

Up to the present time no effort has been made to recover arsenic from flue dust, as the market for this material, until recently, has not justified the necessary outlay for its production, but we have just completed a plant which will handle about 20 tons of the dust per day.

For the present only the dust recovered from the upper end of the main dust chamber will be handled in this plant. Later it may be found desirable to increase the recovery of arsenic by the admission of cold air to the chamber, provision for which was made in the original design, as already explained.

## XIII. Conclusion.

Comparing the losses from the old flue system, as determined by tests with the experimental brick flue, with the losses on the new system, as shown by the tests which have just been described, and allowing for the relative amounts of copper produced at that time and at present, the following differences are noted:

| | Loss per 24 Hr. | | |
|---|---|---|---|
| | Copper Lb. | Silver Oz. | Gold Oz. |
| Old system | 3,775 | 105.94 | 0.712 |
| New system | 347 | 28.51 | 0.202 |
| Difference saved | 3,428 | 77.43 | 0.510 |

Assuming a recovery of 95 per cent. of these values, and taking a net value of 10c. per pound for the copper, 50c. per ounce for the silver and $20 per ounce for the gold, it is seen that the new flue system has effected a saving of $372.18 per day, or $130,263 per year.

In addition to this saving it is expected that a substantial revenue will be derived from the treatment of the more arsenical dust, in the recently completed arsenic plant.

In considering the large expenditure for the installation of the elaborate flue system just described, it should also be borne in mind that a reduction in losses of flue dust was not the only object in view. The low elevation at which the gases were discharged from the old chimney was a source of annoyance from smoke complaints. The overtaxed condition of the old flues and chimney prevented the operation of certain furnaces at full capacity, and for the same reason the smoke and fume around the furnaces materially interfered with the efficiency of labor. Furthermore, dust chambers were inadequate and their poor arrangement made the handling of flue dust expensive and troublesome.

All these defects have been eliminated, and the anticipated efficiency of the new flue system has been fully realized.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.
[SUBJECT TO REVISION].

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Butte meeting, Aug. 18 to 21, 1913, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1913, when Vol. XLVI. of the *Transactions* will go to press. Any discussion offered thereafter should preferably be in the form of a new paper for publication in Vol. XLVII. (with suitable cross references in both volumes).

# Notes on the Great Falls Electrolytic Plant.

BY WILLIS T. BURNS, GREAT FALLS, MONT.

(Butte Meeting, August, 1913.)

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## I. INTRODUCTION.

These notes are submitted, not as a discussion of the modern practice of electrolytic-copper refining, but as the record of a refinery that was among the pioneers in the field and that is to-day, and has been for 17 years, operating under conditions not to be found in any similar plant in the country. I refer to the current density, which is at least 60 per cent. higher than that regularly employed elsewhere, and to the fact that the anodes treated are the direct product of the converter rather than of the reverberatory furnace. The writer has no knowledge of any other refinery treating anodes of this character.

## II. HISTORY.

This refinery, now one of the oldest plants of its kind operating in this country, was built in 1892 by the Boston & Montana Consolidated Copper & Silver Mining Co. to refine the product of its Great Falls reduction works. The estimated capacity of the plant as designed was 65,000 lb. of cathodes per day at a current density of 16 amperes per square foot.

The plant produced its first cathodes in February, 1893, and has been in almost continuous operation since that date. The output of the plant for the year 1893 was 3,179 tons. As the smelter output increased demands for a larger refinery output were made and as no appropriation for enlarging the plant was forthcoming increased production could be obtained only by increasing the current density. As an abundance of water power was available and preliminary experiments had demonstrated that cathodes, of satisfactory quality, could be economically produced at more than double the density the plant was then operating at new and larger generators were installed in 1896 and the current density brought up to 40 amperes per square foot. Since that date the current density has been reduced by adding 76 refining tanks and by increasing the number of electrodes per tank.

In 1893, 3,179 tons of copper were refined at an average current density of 10 amperes per square foot; the production for the year 1912 was 31,596 tons and the average current density 34.1 amperes per square foot.

## III. General Description of Plant.

The refinery was built at the reduction works of the Boston & Montana Consolidated Copper & Silver Mining Co. situated at the Black Eagle falls of the Missouri river three miles from the city of Great Falls, Mont. Power for the operation of the reduction works and the electrolytic plant was obtained from water power developed at this point.

As originally designed and constructed the refinery consisted of the tank house, a brick building 171 ft. 6 in. by 108 ft. (Fig. 1) containing 288 electrolytic-refining tanks of the following inside dimensions: 9 ft. 7 in. long, 2 ft. 4 in. wide and 3 ft. 9 in. deep, built of Western pine. The sides and bottoms were of 3-in. lumber and the ends of 2-in. material. The tanks were tied at the ends with 3 by ½ in. iron straps held together by 1-in. rods as shown in Fig. 2. The tanks were lined with ⅛-in. chemical sheet lead weighing 8 lb. per square foot.

The tanks, which were supported in place by two 8 by 8 in. timbers which rested on stone piers 18 in. high, were insulated from the timbers by 4 by 4 by 1 in. glass insulators, eight under each tank. Had 5 or 6 ft. of head room, instead of 18 in., been provided under the tanks, subsequent operations would have been greatly facilitated.

The tanks were arranged in 18 double rows, eight tanks deep (Fig. 1) with an aisle between each double row. The tanks as originally built were all on the same level, making it necessary to establish an individual system of circulation of the electrolyte for each tank. This was accomplished by providing two circulating tanks for each section of 96 tanks

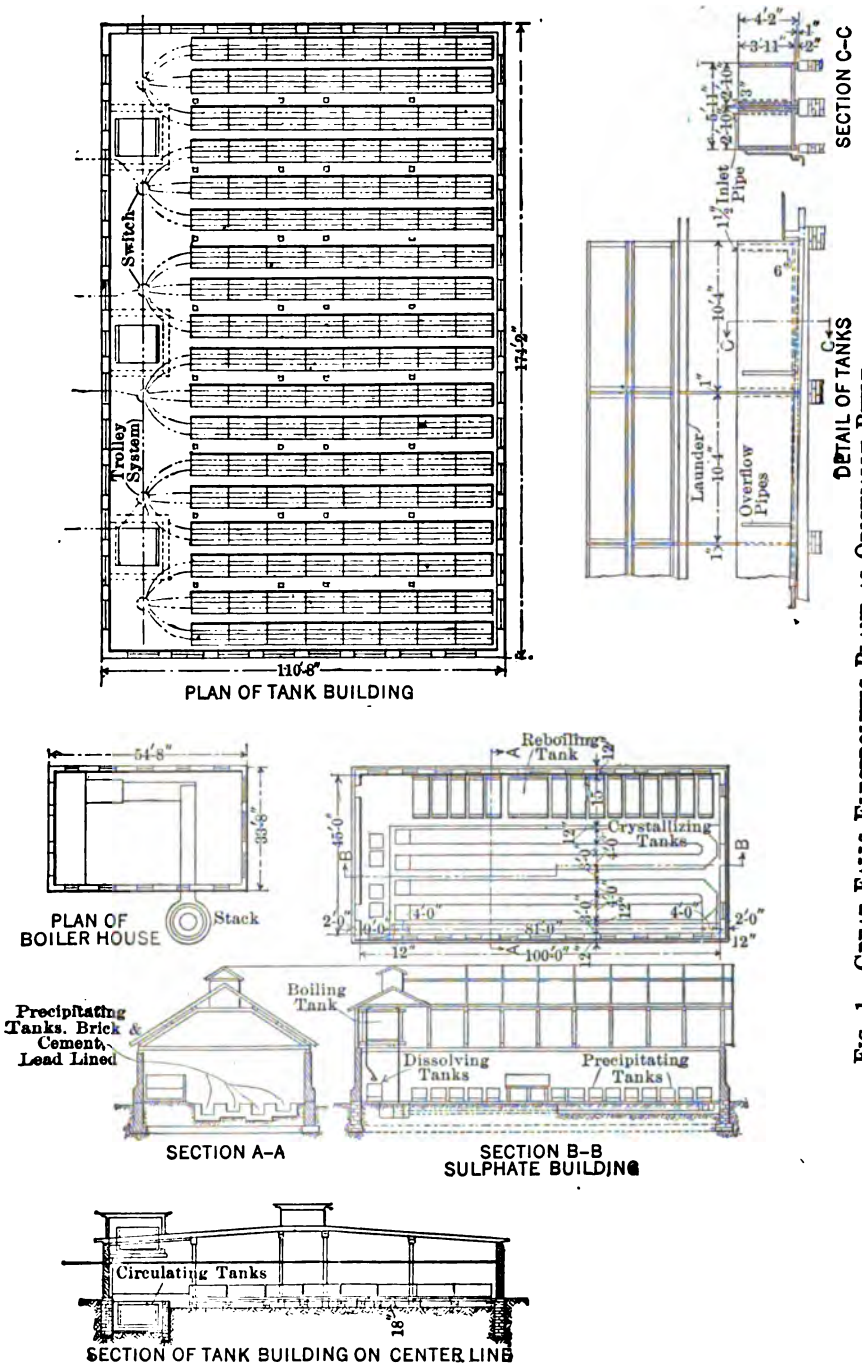



FIG. 1.—GREAT FALLS ELECTROLYTIC PLANT AS ORIGINALLY BUILT.

these tanks were 10 ft. square by 8 ft. deep, lined with 12-lb. chemical sheet lead. One of these tanks was placed above the level of the refining tanks and one below, as shown in sketch. From the overhead tank the electrolyte was delivered, through an overhead system of lead pipes, to each separate refining tank. The solution was discharged into the refining tank at one end, near the bottom of the tank, and was allowed to overflow through a 1-in. lead pipe entering the tank 3 in. from the top near the other end. This pipe discharged into an open launder leading to the lower circulating tank, from where it was raised to the overhead tanks by various devices to be described later. This system, which permitted only a periodical circulation of the solution, soon gave way to a more efficient system to be described further on. Heating the electrolyte was effected by passing live steam through 125 ft. of 1-in. 8-lb. lead pipe in each of the overhead circulating tanks.

A ¾ by 4 in. steel trolley track was suspended from the roof timbers over each row of eight tanks. The anodes and cathodes were removed in and out of the tanks by means of chain blocks suspended from two-wheel trolleys on this track.

The current was distributed by means of ¾ by 4 in. rolled-copper busbars attached to the sides of the tanks with wooden brackets. The tanks were all connected in series and the electrodes in multiple.

The original installation included a sulphate plant for the manufacture of bluestone and a boiler house containing one 100-h.p. locomotive-type boiler to furnish steam for heating and boiling solutions.

The sulphate plant, shown in Fig. 1, was well appointed for the manufacture of bluestone as described later. The capacity of this plant was 2,500,000 lb. of bluestone per year.

The above is a brief description of the plant as it originally existed. Fig. 3 is a sketch of the plant as it exists to-day. One division of 32 refining tanks has been added to the main tank room, 44 new electrolytic-refining tanks for the production of starting sheets have been housed in additions as have the 10 new insoluble-anode tanks shown in sketch. An addition has been built south of the tank room in which is installed the equipment for pumping and heating the electrolyte. An addition has been built on the east containing the scale house and a 7-ton traveling crane for handling the cathodes and scrap after weighing. The addition on the north side of the building contains store room, shops and office. Fig. 4 is a flow sheet of the electrolytic plant.

In the new building south of the bluestone building are the carpenter and lead-burning shops and the machinery used in trimming sheets. The four large round tanks situated further south are for the storage of solutions.

The original boiler house has been replaced by a brick building con-

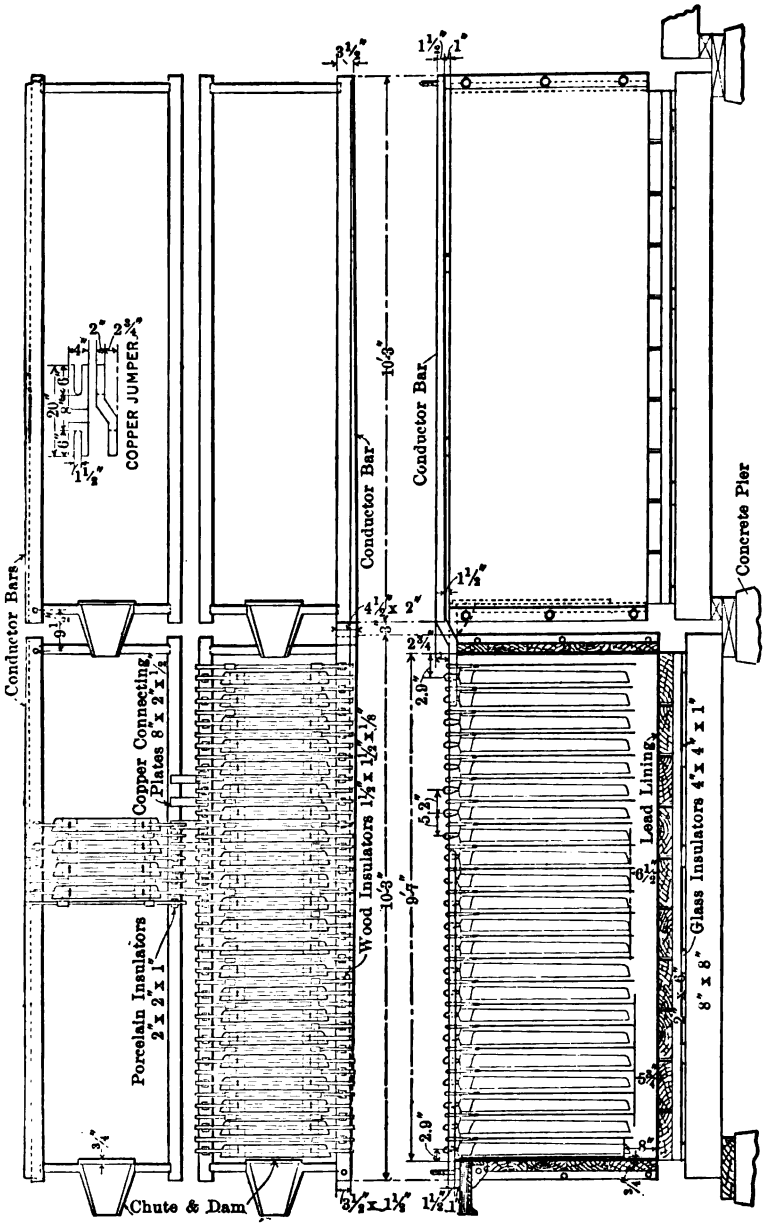



FIG. 2.—TANK DETAILS.

taining two 150-h.p. Heine and one 100-h.p. Scotch marine boiler from which steam for heating and boiling solutions is obtained.

As previously stated the refining tanks, as originally installed, were all on the same level and to obtain a circulation of the electrolyte it was necessary to deliver the electrolyte to each tank through a separate pipe line. This system was found unsatisfactory and the refining tanks were rearranged in a cascade of eight tanks with a drop of 2¾ in. between

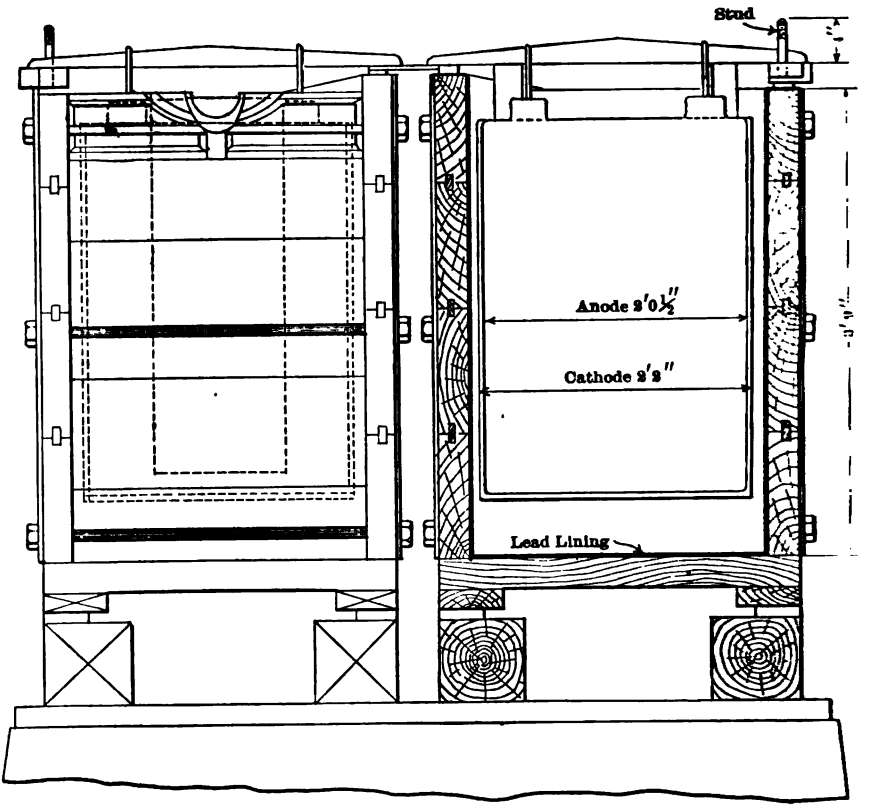



FIG. 2A.—TANK DETAILS.

tanks as shown in Fig. 3. The electrolyte is now delivered to the upper tank of the cascade at the surface and is drawn off from a point 8 in. from the bottom at the lower end of the tank by means of the dam shown and is delivered to the next tank through the cast-lead and antimony chute shown in Fig. 2. From the last or lower tank of the cascade the electrolyte is returned to the pump room through a 2-in. 7-lb. lead pipe attached to the sides of the tanks.

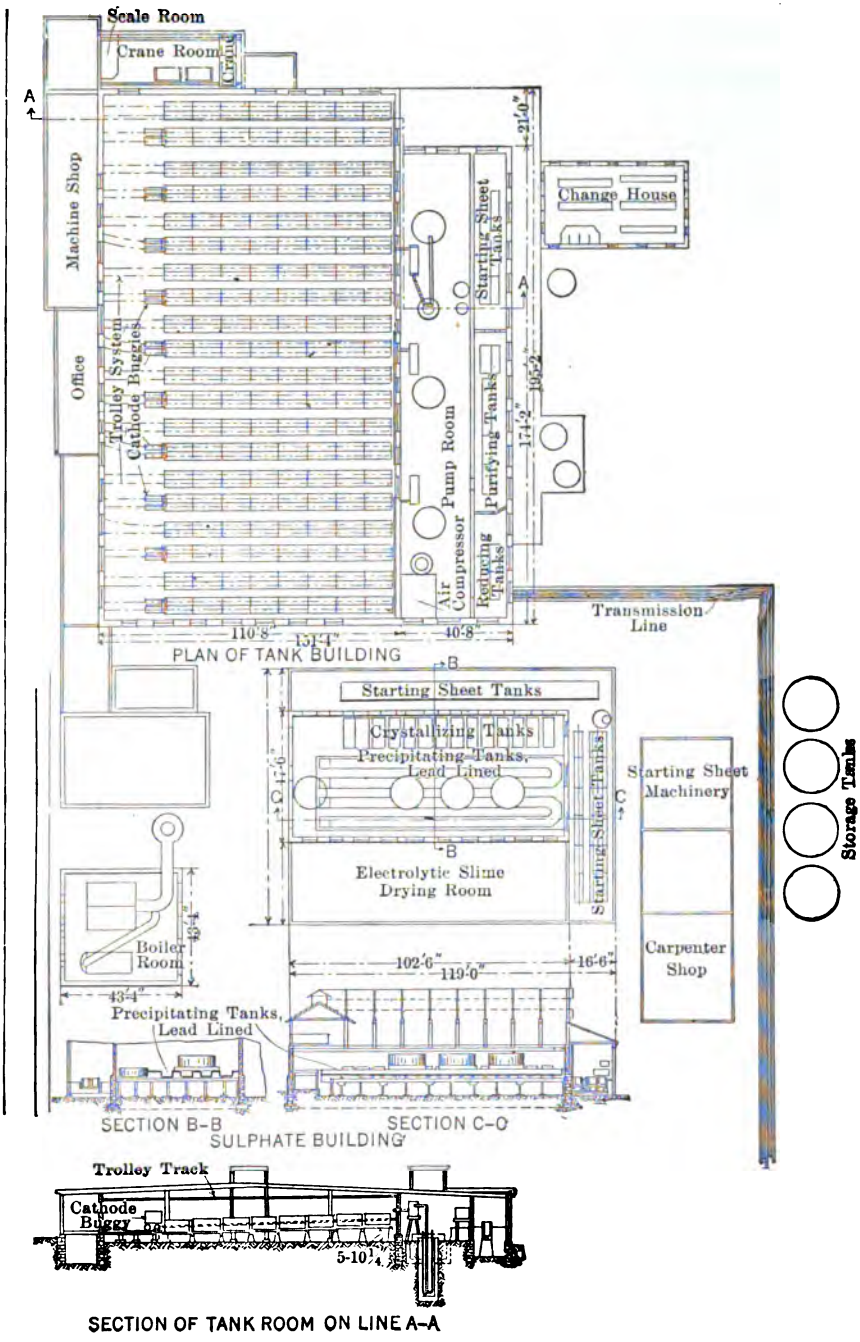



FIG. 3.—GREAT FALLS ELECTROLYTIC PLANT AS REMODELED.

The original refining tanks were lined with ⅛-in., 8-lb. chemical sheet lead and the overflow consisted of a 1-in. lead pipe burned to the lining. Tanks constructed in this manner were found to last about 10 years in service. As they gave out they were replaced by tanks of the same dimensions lined with ⅛-in., 7-lb. lead containing 6 per cent. antimony. The cast chutes of the same material were built into the ends of the tanks. Tanks thus constructed have now been in service for 11 years and do not show much deterioration. The antimonial lead is harder and

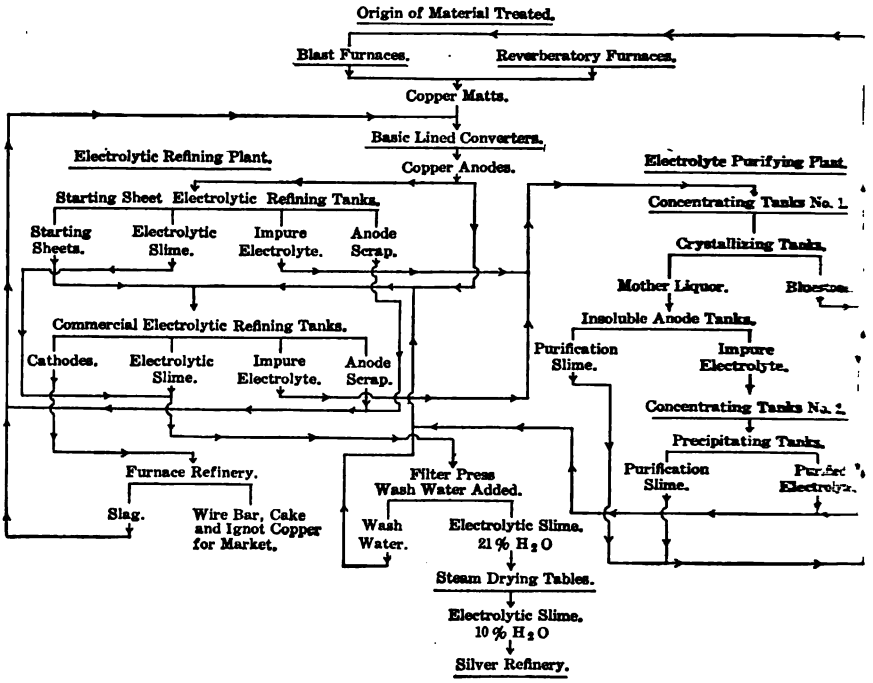



FIG. 4.—FLOW SHEET OF GREAT FALLS ELECTROLYTIC PLANT.

has a less tendency to creep and buckle under extreme changes of temperature than is the case with the chemical lead. Practically all of our tanks are now lined with antimonial lead.

When the change from 4,000 to 9,000 amperes was made in 1896 it became necessary to increase the size of the tank busbars. The ¾ by 4 in. rolled-copper conductors were removed from the sides of the tanks and a taper bar 4½ by 2 in. in the center and 1.5 by 3 in. at the ends was laid on the top of the tanks, as shown in Fig. 2. The bars were cast at our works from wire-bar copper. These bars are insulated from the tanks by means of ½ by 4 in. pine strips previously boiled in pine tar. These strips are renewed once in six months.

Into the ends of the busbars 7/8 by 6 in. iron studs are screwed to receive the copper shunt used to cut four tanks out of circuit for removal of slime.

The anode and cathode rods and the connection plates, used between the tanks, are also cast at our furnace refinery in copper molds. Details of tank connections are shown in Fig. 2.

## IV. Anodes.

As previously mentioned this refinery is the only plant of its kind treating converter-copper anodes. The general practice elsewhere is to

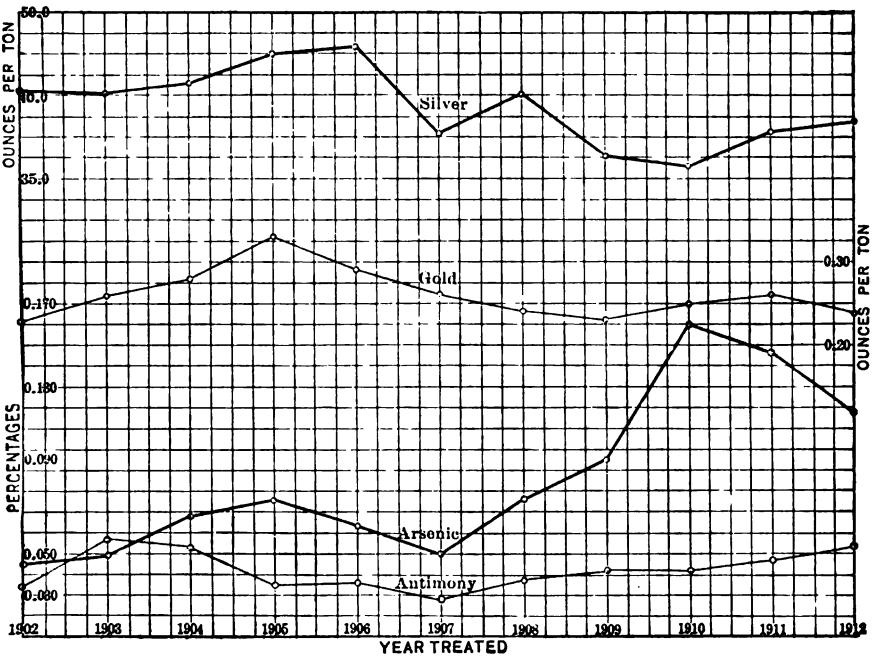



Fig. 5.—Converter Anodes.

transfer the molten copper from the converter to the reverberatory furnace. Here the necessary oxidizing and poling of the copper to bring it to the proper pitch for casting takes place. The copper is then cast into anodes by means of a suitable casting machine.

This refining process has the effect of increasing the copper contents of the metal by from 0.3 to 0.4 per cent., sulphur dioxide being the principal impurity eliminated. The resulting anode is much denser than the converter anode and the casting is free from the uneven surfaces which appear on the converter anode. The analyses of anodes are shown in Table I and Fig. 5.

TABLE I.—*Analyses of Anodes, Electrolyte, Wire Bar and Electrolytic Slime.*

| | Converter Anodes Per Cent. | Electrolyte Per Cent. | Wire Bar Per Cent. | Electrolytic Slime Per Cent. |
|---|---|---|---|---|
| Copper | 99.1300 | 3.280 | 99.9500 | 43.3400 |
| Arsenic | 0.1183 | 0.500 | 0.0016 | 3.0300 |
| Antimony | 0.0534 | 0.041 | 0.0015 | 3.4600 |
| Nickel | 0.0420 | 0.377 | 0.0006 | 0.0800 |
| Cobalt | 0.0018 | 0.016 | Trace | 0.0060 |
| Bismuth | 0.0038 | 0.021 | 0.0004 | 0.1100 |
| Iron | 0.0110 | 0.600 | 0.0006 | 0.3640 |
| Silver | 0.1371 | None | 0.0030 | 17.1870 |
| Gold | 0.0008 | None | Trace | 0.1200 |
| Selenium | 0.0090 | None | ..... | 1.2000 |
| Tellurium | 0.0170 | None | ..... | 2.1000 |
| Lead | 0.0065 | Trace | Trace | 0.7600 |
| Zinc | 0.0035 | 0.418 | 0.0001 | 0.0900 |
| Sulphur | 0.2610 | ..... | 0.0025 | 13.2100 |
| Oxygen | ..... | ..... | 0.0350 | ..... |
| Silicon | ..... | ..... | ..... | 0.1770 |
| Chlorine | ..... | 0.0040 | ..... | 0.0260 |
| Carbon | ..... | ..... | ..... | 0.5900 |
| Platinum | ..... | ..... | ..... | 0.000166 |
| Free sulphuric acid | ..... | 13.0300 | ..... | ..... |
| Specific gravity | ..... | 1.220 | ..... | ..... |

Table II shows one of, a number of comparisons, made at this plant, between converter and refined anodes. In conducting this test three divisions, of 32 refining tanks each, were employed. One-half of the tanks in each division were treating refined anodes while the other half contained converter anodes. By this arrangement the effect of the persona equation of the men in charge of the tanks was reduced to a minimum.

It will be noted that while the ampere efficiency of deposit was 3.6 per cent. higher in the refined-anode tanks than in the tanks containing converter anodes the production per kilowatt-hour was slightly higher in the converter-anode tanks. Subsequent experiments have shown that the refined anodes admit of closer spacing than do converter anodes, without the same reduction in ampere efficiency, the effect of which is to decrease the resistance and increase the production per kilowatt. The difference, however, was not found to be great.

The advantages in favor of the refined anodes, as they appear in Table II, are: First, higher grade of slime resulting in a saving in cost of slime refining; second, lower silver contents of cathodes, all of the silver in the cathode representing a dead loss; third, the lower percentage of anode scrap.

When reduced to dollars per ton of copper refined the sum of the

savings that would result from the use of refined anodes, in a plant of this type, was found to be less than one-half the cost per ton of the reverberatory-furnace treatment of the copper. The cathodes produced from the two types of anodes differed but little in physical appearance.

TABLE II.—*Comparison of Converter and Refined Anodes Cast in the Same Molds.*

| | Converter Anodes | Refined Anodes |
|---|---|---|
| Number of days covered by test | 50 | 50 |
| Number of refining tanks employed | 48 | 48 |
| Average Analyses of Anodes: | | |
| Per cent. Cu | 98.91 | 99.27 |
| Per cent. As+Sb | 0.072 | 0.071 |
| Oz. Ag per ton | 59.09 | 61.14 |
| Oz. Au per ton | 0.200 | 0.219 |
| Average Analyses of Electrolyte: | | |
| Specific gravity | 1.20 | 1.20 |
| Grams per liter Cu | 43.5 | 43.5 |
| Grams per liter free acid | 160 | 160 |
| Grams per liter As | 11.97 | 11.97 |
| Grams per liter Sb | 0.49 | 0.49 |
| Grams per liter Fe | 10.09 | 10.09 |
| Grams per liter Cl | 0.045 | 0.045 |
| Average Temperature of Electrolyte: | | |
| Inlet of 8-tank cascade, C° | 58 | 58 |
| Outlet of 8-tank cascade, C° | 54 | 54 |
| Rate of circulation of electrolyte, gal. per min | 6 | 6 |
| Number of anodes per tank | 20 | 20 |
| Number of cathodes per tank | 20 | 20 |
| Average weight per new anode, lb | 525 | 632 |
| Average thickness per new anode, in | 3 | 3 |
| Distance, center of anode to center of cathode, in | 2.87 | 2.87 |
| Active cathode surface per tank, sq. ft | 252 | 252 |
| Average amperes per tank | 8,387 | 8,387 |
| Average volts for 48 tanks | 27.21 | 28.53 |
| Average volts per tank | 0.567 | 0.594 |
| Average kilowatts for 48 tanks | 228.2 | 239.3 |
| Total copper deposited, lb | 1,103,749 | 1,148,749 |
| Average age of cathodes drawn | 2½ | 2½ |
| Average ampere efficiency of deposit, per cent | 88.3 | 91.9 |
| Average amperes per sq. ft. cathode surface | 33.3 | 33.3 |
| Average lb. copper deposited per kilowatt-hour | 4.03 | 4.00 |
| Average oz. per ton silver in cathodes | 1.25 | .95 |
| Average per cent. As+Sb in cathodes | 0.0043 | 0.0043 |
| Average per cent. anode scrap | 8.00 | 5.30 |
| Analyses of silver slime: | | |
| Per cent. Cu | 40.3 | 18.80 |
| Oz. Ag per ton | 6,755.00 | 14,079 |
| Oz. Au per ton | 18.34 | 38.45 |

In a crane-operated electrolytic refinery, where the anodes and cathodes are handled in tank units and not individually as here, difficulties would result if converter anodes were used because of the fact that converter anodes corrode less uniformly than refined anodes. Many of the converter

anodes would become scrap before the date set for drawing the scrap from a tank and an equal number of pieces would not be scrap at the appointed time for drawing.

The above comparison, together with other tests made at this plant, has demonstrated that refined anodes, while very desirable, would for this plant be an expensive luxury.

Fig. 6 shows the earlier form of anode used in this plant and the Morrow clip type[1] now used. The advantage of the clip type of anode is that the amount of inactive copper above the solution line is much less than is the case with the lug type. The average percentage of anode scrap for the year 1912 was 5.9; the average weight of the converter anode is 500 lb. The scrap resulting from each anode weighs about 30 lb.; this is resmelted in the converters. The clip type of anode also has the advantage, as compared with the lug type, of being suspended in such a manner as always to hang plumb in the tank. The standard converter anode measures 24.5 by 35.75 in.

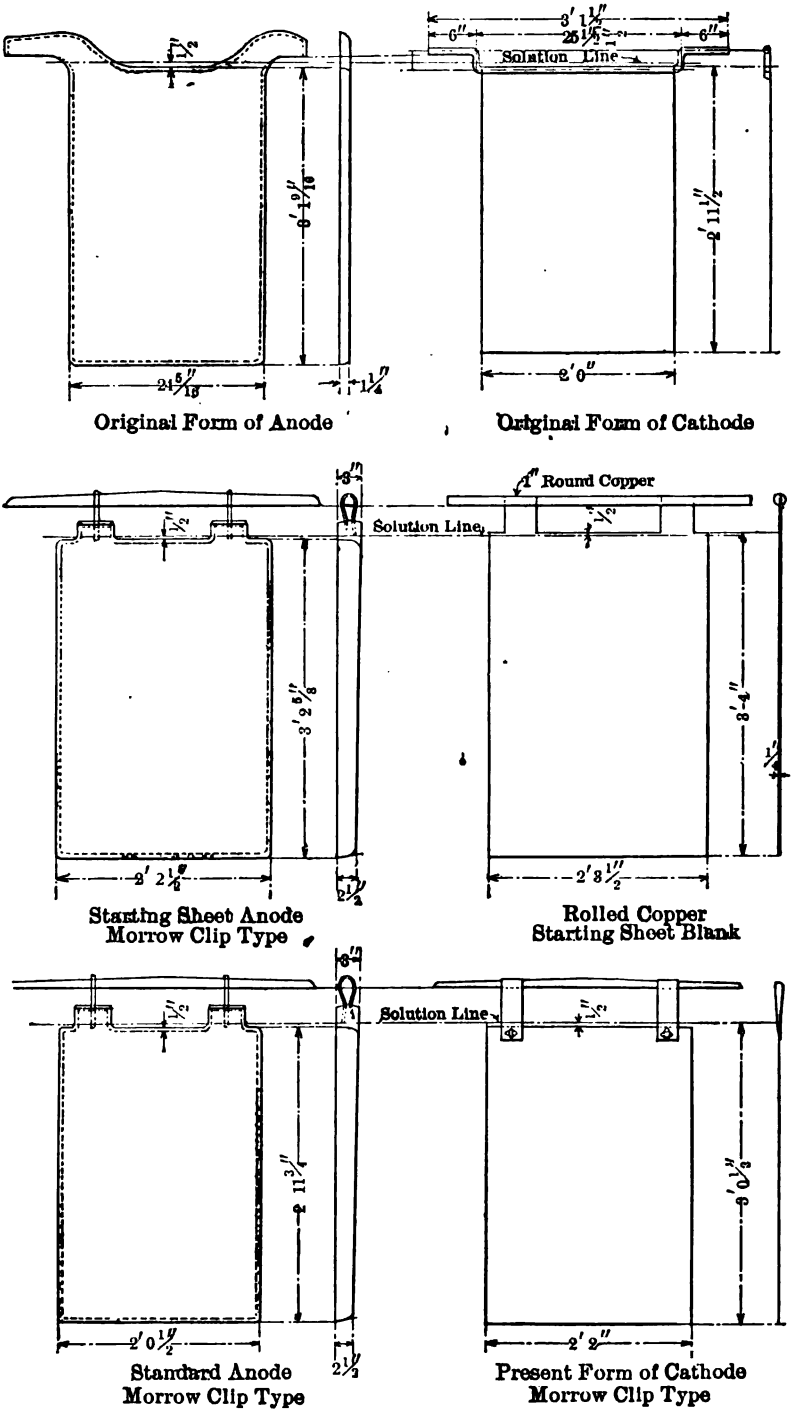
The clip or hanger is made from a ¼-in. round copper rod 60 ft. long rolled in a rod mill from a bar cast at the refining furnace. This rod is cut and bent into the loop form, in an automatic machine, and placed in the anode mold. Both the anodes and cathodes are supported in the tanks by cast-copper rods as shown in sketch. The anodes are spaced 5.2 in. center to center. It will be noted in Fig. 6 that the anode is 3 in. thick at the top and 2.5 in. thick at the bottom. It is found that the anode corrodes more rapidly at the upper end where the current enters than at the lower end, that the wedge shape yields a lower percentage of scrap and that the anode retains its original shape for a longer time than is the case with an anode of uniform thickness.

## V. Cathodes.

The starting sheets are produced in 44 standard refining tanks divided into two circuits connected in parallel in the main circuit and are thus operating at one-half the current density of the commercial refining tanks. Each tank contains 21 cathode blanks and 22 anodes. The blanks are made of ¼-in. rolled copper 27.5 by 39.5 in. below the solution line, as shown in Fig. 6. The anodes used in these tanks are of the same type as the standard anode but larger, being 26.5 by 38⅝ in.

The 12-hr. sheets are stripped at an average weight of 4 lb. each, measuring 26 by 36.5 in. after trimming. The men employed in stripping the plates do their work directly over the tanks using a traveling bench suspended from the trolley track as shown in Fig. 7. This bench, on which the sheets are piled after stripping, is moved from tank to tank as the

[1] U. S. patents No. 621,121 and 631,471.




Original Form of Anode

Original Form of Cathode

Starting Sheet Anode
Morrow Clip Type

Rolled Copper
Starting Sheet Blank

Standard Anode
Morrow Clip Type

Present Form of Cathode
Morrow Clip Type

FIG. 6.—ANODES, CATHODES, AND STARTING SHEET BLANK.

men advance. The blanks are covered with a thin coating of low-grade mineral oil to which 0.5 lb. of artificial graphite per gallon of oil is added. Grooved wooden strips are used on the edges of the plates to prevent deposition of copper. The strippers work independently and each man strips and delivers to the trimmer 360 sheets per 8-hr. shift. The electrolyte used in these tanks differs slightly from that of the main tank room. An average analysis is given in Table III.

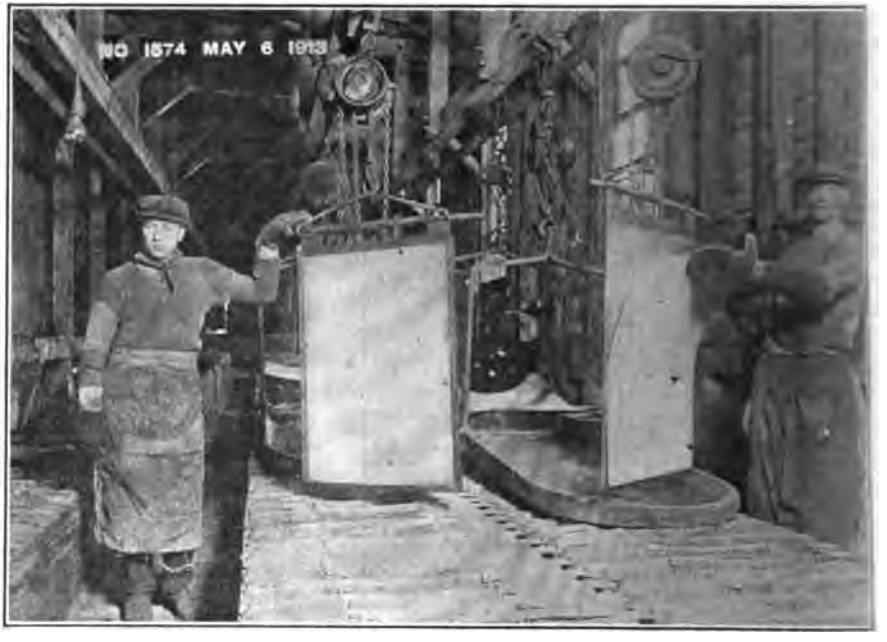



FIG. 7.—STRIPPING STARTING SHEETS.

TABLE III.—*Starting-Sheet Tank Electrolyte.*

| | |
|---|---|
| Specific gravity | 1.175 |
| Free $H_2SO_4$, grams per liter | 120.0 |
| Cu, grams per liter | 40.0 |
| As, grams per liter | 5.0 |
| Sb, grams per liter | 0.4 |
| Fe, grams per liter | 4.5 |
| Cl, grams per liter | 0.04 |

Table IV shows the effect of sulphuric acid in the electrolyte on the efficiency of deposition, the production per kilowatt-hour increasing as the acid increases. The sheets produced when the acid is about 120 g. per liter are found to be smoother and tougher than when acid is higher.

The starting sheets are trimmed on the four edges to prevent short circuits. Two clips or loops, cut from similar sheets, are then attached to each sheet by means of the Morrow clip machine[2] (see Fig. 6).

TABLE IV.—*Effect of Free Sulphuric Acid on Starting-Sheet Production.*

Converter anodes, 39.5 by 27.5 by 0.25 in. blanks, 3 in. anode; 2.5 in. center of anode to center of blank.

Circulating electrolyte at 4 gal. per minute.

| Amperes | Amperes per Sq. Ft. | Efficiency | E. M. F. per Tank Volts | Lb. Cu per Kilowatt-hour | Average Weight per Sheet | Electrolyte | | |
|---|---|---|---|---|---|---|---|---|
| | | | | | | Gal. per Liter Free $H_2SO_4$ | Cu | Temperature C° |
| 4,580 | 16.2 | 86.5 | 0.475 | 4.75 | 5.86 | 74 | 42.6 | 52.5 |
| 4,490 | 15.9 | 86.5 | 0.462 | 4.84 | 5.78 | 74 | 42.6 | 52 |
| 4,453 | 15.7 | 85.2 | 0.462 | 4.80 | 5.61 | 70 | 40.1 | 55 |
| 4,450 | 15.7 | 83.6 | 0.475 | 4.55 | 5.50 | 69 | 39.8 | 49 |
| 4,390 | 15.5 | 90.3 | 0.482 | 4.84 | 5.87 | 69 | 40.1 | 50 |
| 4,403 | 15.6 | 90.2 | 0.496 | 4.71 | 5.87 | 73 | 38.4 | 47.5 |
| 4,290 | 15.2 | 88.2 | 0.462 | 4.94 | 5.60 | 80 | 38.9 | 51 |
| 4,397 | 15.5 | 91.6 | 0.470 | 5.04 | 5.93 | 82 | 38.8 | 52 |
| 4,538 | 16.0 | 90.3 | 0.476 | 4.90 | 6.06 | 88 | 38.8 | 51 |
| 4,461 | 15.8 | 91.9 | 0.450 | 5.28 | 6.06 | 89 | 39.9 | 52 |
| 4,408 | 15.6 | 92.5 | 0.426 | 5.63 | 6.03 | 110 | 40.4 | 52 |
| 4,506 | 16.0 | 92.5 | 0.452 | 5.29 | 6.17 | 109 | 38.9 | 53 |
| 4,496 | 15.9 | 91.7 | 0.424 | 5.55 | 6.10 | 116 | 40.6 | 53 |
| 4,512 | 16.0 | 90.3 | 0.426 | 5.50 | 6.02 | 126 | 40.5 | 50 |
| 4,550 | 16.1 | 89.7 | 0.420 | 5.53 | 6.04 | 128 | 40.4 | 48 |
| 4,430 | 15.7 | 90.7 | 0.400 | 5.86 | 5.94 | 129 | 42.6 | 48 |
| 4,435 | 15.7 | 84.7 | 0.385 | 5.69 | 5.56 | 128 | 42.0 | 48 |
| 4,420 | 15.6 | 87.3 | 0.365 | 6.09 | 5.62 | 126 | 42.4 | 50 |
| 4,445 | 15.7 | 91.8 | 0.357 | 6.23 | 6.03 | 124 | 43.6 | 48 |
| 4,453 | 15.7 | 85.1 | 0.360 | 6.13 | 5.60 | 132 | 42.1 | 46 |
| 4,404 | 15.6 | 87.0 | 0.342 | 6.55 | 5.66 | 148 | 40.8 | 49 |
| 4,450 | 15.7 | 88.6 | 0.350 | 6.62 | 5.82 | 149 | 42.3 | 46 |

NOTE: The sheets produced with between 116 and 126 g. per liter of acid were the toughest and best.

The starting sheets are now ready for the tanks. It will be noted that the cathodes hang ¾ in. lower in the tank than the anode and are 1.5 in. wider than the anode. When using anodes and cathodes of the same size a cathode very rough on the edges resulted; increasing the size of the cathode has resulted in a cathode with comparatively smooth edges and an increase of 5 per cent. in ampere efficiency.

When the output of the generators is 2,000 kw. or more the best results are obtained while drawing 2- and 3-day cathodes. With a lower generator output older cathodes are drawn. The generator output is

[2] U. S. Patent 600,498, Mar. 8, 1898.

governed by the head on the water wheels, which is dependent upon the stage of water in the river.

To determine the most economical age of cathodes and number of electrodes per tank at which to operate, the labor, the production per kilowatt-hour, the silver lost in cathodes and the impurities in the anodes and cathodes must be considered. Assuming the quality of the product

TABLE V.—*Effect of Varying the Age of Cathodes and the Number of Electrodes per Tank.*

| Age of Cathodes Days | Electrodes per Tank | | Average Amperes | Average Amperes per Sq. Ft. | Ampere Efficiency Per Cent. | Cu per Kilowatt-hour Lb. | Cathode Oz. Ag per Ton | Analyses As + Sb Per Cent |
|---|---|---|---|---|---|---|---|---|
| | Anodes[a] | Cathodes | | | | | | |
| 4 | 20 | 20 | 9,300 | 36.9 | 88.0 | 3.93 | 1.32 | 0.0036 |
| 2 | 20 | 20 | 8,808 | 35.0 | 90.85 | 3.72 | 0.83 | 0.0030 |
| 3 | 20 | 20 | 8,877 | 35.2 | 89.00 | 3.75 | 0.83 | 0.0032 |
| 2 | 21 | 21 | 9,035 | 34.1 | 90.90 | 3.84 | 0.89 | 0.0032 |
| 3 | 21 | 21 | 9,223 | 34.8 | 89.40 | 3.87 | 1.02 | 0.0029 |
| 2 | 22 | 22 | 9,071 | 32.6 | 90.50 | 4.02 | 0.89 | 0.0033 |
| 3 | 22 | 22 | 9,167 | 33.0 | 88.80 | 4.07 | 0.95 | 0.0030 |

[a] Converter anodes. Average analysis of anodes: Cu, 99.13; As, 0.127; Sb, 0.055 per cent.; Ag, 33.97 oz. per ton; Au, 0.22 oz.

as regards impurities to remain satisfactory under the different conditions the cost per ton remains as the only consideration. Table V is a summary of various comparisons made between different ages of cathodes with varying numbers of electrodes per tank.

Table VI is the comparison, in detail, between 21 and 22 pairs of electrodes per tank while drawing 2- and 3-day cathodes.

## VI. THE ELECTROLYTE.

The 320 commercial refining tanks are divided into three sections, two sections containing 96 tanks each and the third section 128 tanks.

Each section is provided with a separate electrolyte and circulating system. Copper and acid determinations are made daily on samples of each electrolyte. As, Sb, Fe and Cl determinations are made weekly.

The two groups of starting-sheet tanks each have a separate electrolyte which is sampled and assayed in the same manner as the commercial-tanks electrolyte.

The analysis of the electrolyte in the commercial-refining tanks is shown in Table I in terms of percentage in conformity with the analysis of the other materials shown in the table. The analysis of solutions is ordinarily reported in terms of grams per liter. With a current density

TABLE VI.—*Effect of Varying the Age of Cathodes and Number of Electrodes per Tank.*

Details of 21 and 22 pairs of electrodes per tank in Table V.
Standard Converter Anodes.

| | 21 Anodes and 21 Cathodes per Tank | 22 Anodes and 22 Cathodes per Tank |
|---|---|---|
| Number of refining tanks used | 160 | 160 |
| Copper deposited, 2-day cathodes, lb | 1,778,352 | 1,704,743 |
| Copper deposited, 3-day cathodes, lb | 1,742,678 | 1,889,401 |
| Total copper deposited, lb | 3,521,030 | 3,594,144 |
| Spacing of electrodes, anode centers, in | 5.45 | 5.20 |
| Average amperes for 2-day cathodes | 9,035 | 9,071 |
| Average amperes for 3-day cathodes | 9,223 | 9,167 |
| Average amperes per sq. ft., 2-day cathodes | 34.1 | 32.6 |
| Average amperes per sq. ft., 3-day cathodes | 34.8 | 33.0 |
| Average ampere efficiency, 2-day cathodes, per cent | 90.9 | 90.5 |
| Average ampere efficiency, 3-day cathodes, per cent | 89.4 | 88.8 |
| Average drop per tank in volts, 2-day cathodes | 0.612 | 0.581 |
| Average drop per tank in volts, 3-day cathodes | 0.600 | 0.562 |
| Average copper deposited per kilowatt-hour, 2-day cathodes, lb | 3.837 | 4.021 |
| Average copper deposited per kilowatt-hour, 3-day cathodes, lb | 3.871 | 4.075 |
| Average copper deposited per kilowatt-hour, 2- and 3-day cathodes, lb | 3.854 | 4.048 |
| Analyses of Cathodes: | | |
| 2-day cathodes, As + Sb, per cent | 0.0032 | 0.0033 |
| 3-day cathodes, As + Sb, per cent | 0.0029 | 0.0030 |
| 2-day cathodes, Ag, oz. per ton | 0.89 | 0.89 |
| 3-day cathodes, Ag, oz. per ton | 1.02 | 0.95 |
| | Same both set of Tanks | |
| Analyses of Electrolyte: | | |
| Specific gravity | 1.219 | |
| Cu, grams per liter | 41.1 | |
| Free acid, grams per liter | 153 | |
| As, grams per liter | 6.49 | |
| Sb, grams per liter | 0.51 | |
| Fe, grams per liter | 5.11 | |
| Cl, grams per liter | 0.044 | |
| Average temperature of electrolyte (8 tanks in cascade): | | |
| Inlet tank, C° | 57 | |
| Outlet tank, C° | 53 | |
| Speed of circulation of electrolyte, gal. per min | 6 | |
| Analyses of Anodes (Converter): | | |
| Cu, per cent | 99.13 | |
| As, per cent | 0.127 | |
| Sb, per cent | 0.055 | |
| Ag, oz. per ton | 33.97 | |
| Au, oz. per ton | 0.22 | |

of 34 amperes per square foot and a circulation of 6 gal. of electrolyte per minute the best results, while treating converter anodes, are obtained with an electrolyte carrying 40 g. per liter of copper and 160 g. per liter of free sulphuric acid.

The circulation of the electrolyte is through a cascade of eight tanks

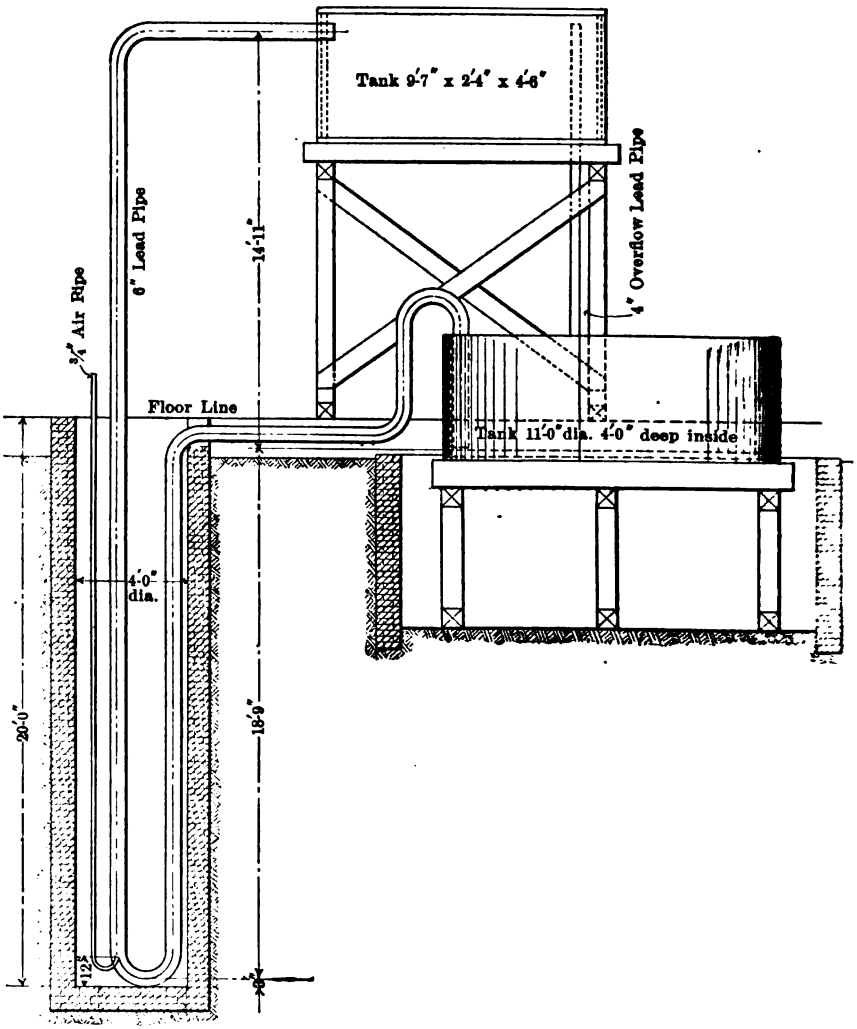



FIG. 8.—ELEVATION OF SOLUTION AIR-LIFT PUMP.

as already described. In the pump room the electrolyte is delivered to the receiving tank, a round lead-lined tank 11 ft. in diameter and 4.5 ft. deep (see Fig. 3). In this tank are 150 ft. of 1-in. 8-lb. antimonial-lead

pipe; live steam is supplied to these coils at a pressure of 65 lb. per square inch, by means of which the electrolyte is heated. From the receiving tank the electrolyte is raised to the discharge tank from which it is again conducted to the head tank of the cascade.

Various means of elevating the electrolyte have been employed in this plant. The first method used was the steam injector constructed of lead and antimony. These pumps were of limited capacity and the dilution of the solution was excessive. They were followed by a system of lead-lined cast-iron eggs in which the electrolyte was collected and blown to

*Table VII.—Effect of Speed of Circulation of Electrolyte on the Concentration of the Solution.*

Current density 36.0 amperes per square foot, 4-day cathodes, converter anodes.

Circulating Electrolyte at Rate of 6 gal. per minute.

| Zone | Specific Gravity | Grams per Liter | | | | | | Copper per Kilowatt-hour Lb. |
|---|---|---|---|---|---|---|---|---|
| | | Acid | Cu | As | Sb | Fe | Cl | |
| 1 | 1.213 | 174 | 38.4 | 7.5 | 0.56 | 2.95 | 0.044 | |
| 2 | 1.216 | 172 | 39.2 | 7.2 | 0.59 | 2.95 | 0.045 | |
| 3 | 1.216 | 171 | 40.2 | 7.1 | 0.59 | 2.95 | 0.043 | |
| 4 | 1.228 | 153 | 49.9 | 7.2 | 0.59 | 2.90 | 0.044 | |
| Average | 1.218 | 168 | 41.9 | 7.3 | 0.58 | 2.94 | 0.044 | 3.92 |

Circulating Electrolyte at the Rate of 4 gal. per minute.

| | | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| 1 | 1.211 | 163 | 40.0 | 7.3 | 0.52 | 2.90 | 0.042 | |
| 2 | 1.214 | 161 | 41.1 | 7.3 | 0.55 | 2.92 | 0.040 | |
| 3 | 1.216 | 159 | 42.0 | 7.4 | 0.58 | 2.95 | 0.040 | |
| 4 | 1.263 | 139 | 67.8 | 7.4 | 0.55 | 2.90 | 0.040 | |
| Average | 1.226 | 156 | 47.7 | 7.4 | 0.55 | 2.92 | 0.040 | 3.85 |

Circulating Electrolyte at rate of 2 gal. per minute.

| | | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| 1 | 1.210 | 167 | 37.8 | 7.6 | 0.52 | 2.90 | 0.042 | |
| 2 | 1.213 | 165 | 39.0 | 7.1 | 0.49 | 2.95 | 0.042 | |
| 3 | 1.217 | 162 | 41.0 | 7.6 | 0.51 | 2.95 | 0.042 | |
| 4 | 1.265 | 146 | 66.4 | 7.9 | 0.55 | 2.95 | 0.041 | |
| Average | 1.226 | 160 | 46.1 | 7.6 | 0.52 | 2.94 | 0.042 | |

Sampled after circulation had been entirely shut off for 7 hr. with current passing at the rate of 36 amperes per square foot.

| | | | |
|---|---|---|---|
| 1 | 1.185 | 179 | 23.4 |
| 2 | 1.210 | 164 | 38.0 |
| 3 | 1.230 | 151 | 51.8 |
| 4 | 1.255 | 138 | 65.5 |
| Average | 1.220 | 158 | 44.7 |

the overhead tank with compressed air. This system was very inefficient and gave way to a type of boiler-feed pump, the water end of which was of bronze. These soon became leaky from corrosion and were abandoned. A type of plunger pump with a lead-lined barrel and rubber valve and plunger was next tried. This pump was continued in service for several years with satisfactory results but was replaced 10 years ago by the air lift, which has continued to hold the field. The air lift as used here is constructed of cast lead-antimony pipe 6 in. inside diameter, ¾-in. walls. The pipe is cast in 4-ft. lengths with flanged joints. A brick wall 20 ft. deep and 4 ft. diameter, made waterproof, with a cement lining, contains that portion of the lift that is below the floor level. Working against a head of 14 ft. 8 in., 160 gal. of electrolyte (1.22 specific gravity) is raised per minute with a consumption of 80 cu. ft. of free air compressed to 16 lb. per square inch (see Fig. 8).

The lift, while it is of low mechanical efficiency and sensitive to variations in the air pressure, is very satisfactory in its operation, having no moving parts and requiring but little attention. The loss in temperature of the electrolyte during its passage through the air lift from the receiving tank to the discharge tank averages 0.5° C.

The compressed air for the operation of the lift is obtained from the converter-plant blowing engines which are operated by water power. A motor-driven compressor at the tank house supplies air in cases of emergency.

Circulation of the electrolyte is maintained at a speed of 6 gal. per minute, a lower circulation reducing the production per kilowatt-hour while a higher rate has the effect of disturbing too much the settlement of the slime and increasing the silver in cathodes. Table VII shows the effect on the concentration of the electrolyte of varying the speed of circulation. The samples which were taken in the regular refining tank and marked zones 1, 2, 3, and 4 were obtained in the following manner: A glass tube, closed at the top, was lowered to a point 6 in. below the surface of the electrolyte and a sample collected at this point and called zone 1. Zone 2 sample was taken in the same manner at a point 17 in. below the surface; No. 3, 28 in. below the surface and No. 4, 39 in. below the surface. The column of electrolyte in the tanks was 40 in. high, there being 3 in. of slime in the tanks. These figures represent the averages of 104 samples taken during seven consecutive days from 48 refining tanks at 36 amperes per square foot. The samples were all taken between the center pair of electrodes.

These figures show a rather remarkable concentration of the electrolyte as regards copper and acid with negative results regarding impurities in solution and demonstrated that, while operating at 36 amperes per square foot, circulating the electrolyte at the rate of 6 gal. per minute,

delivering the solution at the surface of the tank and discharging from a point near the bottom, fails to maintain the electrolyte in a homogeneous state but is found to be the most economical speed at which to operate.

The arsenic and antimony are the impurities with which we are chiefly concerned in our refining operations, and as their effects on the conductivity of the copper are much alike our practice is to report them combined as arsenic plus antimony in the cathodes, making a separation on the average monthly sample to note the relations. These impurities in the electrolyte and in the anodes are determined separately.

There is no silver in solution in the electrolyte and all of the silver present in the cathodes is deposited mechanically from the slime in suspension; the slime is entrapped on the rough surfaces of the cathodes, the silver contents increasing with the age and roughness of the cathode. The arsenic and antimony are deposited both mechanically from the slime and electrolytically from the electrolyte.

When refining converter anodes, carrying 0.12 per cent. As, at 34 amperes per square foot the arsenic in the electrolyte is not allowed to exceed 7 g. per liter.

The results of an experiment with covered cathodes and converter anodes may be of interest. Three regular cathode starting sheets were placed in separate frames covered with 8-oz. duck in such a manner as to entirely inclose the sheets. These were substituted for three regular cathodes in a commercial-refining tank and after remaining 2.75 days they were removed, sampled and assayed as follows:

*Analysis of Electrolyte.*

| Specific Gravity | Grams per Liter | | | | | |
|---|---|---|---|---|---|---|
| | Acid | Cu | As | Sb | Fe | Cl |
| 1.200 | 152 | 41.2 | 7.0 | 0.44 | 1.80 | 0.04 |

*Analysis of Anodes.*

| Per Cent. Cu | Oz. Ag per Ton | Oz. Au per Ton | Per Cent. As | Per Cent. Sb |
|---|---|---|---|---|
| 99.15 | 39.0 | 0.25 | 0.211 | 0.04 |

*Cathodes Produced.*

| | Inclosed Cathode | Regular Cathode |
|---|---|---|
| Current density, amperes per sq. ft. | 31.6 | 35.7 |
| Age of cathodes, days | 2.75 | 4.0 |
| Ag, oz. per ton | 0.2 | 1.2 |
| As + Sb, per cent | 0.0036 | 0.0042 |

The lower current density of the covered cathodes is due to the increased resistance offered by the covering.

The canvas-covered cathodes were drawn at 2.75 days rather than four days as the canvas was showing signs of failure. The results show that the canvas covering was effective in preventing but little of the silver-bearing slime in suspension from coming in contact with the cathodes but that it had a lesser effect on the arsenic and antimony contents of the deposited copper.

The original method employed at these works of purifying the electrolyte was the manufacture of bluestone. An amount of electrolyte sufficient to maintain the working solutions at the required degree of purity was daily run off from the tank house to the sulphate plant. Here it was heated to the boiling point by means of steam coils and passed through open tanks containing shot copper into which compressed air was blown until all of the free acid was neutralized. After the necessary concentration by boiling the solution was allowed to cool in an open chute where most of the copper crystallized out. The crystals from the chute were redissolved in water, concentrated, and run into the final crystallizing tanks. The crystals from these tanks were washed, dried and packed in barrels for shipment. The mother liquor, after it had become too foul for further use, was run over scrap iron for the recovery of the small amount of copper remaining in solution. After passing over the iron the solution, which contained only the impurities, was wasted.

This was an efficient method of freeing the electrolyte of impurities and would probably still be employed had not the market for the bluestone failed during 1899. In 1897 2,300,000 lb. of bluestone were produced and shipped from our sulphate plant.

The electrolytic method was next employed for purifying the electrolyte. This process consisted in removing from the working solutions each day such an amount of electrolyte as was necessary to maintain the purity of the electrolyte. This solution was passed through four standard refining tanks containing lead anodes and copper or lead cathodes. These tanks were arranged in a cascade the electrolyte flowing from one tank to another. The Cu, As and Sb contents of the solution, as it left the last tank, depended on the speed of flow. With a circulation of 2 liters per minute but a trace of Cu, As and Sb remained. The ampere efficiency of deposit, however, was extremely low at this rate of flow.

Table VIII shows the results obtained while circulating at 4 liters per minute. The percentages of elimination as shown in this table are high while the ampere efficiency is very low, the two lower tanks in the cascade doing but little work. In subsequent experiments the most economical speed of flow was found to be 7 liters per minute with an elimination of 99 per cent. of the copper, 78 per cent. of the arsenic and 91.1

per cent. of the antimony with a total ampere efficiency of about 50 per cent.

The increase in the iron contents of the electrolyte between the inlet of the first and outlet of the last tank, as shown in Table VIII, is due to

TABLE VIII.—*Removal of Copper, Arsenic and Antimony from Electrolyte in Insoluble Anode Tanks.*

Circulation, four liters per minute. Lead anodes, copper cathodes, 9,000 amperes, 31.8 amperes per square foot.

| | Grams per Liter | | | | | Volts per Tank | Temperature, C.° |
|---|---|---|---|---|---|---|---|
| | Acid | Cu | Fe | As | Sb | | |
| Inlet tank No. 1... | 144 | 37.060 | 6.242 | 3.200 | 0.463 | ..... | 17 |
| Outlet tank No. 1. | 184 | 7.376 | 6.813 | 2.240 | 0.260 | 2.22 | 42 |
| Outlet tank No. 2. | 194 | 0.504 | 7.364 | 0.400 | 0.061 | 2.25 | 57 |
| Outlet tank No. 3. | 208 | 0.088 | 7.701 | 0.056 | 0.038 | 2.25 | 64 |
| Outlet tank No. 4. | 216 | 0.048 | 7.915 | 0.028 | 0.028 | 2.25 | 65 |

the concentration of the electrolyte by reason of the heat evolved by the electrolysis. These analyses are corrected on the basis of an unchanged volume of electrolyte in Table IX.

The process thus far described has removed only the Cu, As and Sb from the solution, leaving the Fe, Ni, Bi and Zn. To remove these the electrolyte, as it leaves the insoluble-anode tanks, is run into a lead-lined tank 13 ft. in diameter and 4.5 ft. deep lined with 12-lb. chemical sheet lead and containing 600 ft. of 1-in., 8-lb. chemical-lead pipe. In this tank it is concentrated to 55° Bé., decanted to an open tank 10 ft. long, 4 ft. wide and 3 ft. deep, where it is allowed to stand for four days, during which time the salts of Fe, Ni, Bi and Zn crystallize out leaving a solution of

TABLE IX.—*Corrected Analyses.*

| | Grams per Liter | | | | | Percentage Elimination of Original Amounts | | | Ampere Efficiency Per Cent. |
|---|---|---|---|---|---|---|---|---|---|
| | Acid | Cu | Fe | As | Sb | Cu | As | Sb | |
| Inlet tank No. 1.... | 144 | 37.060 | 6.242 | 3.200 | 0.4630 | ..... | ..... | ..... | ..... |
| Outlet tank No. 1.. | 169 | 6.760 | 6.242 | 2.050 | 0.2380 | 81.8 | 35.9 | 48.7 | 71.70 |
| Outlet tank No. 2.. | 165 | 0.427 | 6.242 | 0.339 | 0.0517 | 17.1 | 53.5 | 40.2 | 19.50 |
| Outlet tank No. 3.. | 169 | 0.071 | 6.242 | 0.045 | 0.0308 | 0.9 | 9.2 | 4.7 | 1.68 |
| Outlet tank No. 4.. | 170 | 0.038 | 6.242 | 0.022 | 0.0220 | 0.1 | 0.7 | 1.7 | 0.15 |
| Totals and averages..... | .... | ..... | ..... | ..... | ..... | 99.9 | 99.3 | 95.3 | 23.26 |

approximately the following composition to be returned to the tank room: $H_2SO_4$, 1,100; As, 1; Sb, 0.2; Fe, 1; Ni, 5.3; and Zn, 1.5 g. per liter.

The Cu, As and Sb deposited in the insoluble-anode tanks are in the form of a black slime, some of the slime adhering to the cathode but the greater portion settling to the bottom of the tanks. An average assay of this slime is given in Table X.

TABLE X.—*Slime from Insoluble-Anode Tanks.*

(Treating electrolyte direct from tank room.)

| | |
|---|---|
| Moisture, per cent | 10.0 |
| Cu, per cent | 55.1 |
| $SiO_2$, per cent | 1.1 |
| FeO, per cent | 0.4 |
| $Al_2O_3$, per cent | 0.4 |
| CaO, per cent | 0.3 |
| S, per cent | 4.1 |
| As, per cent | 10.3 |
| Sb, per cent | 2.5 |
| Ni, per cent | 0.35 |
| Zn, per cent | 0.32 |
| Ag, oz. per ton | 3.4 |
| Au, oz. per ton | 0.02 |

This material was sent to the blast-furnace plant for treatment.

The method of purifying the electrolyte now in use is a modification of the one just described and consists in running off daily from the tank house about 25,000 liters of solution, concentrating this, by boiling, to 48° Bé. From the boiling tank the concentrated solution is run to crystallizing tanks and allowed to stand for four days. At the end of this period 82 per cent. of the copper will have crystallized out. The mother liquor has the following analysis: Acid, 475; Cu, 17.4; As,

TABLE XI.—*Analysis of Insoluble-Anode Tank Slime.*

(Treating mother liquor from crystallizing tanks.)

| | |
|---|---|
| Moisture, per cent | 9.66 |
| Cu, per cent | 46.30 |
| $SiO_2$, per cent | 0.38 |
| FeO, per cent | 1.66 |
| $Al_2O_3$, per cent | 0.4 |
| CaO, per cent | 1.08 |
| S, per cent | 5.02 |
| As, per cent | 21.48 |
| Sb, per cent | 2.28 |
| Ni, per cent | 0.35 |
| Zn, per cent | 0.32 |
| Ag, oz. per ton | 3.61 |
| Au, oz. per ton | 0.03 |

20.2; Sb, 1.1 and Fe, 15.2 g. per liter. This is run to the insoluble-anode tanks where it is treated for the removal of Cu, As and Sb as described above.

Table XI gives an analysis of the slime obtained when using this method.

After treatment in the insoluble-anode tanks the solution is either returned to the tank room or treated, by further concentration and crystallization as described above, for the removal of the Fe, Ni and Zn.

The analysis of slime from the second concentration and precipitation for the removal of Fe, Ni and Zn is as follows: Moisture, 22.1; S, 16.6; Ni, 5.6; Cu, 1.3; Fe, 10.3; Zn, 2.7 per cent.

The advantage of crystallizing the copper from the solution before treatment in the insoluble-anode tanks is that the ampere efficiency of these tanks is greatly increased because of the concentrated solution treated and that the amount of slime produced is reduced by nearly one-half, as follows:

*Production of Insoluble-Anode Tank Slime.*

| | Wet Slime per Month Lb. | As Per Cent. |
|---|---|---|
| Treating electrolyte direct from tank room..... | 38,333 | 10.30 |
| Removing 82 per cent. of copper before treating..... | 21,600 | 21.48 |

The slime from the insoluble-anode tanks is sent to the blast furnaces for treatment, the iron slime is washed for the removal of free acid and sent to the blast furnace or wasted if free of copper.

The arsenic liberated from the anodes in the refining tanks finds its way into the following products in the proportions shown: In cathodes, 1.22; in silver slimes, 17.19; and in purification slimes and cathodes, 81.59 per cent. It is therefore necessary to remove from the electrolyte 81.59 per cent. of the arsenic liberated at the anode.

## VII. Electrolytic Slime.

From a converter anode carrying 40 oz. silver and 0.24 oz. gold per ton, a slime carrying 43.3 per cent. Cu, 5,000 oz. silver and 34 oz. gold per ton is obtained: See Table I for complete analysis.

This slime is removed from the refining tanks once in 60 days in the following manner: A copper jumper, held by the iron studs provided, connects the two ends of tank bars on adjacent tanks and short circuits four tanks, as shown in Fig. 2. The anodes and cathodes are removed

and the slime, 5.5 in. having accumulated, is diluted with water and pumped by means of a bronze steam injector to a round lead-lined tank, 12 ft. diameter, 4 ft. deep, situated 200 ft. distant in the slime-sampling room. Before entering this tank the slime passes through a lead screen with ⅛-in. holes, ½-in. centers. In this tank the slime is washed with hot water to remove the free acid and soluble copper.

From this tank the slime is dropped into a cast-copper, lead-lined egg from which it is forced by compressed air at 100 lb. pressure into a modified Bushnell filter press containing hard-copper plates and rings, 8-oz. duck being used for filter cloth. Thirteen cakes 26 in. in diameter and 1.25 in. thick are made at each charge; the wet weight is 865 lb. per charge at 21 per cent. moisture. From the filter press the slime is placed on steam-jacketed copper-drying tables where the moisture is reduced to 10 per cent. The slime is then crushed with a roller on a cement floor and sampled in 22,000-lb. lots for shipment.

The slime is sampled in the following manner. The 22,000-lb. lot is shoveled into a conical pile, each shovelful being delivered upon the apex of the cone. To insure thorough mixing of the material this operation is repeated twice. The material is then split-shoveled four times; these operations reduce the sample to about 300 lb. From this point the material is quartered down with repeated crushing to about 130 oz., the laboratory sample.

The slime is then packed in 10-oz. canvas sacks holding 140 lb. each. Four of these sacks are packed in paper-lined wooden boxes in which shape the slime is shipped to an Eastern refinery for treatment. About 560,000 lb. of slime are produced and shipped per year.

## VIII. Power.

The power for driving the generators, furnishing the current for depositing, has been at all times furnished by water power developed at the Black Eagle Falls dam. The electrolytic plant is situated 2,000 ft. from the power house.

The original generator equipment consisted of three Thompson-Houston multipolar, shunt-wound, separately excited generators, 200 kw., each delivering 1,000 amperes at 200 volts. This equipment was supplemented shortly after being installed, by a third machine of the same type and size. These machines were rope driven from a counter shaft operated by water wheels.

The transmission line between the power house and tank house consisted of 36 ½-in. copper wires, 18 wires on a side. These wires were supported on 6 by 8 in. cross arms carrying heavy wooden pins and double

petticoat insulators. The cross arms were supported about 10 ft. above the ground by wooden poles.

This wire line was taken down a few years ago and recently it was decided to put the copper into merchantable form by charging it into the wire-bar furnaces with cathodes. A preliminary test to determine the quality of the copper gave the following results: Conductivity, annealed, 98.4 per cent.; As, 0.0072 per cent.; Ag, 5.15 oz. per ton.

This wire was purchased, on the market, in 1892 as high-conductivity

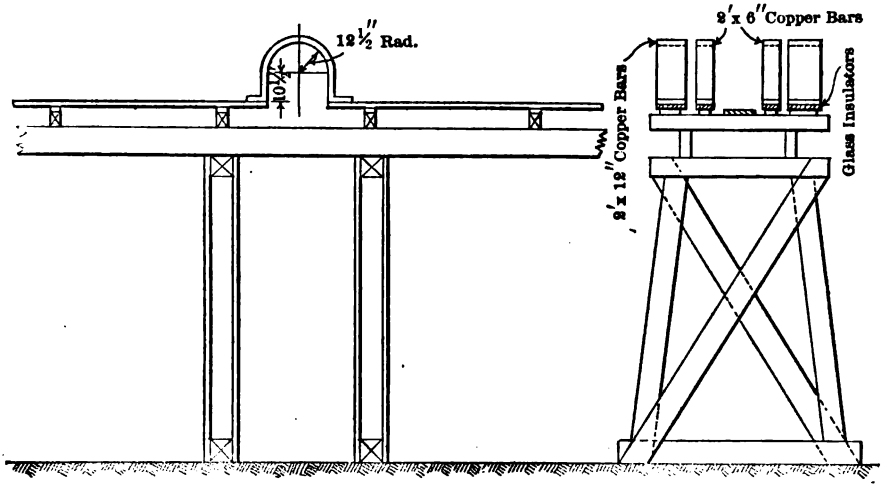



FIG. 9.—SECTION OF TRANSMISSION LINE, SHOWING EXPANSION JOINT.

copper. The consumer of the present day demands copper with a conductivity of 100 per cent. annealed, or over. As previously stated it was decided in 1896 to increase the production by increasing the current density, accordingly, in that year, the four 200-kw. generators were replaced by two Westinghouse, shunt-wound, engine-type, 810-kw. generators having a normal capacity of 4,500 amperes at 180 volts at a speed of 130 rev. per min. These machines were direct connected to a pair of 57-in. horizontal New American turbines operating under a normal head of 41 ft. It may be of interest to record that the average output of these generators to-day is 4,600 amperes each at 222 volts or 1,021 kw., equivalent to 26 per cent. overload.

These generators and water wheels were installed in a new stone power house built to accommodate them and constitutes our power plant of to-day.

To conduct the current from the new generators to the tank house a transmission line of rather unusual proportions was built. This line, as

originally built, consisted of 2 by 12 in. cast-copper slabs in parallel with the ½-in. wire line.

These bars, which were cast in 20-ft. lengths in open iron molds, were supported on a wood trestle (Fig. 9). The joints were made by overlapping the ends of the slabs 4 in. and fastening them together with 3.75-in. iron bolts, after the surfaces in contact were well cleansed; attempts to solder the joints failed because of the large body of copper and its high heat-conducting property.

Expansion and contraction was taken care of by expansion joints in the form of cast-copper arches. The copper slabs rested on 4 by 4 by 1 in. glass insulators.

Later the wire line was taken down and 4,377 ft. of 2 by 6 in. cast-copper conductors added so that the transmission line now consists of one 2 by 12 in. and one 2 by 6 in. conductor on each side arranged as shown in Fig. 9.

*Data on Transmission Line.*

| | |
|---|---|
| Effective length of line: | |
| 2 by 12 in. conductors, both sides | 4,895 ft. |
| 2 by 6 in. conductors, both sides | 4,221 ft. |
| Number of joints: | |
| 2 by 12 in. conductors | 270 |
| 2 by 6 in. conductors | 234 |
| Total | 504 |
| Overlap at junction points: | |
| 2 by 12 in. conductors | 180 sq. ft. |
| 2 by 6 in. conductors | 78 sq. ft. |
| Total | 258 sq. ft. |
| Weight of copper: | |
| 2 by 12 in. conductors | 466,992 lb. |
| 2 by 6 in. conductors | 201,342 lb. |
| Total | 668,334 lb. |
| Number of expansion joints: | |
| 2 by 12 in. conductors | 10 |
| 2 by 6 in. conductors | 10 |
| Total | 20 |
| When generator output is 9,295 amperes, 222.1 volts or 2,064.4 kw. | |
| 2 by 12 in. conductors carry | 6,070 amperes |
| 2 by 6 in. conductors carry | 3,225 amperes |
| Total | 9,295 amperes |

| | |
|---|---|
| Resistance of line at 19° C.: | |
| 2 by 12 in. conductors | 0.002190 ohms |
| 2 by 6 in. conductors | 0.004162 ohms |
| Combined resistance | 0.001431 ohms |
| Temperature rise at 9,295 amperes | 11° C. |
| Division of resistance: | |
| Due to copper | 75 per cent. |
| Due to joints | 25 per cent. |
| Power loss: | |
| Drop in line at 9,295 amperes | 13.3 volts |
| Power loss at 2,064.4 kw | 123.6 kw. |
| Power loss, percentage of total power developed | 5.99 per cent. |

The conductivity of the cast-copper bars in this line is approximately 91 per cent. or 9 per cent. less than annealed wire made from the same copper.

When the yearly interest on the investment in this line is compared with the value of the power lost in transmission for a similar period it would appear, from an economical standpoint, that the line contains

*Distribution of Energy.*

| | Volts | Kilowatts | Per Cent. of Total |
|---|---|---|---|
| 320 commercial refining tanks in series, 33.4 amperes per sq. ft | 190.4 | 1,769.8 | 85.73 |
| 44 starting-sheet tanks, 22 tanks in series, 16.7 amperes per sq. ft | 8.4 | 78.1 | 3.78 |
| 4 insoluble-anode tanks in series | 10.0 | 92.9 | 4.50 |
| Transmission line | 13.3 | 123.6 | 5.99 |
| Total | 222.1 | 2,064.4 | 100.0 |

*Analysis of Tank Resistance.*

320 commercial refining tanks, 22 anodes, 22 cathodes per tank, 90 per cent ampere efficiency, 2-day cathodes.

| | Volts per Tank | Volts 320 Tanks | Per Cent. of 190.4 Volts |
|---|---|---|---|
| Drop between anode busbar and anode | 0.044 | 14.08 | 7.40 |
| Drop between cathode busbar and cathode | 0.055 | 17.60 | 9.24 |
| Drop across electrolyte | 0.496 | 158.72 | 83.36 |
| Total | 0.595 | 190.40 | 100.00 |

far too much copper, but when the effect on the ultimate cost per ton to refine is taken into account it is apparent that the investment is fully justified.

Other power data follow: Generator output, 2-day cathodes, 33.4 amperes per square foot; head on water wheels, 44.1 ft.; amperes, 9,295; volts, 222.1; kilowatts, 2,064.4.

## IX. Operating Details.

The main tank room is divided into 10 divisions of 32 tanks each. Each division is taken care of by two tank men who are held responsible

Fig. 10.—Drawing Cathodes.

for these tanks. One inspector, locating and removing short circuits between anodes and cathodes, is employed for each two divisions. The inspector's work is done on night shift.

Sixteen tanks of 2-day cathodes, per division, or one-half of the tanks, are drawn daily, each tank containing 22 anodes and 22 cathodes. The day's starting sheets having been delivered to the different divisions by the night shift, the tank men begin drawing cathodes in the morning. Four cathodes are drawn at a time with a 1-ton Yale & Towne triplex chain block and multiple hook as shown in Fig. 10. The cathodes after

being replaced with starting sheets are deposited in the lead-lined wheel carriages shown in Fig. 11; each carriage holds 66 sheets. The solution dripping from the cathodes is held in these carriages to be later drawn off through an outlet in the bottom. When the carriage is loaded it is moved by hand to the scales, situated as shown in Fig. 3. After weighing the cathodes are lifted from the carriage by the traveling crane. The ends of the angle irons on which the cathode rods rest in the carriage are punched to receive the crane chain hooks by means of which the crane raises the load. After the cathodes are removed, the carriage is weighed

FIG. 11.—VIEW OF WORKING FLOOR.

to obtain the tare. The crane then raises the cathodes and loads them on to a 4-wheel truck for delivery to the furnaces.

After the cathodes are drawn the tank men inspect the anodes, marking such as are ready to be removed as scrap. The same carriages as were used for the cathodes are now spotted under the trolley track at the end of the tanks and the anode scrap is deposited in them. When all of the scrap has been removed in this manner the carriages are moved to the scales and weighed. The crane then lifts the scrap from the carriages, by means of the angle irons, and conveys it to a tank where the slime is washed from the surfaces with a stream of water. The scrap is then placed on trucks, by the crane, from which it is loaded into box cars for shipment to the converters.

After the scrap has been all drawn from the tanks and the carriages removed the tank men proceed to fill the vacancies, thus made in the tanks, with new anodes as shown in Figs. 11 and 12, one anode at a time being handled.

As but four cathodes are removed from one tank at a time it is not necessary to cut the tanks out of circuit while drawing copper.

As compared with the traveling-crane system of handling the anodes and cathodes in tank units, the hand chain-block system of handling, as

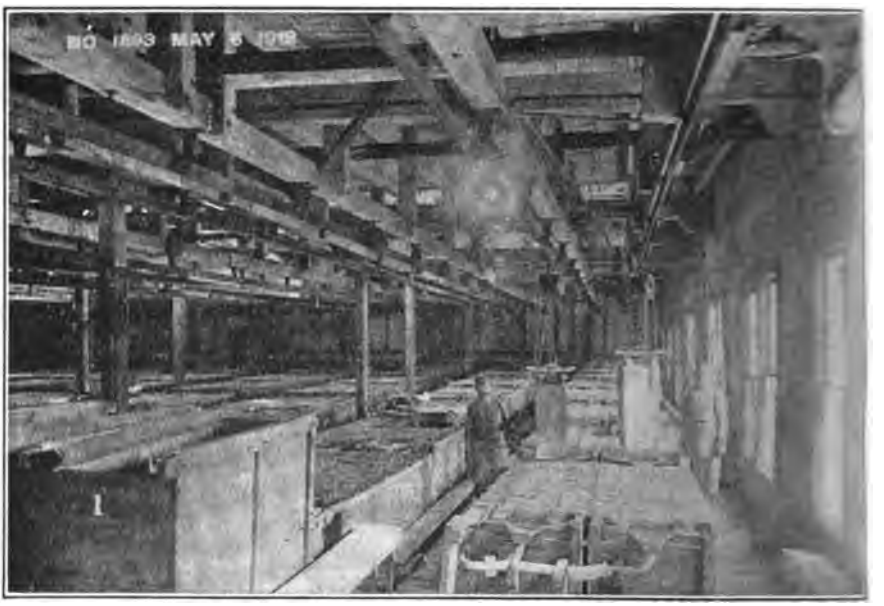


FIG. 12.—CHANGING ANODES.

employed here, admittedly required more labor but is not without its advantages, which are as follows:

First, the chain-block system permits the use of converter anodes.

Second, the chain-block system, with aisles between the refining tanks, permits of starting sheets being straightened and placed in the tanks in such a manner that subsequent removal and straightening is not necessary.

Third, respacing of anodes and cathodes, from time to time, as the anodes corrode, to reduce the resistance of the tank, is unnecessary as the tank men properly space each separate anode and starting sheet as placed in the tank.

Fourth, there is a lower percentage of scrap and no scrap to be returned to the tanks.

Cash prizes are awarded to the tank men on the divisions making the

best and second best records for the month. The division producing the highest tonnage of cathodes, after deducting the weight of the anode scrap, is considered as having the best record. This plan has been found very effective in stimulating rivalry between the different divisions and thereby improving the efficiency of operations.

Four insoluble-anode tanks to take care of the increase of copper in solution are provided. These tanks, as shown in Fig. 3, are situated in a separate room south of the pump house and, by means of a system of circulating pipes, can be run on any of the three electrolytes. It is not usually found necessary to operate more than two of these tanks at a time.

Four-day cathodes at 34 amperes per square foot are produced in these tanks. The product of these insoluble-anode tanks is of the same purity as the product of the soluble-anode tanks provided the rate of flow of the electrolyte is sufficiently high to insure ample copper in solution in the lower tanks. When the rate of flow is greatly reduced or altogether stopped the electrolyte soon becomes impoverished in copper and arsenic is deposited out of the solution with the copper. The circulation is maintained in these tanks at the rate of 10 gal. per minute.

The following is a summary of the average amount of copper, silver and gold locked up in the process while drawing 2-day cathodes and refining at the rate of 174,000 lb. copper per day.

*Metals Locked up in Process.*

| | Copper, Lb. | Silver, Oz. | Gold, Oz. |
|---|---|---|---|
| In anodes | 2,300,000 | 44,000 | 316 |
| In slime | 22,300 | 140,000 | 850 |
| In cathodes | 180,000 | ..... | ..... |
| In solutions | 95,000 | ..... | ..... |
| Totals | 2,597,300 | 184,000 | 1,166 |

To arrive at the weight of copper in the refining tanks, for inventory purposes on the first of the year, a spring balance is attached to the chain-block hook and the anodes and cathodes are raised clear of their supports and weighed in solution. A factor, previously determined, representing the difference in weight of the average electrode, weighed in solution in this manner, and its actual weight, is applied to the weights thus obtained. Weights obtained in this manner are found to check actual weights very closely.

The amount of slime in the bottom of the refining tanks at any time is calculated from the tank cleaning record, the number of days the slime

has been accumulating and the amount of slime that is liberated at the anode per tank per day per ampere being known.

Current readings are taken and recorded hourly from a 10,000-ampere Weston station ammeter the shunt of which is situated in the center of the tank house. At the power house on each generator-switchboard panel is a 5,000-ampere instrument of the same type. The averages of the hourly readings from these instruments are compared daily with the readings of the tank-house instrument to detect possible current leakage.

A system of pressure wires connects the refining tanks and the generators with a voltmeter switchboard situated in the office at the electrolytic plant. Hourly readings of the voltages at generators, at the tank house and at the different divisions and subdivisions of refining tanks are taken and recorded. In this manner abnormal conditions in any group of refining tanks or any change in resistance of the transmission line are promptly made known.

The daily report contains, on one sheet, the cathode production for the day and to date for the month, the number and weight of starting sheets used, the pounds of copper deposited per tank per day per ampere in the different sections, the percentage ampere efficiency of the total production, the amount and percentage of anode scrap for the day and to date for the month, the average amperes, volts and kilowatts for the day and for the age of the cathodes drawn, the average weight per cathode drawn, the pounds of anodes, cathodes and anode scrap on hand, the pay roll for the day and a digest of the electrolyte men's reports showing amount of solution treated in the purifying plant, character and speed of circulation of electrolyte and other minor data.

The present practice is to draw 3-day cathodes on Mondays and Tuesdays, 2-day cathodes on the four days following and none on Sundays, the generators running continuously during the seven days.

Under the head of 22 anodes and 22 cathodes per tank in Table VI, the results obtained are shown in detail.

The average daily production of refined copper is 174,000 lb.

## X. Furnace Refining.

The refining-furnace plant, to be operated in connection with the electrolytic refinery, began producing anodes from converter copper in October, 1892.

The original installation consisted of two coal-fired reverberatory furnaces. These furnaces were identical in design, having a hearth 15 ft. 6 in. by 10 ft. 6 in. with a firebox 4 by 4 ft. Each furnace was provided with a separate brick stack 65 ft. high, 28 by 28 in. inside. The capacity of these furnaces was 30,000 lb. of copper each.

The furnace building was 80 by 128 ft. steel frame covered with corrugated iron.

During 1893 and 1894 two additional furnaces of the same type and capacity were added. Two of the furnaces were employed in producing anodes from converter pig and anode scrap from the electrolytic plant and two were used to make wire bar, cake and ingot from the cathodes.

All of the furnaces were dipped by hand, 9-in. ladles being used. The copper was cast in iron molds, the electrolytic copper being dumped in boshes containing water. The anodes were removed from the molds, by means of hooks, and allowed to cool in the air.

Later these four furnaces were replaced by two larger furnaces of an estimated capacity of 50,000 lb. of copper each, subsequently increased to 100,000 lb. each. The hearths of these furnaces are 14 by 24 ft. and the fireboxes are 7 ft. long by 8 ft. wide. One of these furnaces is connected to one of the original brick stacks, described above, and the other is provided with a new brick stack 74 ft. high with a 34 by 34 in. flue.

These two furnaces with a combined capacity of 200,000 lb. per day now handle the cathode output of the electrolytic refinery, the anodes coming direct from the converters.

The product of the refining furnaces is dipped into copper molds, made at the furnaces. These molds are hung in trunnions over cast-iron boshes through which cold water is circulated. The mold is capsized, as soon as the metal is set, and the copper drops into the cooling water. Cooling in this manner prevents the formation of cupric oxide on the surface of the castings.

The copper is conveyed from the furnaces to the molds by means of large ladles suspended by chains from a trolley running on a ¾ by 4 in. track suspended over the boshes. Hand-forged Welsh ladles of the following sizes and capacities are used: 14 in., capacity 200 lb. of copper; 16 in., 250 lb. of copper, and 19 in., 350 lb. of copper. The size of the ladle used is governed by the weight of the casting to be made.

The principal forms into which the copper is cast are as follows: Ingot, ingot bar, cake, wire bar and a special form of round billet used in the manufacture of seamless copper tubing. The copper is shipped direct to the consumer. A 100,000-lb. charge is taken out of each furnace once in 24 hr.

The usual practice of rabbling and poling the copper, for the oxidation of impurities and bringing of the metal to the proper pitch for casting, is carried on. The rabbling is effected by introducing compressed air at a pressure of 16 lb. per square inch into the molten bath by means of two ¾-in. iron pipes, the end of the pipes being kept below the surface of the metal. About two hours is required for this operation. The re-

ducing action is obtained by forcing green pine poles into the bath and covering the surface of the metal with charcoal. About 28 poles, 8 in. at the butt and 4 in. at the top, and 35 bushels of charcoal are required per charge of 50 tons of copper.

The condition of the bath, as regards oxygen contents, is noted from time to time by the refiner as he examines the successive button samples taken from the bath in a small ladle. When the physical appearance of the button indicates that the "tough-pitch" stage has been reached the poles are removed and the dipping operation is begun.

As the dipping proceeds the refiner observes the set of the castings made and adds charcoal or logs of wood to maintain the oxygen at the proper point.

The skimming of the furnace takes place just before the rabbling or oxidizing is started. The slag obtained is equivalent in weight to about

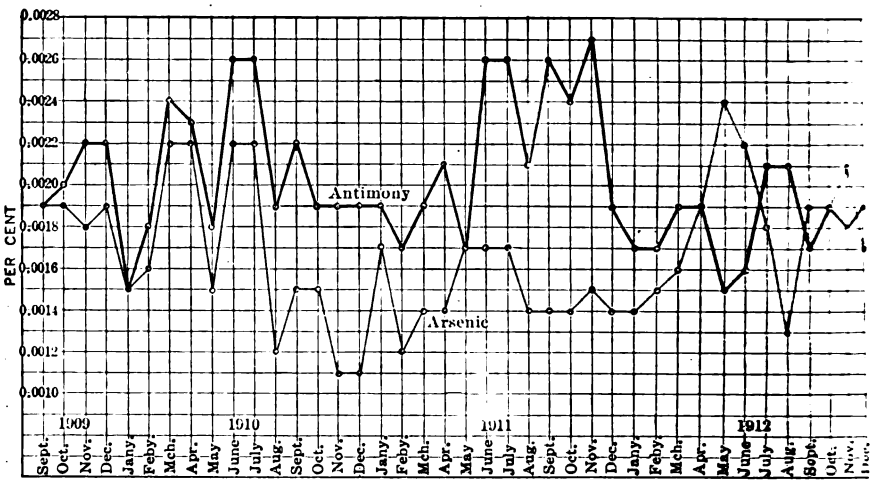



FIG. 13.—RELATION OF ARSENIC TO ANTIMONY IN WIRE BAR.

3.5 per cent. of the copper charged, varying with the amount of lime used. The following is the average analysis of slag produced at furnace No. 2 producing wire bar: Cu, 62.89; $SiO_2$, 19.1; FeO, 2.0; $Al_2O_3$, 1.8; CaO, 7.2; MgO, 1.4; S, 0.17; As, 0.02; Sb, 0.025 per cent.; Ag, 0.350 oz. per ton.

The fuel used in these furnaces is known as Lochray lump coal, a bituminous coal mined at Tracy, Mont., 15 miles from these works. The analysis is as follows: Moisture, total, 7.01; moisture, combined, 1.7; volatile matter, 24.7; fixed carbon, 48.0; ash, 18.5; sulphur, 3.5 per cent.; B.t.u. per pound of dry coal, 10,803.

It will be noted that this coal is high in sulphur. To protect the cop-

per as it melts from the sulphur dioxide gases, resulting from the combustion of the fuel, about 30 per cent. of the cathodes going to make up wire bar charges are dipped in milk of lime before charging. As the copper melts, the lime forms a protective covering for the metal, hindering in a large measure the absorption of sulphur by the molten copper. This whitewashing process accounts for the lime and magnesia in the furnace slag.

The average analysis of wire bar produced at these furnaces is shown in Table I, and the relation of arsenic to antimony in wire bar is shown in Fig. 13. The analysis of cake, ingot and other furnace products, in which high electrical conductivity is not essential, will contain 0.02 in 0.05 per cent. less copper, due to the increase in oxygen contents.

## XI. Physical Testing.

Samples for chemical analysis and physical tests are taken from each furnace charge as follows. A shot sample is made when the furnace is ready to dip. This sample is used for the As + Sb and silver determinations; the month's accumulation of silver buttons, from each furnace, is parted for gold.

Four test bars measuring 3.5 by 3.5 by 11 in. long are cast from each charge of wire bar dipped. The first bar is made when dipping begins, the second when the charge is one-third dipped, the third bar when the charge is two-thirds dipped and the last bar when the last round is being dipped. From each of these bars a billet 1 in. square and 8 in. long is sawed on a band saw; this billet is heated in a Hoskins electrical furnace and rolled down to a ¼-in. rod in an 8 by 13 in. three-high rolling mill. The rod, after being annealed in the electric furnace, is drawn to a No. 12 B. & S. gauge wire on a No. 1 bull block. The rolling mill and the bull block were furnished by the Waterbury Farrel Foundry & Machine Co. of Waterbury, Conn. In drawing the ¼-in. rod to a No. 12 gauge the wire passes successively through the following dies: B. & S. gauge, Nos. 2, 4, 6, 8, 10, and 12. As the wire is not annealed during the process of drawing the product is known commercially as hard-drawn wire.

The No. 12 finishing die is a Waterbury Die Co.'s diamond die which finishes the wire accurately at 0.0808 in. diameter. The preceding dies are ordinary steel dies.

In the physical laboratory these wires are tested for conductivity, tensile strength, elongation and torsion.

The first operation in the laboratory is to anneal the specimen to be tested for conductivity. This is done by suspending the sample between two copper blocks, with 7-ft. centers, and allowing an average current of 124 amperes to pass through the wire for a period of 40 sec. The

initial current is 130 amperes. As the temperature of the wire rises the current decreases until but 118 amperes are passing at the finish. The wire is then allowed to cool in the air. Plunging the wire into water immediately after shutting off the current results in the same conductivity as when cooled in air.

The annealing operation has the effect of increasing the conductivity of the copper about 2.7 per cent. While conductivity tests are made on both the hard-drawn and annealed wire the results on annealed wire are considered far more reliable than those obtained on the hard-drawn wire. This is due to the fact that wire may be drawn to different degrees of hardness by varying the speed at which the wire is drawn and the size and number of dies employed.

Assuming two wires be made from the same bar but drawn to such different degrees of hardness that their conductivities will vary as much as 1 per cent., if these wires are subsequently annealed their conductivities will be found to check very closely.

No. 12 B. & S. gauge wire is used for all physical tests at these works. Conductivity tests are made on both hard-drawn and annealed wire: the tests on hard-drawn wire being made more as a check on the annealed-wire results than for any other value they may have.

The mechanical tests are all made on hard-drawn wire. The physical-testing laboratory building, 15 by 52 ft., was erected and equipped in 1897 and since that date regular tests have been made on all wire-bar charges dipped and for 10 years past conductivity tests have been made on all charges of refined copper.

The first conductivity-testing apparatus installed was designed and furnished us in 1897 by Prof. W. L. Puffer of the Massachusetts Institute of Technology. This bridge required a wire sample 50 ft. long. Accurate results were obtained with the apparatus but its size made it rather unwieldy.

In 1899 the Puffer apparatus was replaced by a Willyoung conductivity bridge furnished by J. G. Biddle of Philadelphia. This bridge handled samples 30 in. long, the conductivity of the samples being calculated from the bridge readings. This instrument was continued in service until 1906 when it gave way to a Hoopes conductivity bridge furnished by the Leeds & Northrup Co. of Philadelphia. This is a most satisfactory instrument. This bridge tests samples 38 in. long and the percentage conductivity is read directly from the scale, no calculation being necessary.

Wire samples are sent from time to time to the Bureau of Standards at Washington. The wires are measured for conductivity by them and returned to us with their certificates of tests. These wires we retain as standards with which we check our conductivity bridge.

The mechanical tests are made on a Riehlé 10,000-lb. horizontal wire-

testing machine and a Riehlé torsional wire-testing machine both furnished by the Riehlé Bros. Testing Machine Co. of Philadelphia.

The first test bar made at the furnace, just as the dipping of the charge is begun, is immediately made into wire and tested for conductivity; about 40 min. is required after the bar is received to obtain the conductivity. With this knowledge the refiner is aided in the subsequent handling of the charge.

Wire is made from the three remaining bars on the following day and the averages of the tests made on the four bars are reported as representing the charge.

The following are the averages of the physical tests made on the wire-bar product at this plant during the year 1912.

| | |
|---|---|
| Conductivity, hard-drawn wire, per cent | 97.32 |
| Conductivity, annealed wire, per cent | 100.08 |
| Elongation in 5 ft., per cent | 1.00 |
| Tensile strength, lb. per sq. in | 64,400 |
| Torsion, twists in 6 in. | 37 |

NOTE: All mechanical tests made on No. 12 B. & S. gauge hard-drawn wire.

Oxygen in the refined copper is made the subject of a separate paper.[3]

[3] Notes on the Metallography of Refined Copper, by E. S. Bardwell, p. 1529, July *Bulletin*.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.
[SUBJECT TO REVISION.]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Butte meeting, Aug. 18 to 21, 1913, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1913, when Vol. XLVI. of the *Transactions* will go to press. Any discussion offered thereafter should preferably be in the form of a new paper for publication in Vol. XLVII. (with suitable cross references in both volumes).

# Determination of Gases in Smelter Flues; and Notes on the Determination of Dust Losses at the Washoe Reduction Works, Anaconda, Mont.

BY EDGAR M. DUNN, ANACONDA, MONT.

(Butte Meeting, August, 1913.)

## OUTLINE.

### PART I. DETERMINATION OF GASES IN SMELTER FLUES.

General considerations.
Regular methods for carbon monoxide, oxygen, nitrogen, moisture.
Sulphur oxides—Methods of Lunge and Hempel tried, and either modified or discarded, with reasons therefor.
Gravimetric method developed at Anaconda.
Description of apparatus used.
Brief notes on Richter and Hawley methods for the direct determination of sulphur trioxide.
Carbon dioxide, with complete notes on same.
Arsenic trioxide.
Table of percentages of flue gas constituents found at a Western smelter.
Some final notations as to the value of flue gas analysis, in connection with an intelligent log of furnace operations.

### PART II. NOTES ON THE DETERMINATION OF DUST LOSSES AT THE WASHOE REDUCTION WORKS, ANACONDA, MONT.

Discussion of problem, with description of flue system.
Various places and methods for determinations of velocity.
Instruments used.
Procedure in making determinations for velocity.
Soundings of cross-section for depths of dust.
Calculation of velocities and areas.
Daily volume of dust, and plant operations during tests.
Place for dust determinations.
Apparatus and methods of sampling.
Results obtained, with table of daily amount of dust losses.
Analysis of dusts for values, from various sampling stations.
Complete analysis of composite sample of stack emanation.
Last analysis totaled to show form in which present.
Discussion of recovery as a commercial proposition.
References on Pitot tube and calculations.

## PART I.—DETERMINATION OF GASES IN SMELTER FLUES.

IN 1907, upon arriving in Anaconda to take up work in the testing department of the Washoe Reduction Works, the following problem was met at the car step, so to speak: complete and exact analysis of gases, not only at the stack, but also in all departmental flues. Taken up to employ time not otherwise needed for regular testing department work, instructions were extremely liberal for a commercial plant, to the effect that accuracy should not be sacrificed to speed. In December, 1907, the work was temporarily transferred to the Great Falls Smelter, and some of the methods here given were there worked out. Differences in smelting operations at Great Falls and Anaconda, however, necessitated further investigations and the development of a new method for $SO_2$ determinations at the latter plant, where work was resumed in July, 1908, and completed about a year later.

At the time of inception of the work, data (not only on percentages, but even on methods) were found to be meager in the extreme, for one or the other of two reasons: either entire lack of experiment at a large proportion of smelters, or disinclination on the part of the managements to divulge data of any description pertaining to either results or methods.

In making determinations of gaseous constituents of flues, two methods are open to consideration: 1, the drawing out of a sample, and collection of the same as a whole, to be removed to the laboratory for later analysis; and 2, the absorption *in situ* on the flue of one certain constituent, disregarding others for the time being, or perhaps combining two or three in one determination. The first is generally the more rapid, but it is also the more inaccurate, method of analysis, even with special type absorption apparatus, being the more largely subject to error, both of reading, and also because of leakage during complicated manipulation in the laboratory. The amount used for analysis must also, because of limit of container, be small for the first method, and an average gas is very seldom obtained, necessitating for a fair average result a large number of determinations. Consequently, when we balance the time spent in a dozen determinations of short-time samples as against the time employed in making two or three long-time determinations of various constituents *in situ*, we are not so certain in our statement as to the relative speed of the two methods. Moreover, the method of analysis in the laboratory of 100 cc. of flue gas is satisfactory only when constituents thus determined are present in considerable quantity, results being simply approximate when determining such compounds as sulphur dioxide, sulphur trioxide, carbon monoxide and dioxide, and arsenic trioxide and water vapor, none of which are carried in *all* smelter flues in amounts in excess of (or even approaching) 1 per cent. by volume. Special condi-

tions at certain smelters may raise the percentage of water vapor to that amount; for instance, employment of wet fine concentrates for roaster feed, or use of briquettes while still wet in the blast, with poorly dried coarse concentrates. Generally speaking, however, five or six of the above-named determinations should be made independently of each other, or at least in special apparatus, *in situ* on the flue; but for carbon monoxide (never present with good practice in any smelter flue in considerable quantity), the difficulties in connection with the removal of other constituents in large quantities are so great as to indicate the rather unsatisfactory (from accuracy standpoint, because of trace quantity) method of its determination in the laboratory by absorption pipette of either ammoniacal or acid cuprous chloride, after removal of interfering gases in suitable reagents.

Determinations of oxygen, carbon monoxide, and nitrogen (residual gas after all other constituents are removed) were made by standard methods, and require no description here. All samples were collected over mercury and were analyzed by means of the Hempel mercury burette. Twelve 16-oz. bottles, filled with mercury and supported in a strong, specially designed, two-deck rack, were used for taking samples, and analyses of the last 10 (first one largely and second one slightly contaminated by air in aspirating tube) were used as a basis for results.

Nor was the method used for water vapor new in any respect. The gases were drawn through a hot filter of glass wool in the flue end of the aspirating tube, and into weighed bulbs containing sulphuric acid. Results obtained in this way are somewhat too high, but are more than approximately correct.

### *Sulphur Dioxide and Trioxide.*

Two standard methods were available for $SO_2$ and $SO_3$ at the time of starting our work. Briefly, Lunge's method consists in drawing gases through standardized alkali solution, with phenolphthalein indicator, determining both $SO_2$ and $SO_3$ as total acidity, as indicated by formation of the respective sulphide and sulphate reactions:

$$4\ NaOH + SO_2 + SO_3 = Na_2SO_3 + Na_2SO_4 + 2\ H_2O;$$

then a determination by the "Reich" method with standardized iodine solution of the amount of $SO_2$ present in the gas by the following reaction:

$$SO_2 + I_2 + 2\ H_2O = H_2SO_4 + 2\ HI;$$

the $SO_2$ factor from the iodine determination is then subtracted from the total as determined by the alkali, with a remainder of $SO_3$. We found Lunge's method (volumetric entirely) accurate only when the flue gases contained no substances of a nature contaminating to either

iodine or alkali value, which is a rare condition, and we have been very much interested by recently published conclusions of a similar nature. At practically all smelter flues some one or more of the adverse conditions are met. These conditions are the following: 1, presence of arsenic, selenium, or tellurium in the ores, which may vitiate any flue gas as far as the Reich test is concerned, if present in sufficient quantity, rendering results approximate rather than exact, by too high a figure for $SO_2$ from attack of iodine by oxides of these elements. 2, the presence of $H_2S$ in roaster flue gas, where a wet sulphide is slowly brought to roasting temperature. This last, if operative, will also cause too high an $SO_2$ figure, from the consequent attack of the iodine by the $H_2S$. We do not believe, however, that this is often the cause, though positive affirmation of same was at one time made to us by a prominent engineer. At least, we have never found, in frequently testing various conditions, hydrogen sulphide present in roaster gases (certainly never more than trace), and simply offer it here as a possible (not probable) explanation of the too high $SO_2$ figures so frequently found to be the actual result. 3, the fact that reverberatory flue gases (coal fired) contain variable amounts of hydrocarbons due to imperfection of combustion at certain stages of their operation; a fact hitherto unrecognized as to the action on iodine, causing by the mutual reaction too high a figure for $SO_2$ content of gas. We have several times obtained appreciable quantities of the red copper acetylene compound by passing reverberatory gases through ammoniacal cuprous chloride. The presence of acetylene points, of course, to other hydrocarbons of a similar character. This hydrocarbon factor seems to be absent in flues from gas-fired reverberatories, probably due to more perfect combustion control. As to oil-fired reverberatories, we have no data.

All of the above-mentioned factors tend to raise the $SO_2$ content of the gas in the Reich part of the Lunge method, and consequently, even if the total acidity (as determined by alkali) remains correct, yet the figures finally obtained for $SO_3$ (by subtracting too high $SO_2$) are correspondingly too low. Suppose, however, that the alkali titration is also open to errors that cause total acidity ($SO_2 + SO_3$) to be low. Of course, the method as outlined by Lunge becomes then not even approximate, the $SO_3$ figure being negative. To illustrate, under perfect conditions the following little arithmetical equation is true:

$$(SO_2 + SO_3) \text{ minus } SO_2 \text{ equals } SO_3.$$

In practice, however, the following is more often the case:
($SO_2 + SO_3$—too low) minus ($SO_2$—too high) equals *less than nothing*.

In an article that appeared the early part of this year (1913), in connection with a direct method for the determination of $SO_3$ by absorption, E. Richter (abstracted in the *Journal of the Society of Chemical Industry*,

Mar. 31, 1913) recognizes the failings of the Lunge-Reich method, the abstract (we have not read the original article) commencing as follows: "The determination of sulphur dioxide and trioxide in furnace gases by Lunge's and Reich's method does not give satisfactory results; either the reactions with iodine and potassium hydroxide do not take place in the theoretical proportions, or else substances are present which combine with a higher proportion of iodine than [does] sulphur dioxide; at any rate the results for trioxide often work out as a minus quantity." Richter's alternative is a method for determination of $SO_3$ direct, by absorption in a condensing tube packed in a freezing mixture, a method capable, we believe, of giving good results. A similar abstract of Mr. Richter's paper appeared in the *Engineering and Mining Journal* of May 10, 1913.

The last-named magazine (vol. xciv, pp. 987 and 988, Nov. 23, 1912) also gives an excellent article on sulphur dioxide and trioxide determination, by F. G. Hawley, chief chemist at Cananea, Mexico. Mr. Hawley emphasizes the cause of error in the alkali titration for total $SO_2 + SO_3$; though our own experience differs somewhat from his. We quote from him as follows: "An objection to this method is that it cannot be used on furnace gases because of the action of $CO_2$ on the phenolphthalein. Even in roaster gases, for which it is recommended, there is likely to be some $CO_2$ formed by the oxidation of small amounts of organic matter usually present in ores, as well as from small amounts of carbonates which may evolve $CO_2$ during roasting. Sulphates of the heavy metals are always present in the fume and will react acid to phenolphthalein, and thus affect the result."

Now we never tried the Lunge-Reich method anywhere else than at Anaconda, and Mr. Hawley may have ores of a character somewhat different from ours. The alkali titration works fairly well on our roaster and converter gases, the figures obtained by titration checking closely with gravimetric amounts obtained by later oxidation of sulphites to sulphates and precipitation of all sulphates with $BaCl_2$. Of course, in all of our work we eliminate all dust and practically all fume by slow aspiration and a hot (flue temperature) glass-wool filter in the flue end of our aspirating tube. But we feed to our roasters large amounts of carbonate in the shape of fine lime rock (later utilized in slag formation in the reverberatories and fed to the MacDougalls to insure better mixing), and yet the $CO_2$ evolution of the gas is but slightly greater than atmospheric amount, the critical temperature of calcium carbonate not being attained in roasting furnaces. Should ores be employed containing carbonates with lower critical temperature than that of calcium carbonate, the method would be worthless for roaster as well as for other flues using carbonaceous fuel. When $CO_2$ is present, there are three anhydrides

attacking the alkali instead of the two ($SO_2$ and $SO_3$), the $CO_2$ also reacting. The salt formed by the last, however, is alkaline, and remains inert as to acid action on phenolphthalein until the last stages of neutralization, when the original alkaline hydroxide is all gone with formation of carbonate, sulphite, and sulphate, the first named, however, with an alkaline reaction toward the indicator. Further passage of $CO_2$, $SO_2$, and $SO_3$ causes an attack of the carbonate by the two latter, but before they have finally changed the carbonate to sulphite and sulphate (which would be a true titration of the original alkali to the two salts formed by $SO_2$ and $SO_3$), the accumulation of $CO_2$ in the solution (both from flue absorption and from liberation from carbonate in the solution itself) swings the phenolphthalein reaction to acid, indicating an end point for $SO_2$ and $SO_3$ on NaOH before it has been reached, and causing *low* results for total $SO_2$ and $SO_3$ by this volumetric method. We early discarded the Lunge-Reich process.

The second method at our disposal in 1908 was the variation given by Hempel and mentioned by Mr. Hawley in his article. Briefly, it is a combination of volumetric and gravimetric methods, the gases ($CO_2$, $SO_2$, and $SO_3$) being passed into standardized iodine as in the Reich test and the $SO_2$ content being determined by the iodine factor, and this $SO_2$ figure being subtracted in terms of $SO_3$ from the total $SO_3$ figure (caused by both $SO_2$ and $SO_3$) later determined by gravimetric precipitation in acidified solution with $BaCl_2$. This method is open to all the objections previously mentioned in connection with the Reich test, the practical result being: $SO_2$ plus $SO_3$ (gravimetric and accurate) minus $SO_2$ (volumetric and too high) equals too low $SO_3$ (occasionally *nil*). $CO_2$, however, does not interfere or affect results in any way. We used this method in specially designed apparatus of our own (which will be later described) but with varying results, and finally evolved a method entirely gravimetric in character.

This last method, adopted by us after thorough experimenting, employs simultaneous determinations with two sets of apparatus, the determination for total $SO_2$ and $SO_3$ being made by passing gases through iodine solution, to convert $SO_2$ to $SO_3$ (at the same time collecting all original $SO_3$ unchanged). A gravimetric determination with $BaCl_2$ gives total $SO_3 + SO_2$, the latter in terms of $SO_3$. The original $SO_3$ has been simultaneously determined on the same gas by passing the gases through the second set of apparatus, the solution for absorption in this second case being weak $BaCl_2$ and from 10 to 15 per cent. of HCl. Mr. Hawley mentions this method in his article, and we suspect our original notes (which were furnished in 1909 and 1910 to a large number of Western smelters) caused him to look further, because of incompleteness of data therein, and finally evolve his scheme of filtering out the $SO_3$ by means

of a moistened filter, with later titration of contents of filter, which is a variation of the dust method abstracted in the *Journal of the Society of Chemical Industry*, vol. xxii, p. 1016.

We find the following opinion on the direct determination of $SO_3$ by means of $BaCl_2$ in the presence of $SO_2$ in Lunge's *Sulphuric Acid and Alkali*, vol. i, p. 317: "It was found that sulphur trioxide cannot, as Scheurer Kestner supposed, be absorbed and estimated by $BaCl_2$, because even chemically pure $SO_2$ with $BaCl_2$ in the presence of oxygen or atmospheric air, at once gives a precipitate of $BaSO_4$." We checked Lunge's work, but found the greater part of his precipitate to be, not $BaSO_4$, but $BaSO_3$, and soluble in HCl. As $BaSO_3$ is theoretically impossible in the presence of HCl, we made several experiments with $SO_2$ generated by means of dilute HCl and $NaHSO_3$, passing same with air through acidulated $BaCl_2$. We found repeatedly: 1, no $BaSO_3$ formed in any of a series of four flasks; 2, faint cloudiness in first flask, which was trace of $BaSO_4$; 3, relatively large to small amounts of $SO_2$ dissolved in all four flasks, which were later boiled out entirely without precipitation of any $BaSO_4$ from the last three flasks, and without changing visible amount of precipitate in first flask; 4, the amount of $BaSO_4$ in first flask after boiling thoroughly to expel $SO_2$ and to collect $BaSO_4$ for filtering was (maximum) 0.0065 g. $BaSO_4$, or somewhat less, for eight runs, using in all cases over 100 times as much $SO_2$ as we aspirate in a regular flue determination. The facts that all four flasks contained $SO_2$ dissolved therein, but none of the last three were precipitated by immediate boiling out of same; while the first flask was not precipitated by boiling, but only by passage of original $SO_2$ + air; force the conclusion that the trace of $BaSO_4$ found in the first flask was caused, not by $SO_2$, but by $SO_3$, formed between the generator and the first flask by admixture with air, *before* entrance to first flask.

Mr. Hawley objects to the boiling out of the $SO_2$, because of oxidation to $SO_3$, and consequent high results for the latter, and he ascribes oxidation found by him to atmospheric oxygen. With substitution of passage of $CO_2$ for removal of dissolved $SO_2$ (rather than a quick boiling out of same), he characterizes the method as accurate, though tedious, because of generation and passage of the $CO_2$. We found the method (without the tedious $CO_2$ passage and with a simple boiling) consistently correct; and at another of the Western smelters where *no* $SO_3$ was present in their stack gases, the results were persistently *nil*, though the stack gases did carry a much larger amount of $SO_2$ than is present at Anaconda. However, if Mr. Hawley boiled out the contained $SO_2$ over a free flame, or in direct sunlight, his conclusions as to high results are in agreement with our own results; for, while in position on the flue or while boiling off the $SO_2$ in the laboratory, the apparatus must be carefully protected

from any other than dull, diffused light, as we later found that the action of direct sunlight, or the light of a free flame at any point in the process between start of aspiration and final complete breaking up and boiling off of contained $H_2SO_3$, was destruction to the retarding action of HCl in the solution. In spite of the HCl, sunlight or free flame light acts as a catalyzer, causing quick and considerable oxidation of $H_2SO_3$ to $H_2SO_4$, with consequent high results for $SO_3$ content of flue gas by this method. Of course, boiling off of the $SO_2$ should be accomplished immediately after aspiration of gas, as standing between aspiration and release of contained $SO_2$ also causes results to run somewhat high, even in diffused light.

*Apparatus.*—The apparatus employed by us was as follows:

Set I of the apparatus, Fig. 1, consists of a wooden box 36 in. long by 9 in. wide. One end is left open for a distance of 9 in., and this compartment is covered with a board 9 by 9.5 in., which contains four slots 1 in. wide for holding the necks of the 4-oz. flasks firmly in place. The rest of the box is sided up with boards 4.75 in. high, and is to contain the first aspirator bottle, and such pieces of apparatus as may be carried conveniently therein. The end pieces of the box are so cut down near the top (12 in. high) as to make handles for carrying the apparatus.

Four 4-oz. flasks are used in making the analysis. These should be supplied with *tightly fitting* two-hole rubber corks.

Two large glass bottles (about 3 liters capacity) are used as aspirating apparatus, for the double purpose of drawing the gas from the flue and also measuring the amount of gas sample taken. The first bottle should be fitted with a three-hole rubber cork; the second requires a two-hole cork. Rubber tubing 10 or 12 ft. long may be used for connecting the two aspirating bottles. The first of the 4-oz. flasks, *B*, is connected with the flue by a one-piece bent tube, *A*, the long arm of which is about 3 ft. long, and the short arm about 9 in. long, which allows the long arm to be inserted in the flue about 2 ft. The flue end contains a loosely packed filter of glass wool, for stoppage of dust.

The four flasks are connected in series; *C*, *D*, and *E* being connected by glass tubes bent in two right-angle bends, the longer arm going nearly to the bottom of each flask (the inlet tube), and its end being nearly closed for slow aspirating purposes.

The shorter arm (outlet tube) is of such a length that, when the flask into which it goes contains 150 cc. of solution, the end of the tube (blown out bulb shaped, and containing one or more fine holes) will be at such a distance above the solution that, when running gas, the beads contained in this short arm will be kept well moistened with solution. The short arm should *not* dip in the solution, but its height above surface

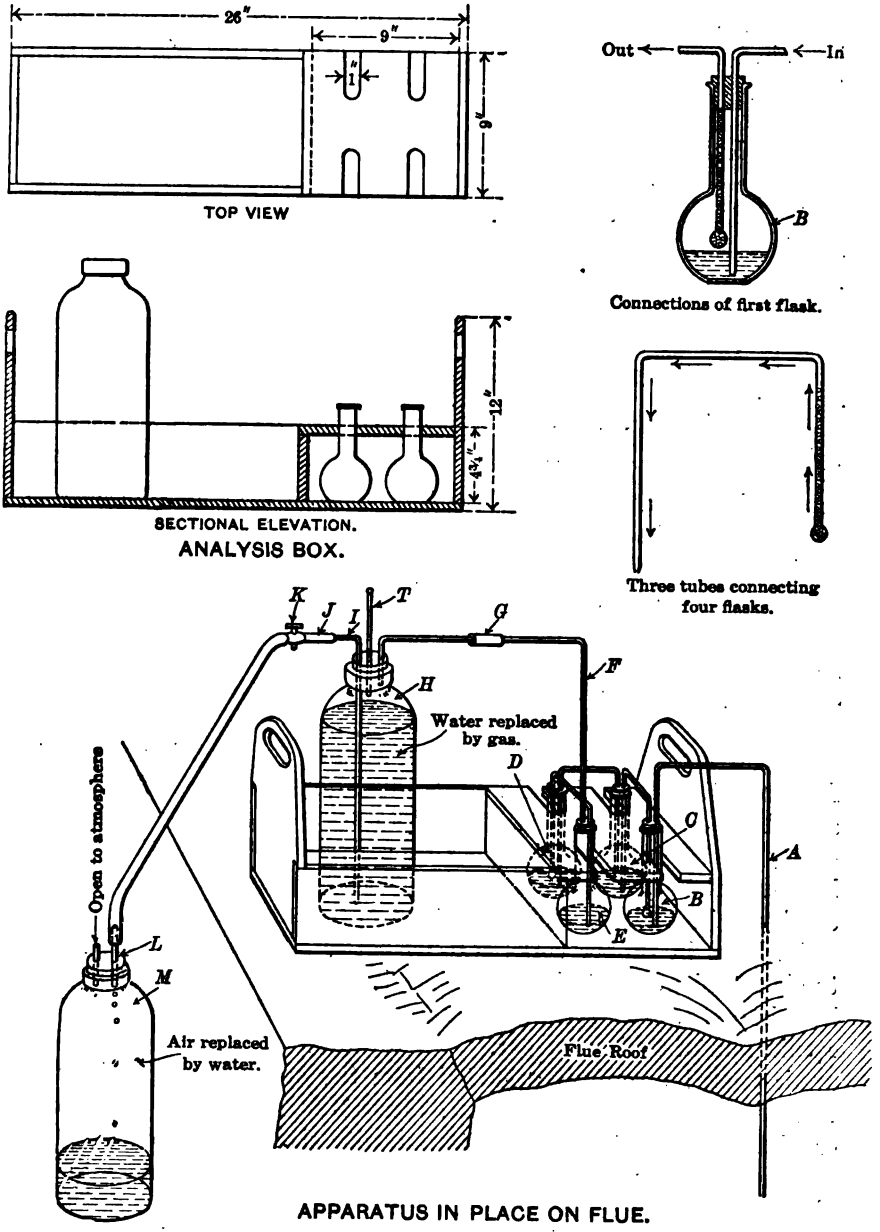



FIG. 1.—APPARATUS FOR DETERMINATION OF SULPHUR OXIDES.

of same will be governed by the speed of aspirating—the faster the aspiration, the higher the short arm, and *vice versa*.

The second hole of the fourth flask, *E*, is connected with one of the three holes in the cork of the first aspirator bottle by means of a two-arm glass tube, *F*, which is connected at about the middle by a heavy piece of rubber gas tubing, *G*, which is well wired to the glass to insure against possible leakage of gas. Both ends of glass tube *F* pass barely through the respective corks of the fourth 4-oz. flask and of the first aspirator bottle. The rubber connection is not necessary (*F* being one piece instead of two pieces), but should a one-piece tube be used, chances of breakage from over-stiffness of apparatus are much increased.

In the second hole of the cork of the first aspirator bottle, *H*, a thermometer, *T* (preferably Fahrenheit for closer reading), should be inserted. By this means the temperature of the gases aspirated may be ascertained.

Through the third hole of the cork in *H* a bent glass tube, *I*, passes to the bottom of the bottle, and is bent at its upper end at a right angle (or more) to connect with a rubber tube, *J*, 10 or 12 ft. long, through which the water runs from *H* to the second aspirator bottle, *M*, the top of which, when in action, should be some little distance below the bottom of the first aspirator bottle. This rubber tube is provided with a stop cock, *K*, for regulating the speed, and connects at its lower end with a piece of glass tubing, *L*, which penetrates one hole of the two-hole cork in the second aspirator bottle *M*, the other hole of which is left open. If preferred, *M* need not be corked, but left open.

Several precautions in taking the sample are of primal importance. All connections from *A* to *J must be* absolutely gas tight. For this reason all rubber connections between flasks are eliminated, also the one commonly used about the middle of tube *A*. Such elimination, of course, increases rigidity of apparatus, and consequently greater care must be employed in handling the same, to guard against breakage.

All holes in rubber corks (except bottle *M*) should be tightly fitted to glass tubing, and to insure loss against leakage all such fittings should be reinforced, after setting apparatus in place, by a liberal and careful application of rubber cement.

Air must be carefully excluded from the aspirating tube *A*. To this end, the tube should be inserted at least 2 ft. in the flue, and should be carefully packed at entrance into flue with slag wool or waste.

Before passing from methods for the oxides of sulphur to carbon dioxide, the processes of Messrs. Richter and Hawley for $SO_2$ determination must be considered. Either the absorption after cooling (Richter) or the filtration through a moistened filter (Hawley), should give good results. Both employ the passage of atmospheric oxygen for remov-

of $SO_2$ from the sponge which has just absorbed total $SO_3$ and more or less $SO_2$, but in one (Hawley) the presence of contaminating dust can have no effect, because of titration rather than gravimetric determination. The same variation might easily be applied to Richter's method. With regard to these sulphur gases, two former ideas of a mistaken character are now exploded: 1, that $SO_3$ (with its ordinary tremendous affinity for water) is easy of absorption, for we now know that it is the hardest gas of any encountered in smelter analysis to stop in water solution, probably because of the minuteness of its quantity, the $SO_3$ in each bubble of gas being surrounded by larger overcoats of more or less insoluble gases; and 2, that the oxidation of $H_2SO_3$ to $H_2SO_4$ is a rapid process, for we have learned that the reverse is true, and that even this slow process can be still further hindered by exclusion of catalyzers and introduction of retarding agents. The accuracy of both Richter's and Hawley's methods is dependent on this latter fact. The $H_2SO_3$ is driven out by passage of air, which requires considerable time, and yet no oxidation to $H_2SO_4$ occurs.

### *Carbon Dioxide.*

For the determination of $CO_2$ in flue gases we followed Pettenkofer's method for determination of $CO_2$ in air, adapting the process to our own apparatus, and changing the strength of solutions to meet requirements of gases in the various flues.

With two sets of apparatus, the three constituents $SO_2$, $CO_2$, and $SO_3$ can be determined simultaneously, using the first set for total $SO_2+SO_3$ (first two bottles) and for $CO_2$ (third and fourth bottles) and the second set for direct $SO_3$.

The method for $CO_2$ as adapted by us to flue conditions is here given more fully perhaps than is necessary, but with a view to illustration of our idea of tabulation of process notes.

*Apparatus.*—The apparatus used for the determination of $CO_2$ is the same as that employed for $SO_2$ and $SO_3$ (Set I). In addition, a small 4-oz. flask is fitted with a two-hole cork, through one hole of which passes a glass tube just through the cork. This tube is for connecting the flask with the flue tube, *A*, of Set I; the other hole of the cork is fitted with a tube which extends from the bottom of the flask up through the cork. Through this second tube the air when operating passes into the flask to the bottom and bubbles up through the flask full of strong NaOH solution. For purposes of reference call this flask *X*.

*Solutions Required for Analysis.*—The solution of NaOH for use in flask *X* need not be of exact strength, as it is simply for the purpose of removing $CO_2$ from air passed through the apparatus. Dissolve about 100 g. of NaOH in about 500 cc. of water.

*Iodine Solution for Removing $SO_2$ and $SO_3$ from the Gases Aspirated.*—Dissolve about 10 g. of iodine and 15 g. of potassium iodide in from 100 to 200 cc. of distilled water. Shake until iodine is entirely dissolved and dilute to 1 liter. If desired, N/10 iodine may be used.

*Barium Hydroxide Solution for Absorbing the $CO_2$ in the Gases Aspirated.*—Dissolve about 14.8 g. of $Ba(OH)_2$ in 2 liters of distilled water, and filter into the stock solution bottle. Mix thoroughly and cork bottle tightly with a two-hole rubber stopper. Through the first hole of this cork passes a long double bent glass tube to the bottom of the bottle inside; the other end extending several inches below the bottom on the outside. This siphon tube is clamped with a stopcock, and when not running out the solution, the second hole of the rubber cork is also closed with a tightly fitting glass rod. This $Ba(OH)_2$ solution should be standardized against the oxalic acid solution so that its value is known in precipitating power against both the standard oxalic acid solution and carbon dioxide. When running $Ba(OH)_2$ out of stock bottle, inflowing air should be passed through flask *X*.

*Standard Oxalic Acid Solution.*—Place 100 cc. of boiling distilled water in a No. 2 beaker, and add oxalic acid crystals until solution is saturated hot. Cool the solution and dry the resulting oxalic acid crystals by repeatedly pressing between folds of filter paper, until the crystals are thoroughly dry, and pour like sand from one sheet of paper to another. Weigh out exactly 5.6293 g. of this recrystallized oxalic acid, and dilute to 1 liter with distilled water, which is preferable freshly redistilled. It is best to double the amounts of $C_2H_2O_4$ and $H_2O$, making 2 liters of solution. After solution is attained place in a dried glass-stoppered bottle. This solution is of such a strength that each cubic centimeter is equal to 1 cc. of carbon dioxide, at standard conditions, in its neutralizing action on barium hydroxide; or, in other words, 1 cc. $C_2H_2O_4 = 1$ cc. $CO_2$. Its strength should be checked against standard $Na_2CO_3$ solution.

*Phenolphthalein Indicator.*—Weigh out 1 g. of phenolphthalein and place in a glass-stoppered bottle of about 250 cc. capacity. Add 210 cc. of strong ethyl alcohol and shake until dissolved.

*Standardization of $Ba(OH)_2$ Solution.*—Measure out 100 cc. of the $Ba(OH)_2$ solution from a burette, add 5 drops of the phenolphthalein, and titrate with the standard oxalic acid to decolorize the solution. The $Ba(OH)_2$ will be found to be about half the strength of the oxalic acid, or the 100 cc. of $Ba(OH)_2$ will require about 50 cc. of $C_2H_2O_4$. The $Ba(OH)_2$ crystals used are never of definite strength (having been more or less acted on by the $CO_2$ in the air). In the determinations made the 100 cc. of $Ba(OH)_2$ was found to be equal (phenolphthalein decolorized) to 48.75 cc. of $C_2H_2O_4$. Therefore, 1 cc. of $Ba(OH)_2$ equaled 0.4875 cc. of $C_2H_2O_4$ (of $CO_2$).

*Method of Taking Sample.*—Fill flask *X* with NaOH solution, and connect with the flue tube *A* (Set I) by means of a small piece of rubber tubing. Tube *A* is connected with the four absorption flasks (Set I), and the last of these flasks is connected by means of a long rubber tube with the air pump in the laboratory.

Draw air through the apparatus for 0.5 hr., by which means the air in the absorption flasks is replaced by air containing no carbon dioxide (which is removed by passage through NaOH solution in flask *X*). Disconnect and fill the first of the four absorption flasks with iodine solution to the 150-cc. mark. In the second absorption flask place about 130 cc. of redistilled water. Into the third and fourth flasks run in from a graduated burette, respectively, 150 cc. and 100 cc. of the $Ba(OH)_2$ solution. Do not dilute the $Ba(OH)_2$ in the fourth flask with ordinary distilled water (which contains trace of $CO_2$); either leave the volume of solution therein at 100 cc. or else dilute to the 150-cc. mark with redistilled water *free from* $CO_2$. Cork the absorption flasks tightly with the corks and bead tubes used in flue aspirating, and remove to the flue. Set up the apparatus (Set I) on the flue, just as in the determination of the sulphur gases, and aspirate for about 90 to 100 min. at the rate of about 2,000 cc. of water drawn from the first into the second aspirator bottle in the total elapsed time of aspiration. A faster rate of aspiration must be carefully avoided to prevent the passing through the first two absorption flasks of any $SO_2$ or $SO_3$, which would, of course, raise the percentage of $CO_2$.

The position of the outlet tubes from the first three absorption flasks must also be carefully watched. The outlet tube from the first flask (containing iodine) should be so set that the beads contained in the tube are at all times impregnated with the iodine to insure stoppage of the sulphur gases in the first flask. To check this fact, a *trace* of the iodine from the first flask should be allowed to run over into the second flask sufficient to impart a yellow tinge to the water contained therein. If at any time the aspiration should become rapid enough to decolorize this yellow *tinge*, it shows that $SO_2$ is passing through the iodine into the $Ba(OH)_2$, and the determination is worthless. The outlet tube from the second flask should be slightly higher in position, so that the beads are simply moistened with the yellow-tinged water, and yet so that *none* of the solution finds its way into the third flask. Should this occur, the determination is also worthless, for two reasons: the percentage of $CO_2$ will be slightly raised by the attacking of the $Ba(OH)_2$ in the third flask by the solution that runs over from the second flask, and the end reaction used in later titration will be vitiated. The outlet tube from the third flask should be so set that the beads are constantly wet with the $Ba(OH)_2$ solution, a small amount of which should be allowed to pass into the fourth

flask. This precaution insures absorption of $CO_2$ in the third flask, largely, but never entirely, the fourth flask being here necessary. With large amounts of $CO_2$ a fifth flask must sometimes be used.

During the aspiration the formation of insoluble barium carbonate ($BaCO_3$) in the third flask and outlet tube tends to clog the beads and the stopcock which controls the aspiration will require constant watching to obtain a steady flow of gas.

At the end of the aspiration period close the stopcock and measure the water in the second aspirator bottle. This will require about 5 min., during which time the absorption apparatus and first aspirator should be left on the flue. Read the temperature of the gas in the first aspirator bottle and remove box to the laboratory without disconnecting flasks and first aspirator. Connect flue tube, *A*, with NaOH bottle, *X*, and aspirate slowly by means of the air pump for 10 min. to insure absorption of the sulphur gases standing in the outlet tube and over the iodine in the first absorption flask. Now raise the outlet tubes slightly in the first two absorption flasks and aspirate at about twice the former speed for 0.5 hr. to draw through the third flask any $CO_2$ which may have been held in physical solution in the first two absorption flasks. Cease aspirating and raise the corks in the absorption flasks, taking out the corks and tubes, and closing the third and fourth flasks with tight rubber stoppers, and proceed at once with the analysis.

*Analysis of Samples.*—To one of the third and fourth absorption flasks add 5 drops of the phenolphthalien indicator and titrate with the standard solution of oxalic acid ($C_2H_2O_4$) until the red solution is fading in one of the flasks. Cork and set aside, and titrate the solution in the other flask to the same point. Now just decolorize the solutions in both flasks and wash out the bead tube connections with the titrated solution. If a pink color develops in the flasks, again decolorize with the oxalic acid. Take the reading of the oxalic acid burette, and proceed to calculate results. If desired, the indicator may be added before flue aspiration.

*Notes.*—The solution ordinarily used in gas absorption pipettes for the removal of $CO_2$ is NaOH. This cannot be used in the flue determinations, for the reason that for absolute accuracy the $CO_2$ *must be removed from the solution as fast as absorbed,* so as to admit of the later titration of the excess of the alkaline solution with the standard acid.

If NaOH is employed, the aspiration of $CO_2$ results in formation of sodium carbonate, which is soluble and remains in solutions to be attacked later by the standard acid used in the titrating; while with $Ba(OH)_2$ as an absorption solution the $CO_2$ precipitates as barium carbonate, which is insoluble in all reagents with which it comes in contact ($H_2O$, $Ba(OH)_2$ and dilute $C_2H_2O_4$) and the titration with dilute $C_2H_2O_4$ is in no way affected by its presence. If NaOH is to be employed the later titration

must be omitted; the indicator must be added to the alkali before aspiration; and the red solution must be decolorized in position on the flue by the aspiration of $CO_2$. This is a more rapid, but by no means a satisfactory, determination, for two reasons: 1, the end reaction between any alkali and $CO_2$ is not as quick and clean cut as that between an alkali and oxalic acid (especially true when phenolphthalein is used for an indicator; but true in some degree of any indicator); 2, this method of determination *loses* the $CO_2$ which is physically absorbed in the flasks of iodine and water (which are placed between the alkali and the flue for the purpose of absorbing the sulphur gases). This latter error is alone sufficient to condemn the determination, as close checking is impossible. We found 20 per cent. of total $CO_2$ held in first two flasks. Experience has proved that another very wide error may be introduced into an otherwise carefully performed determination, by the clogging of the apparatus by the $BaCO_3$ formed in the outlet bead tube from the third absorption flask as well as that in the third flask itself. In the previous methods employed for the determination of the sulphur gases, the products of absorption were soluble, and consequently evinced no desire to interfere with aspiration. This clogging with the insoluble $BaCO_3$ causes an increased suction to be required for drawing the gas through in the later stages of aspiration, and at the end of the aspiration it may be found that the gas in the first aspirator bottle is therefore under diminished pressure, and the water in the second aspirator does not therefore represent the true volume of gas aspirated from the flue.

In one of the preliminary tests of the method outlined about 1,500 cc. of water was drawn through in slightly over an hour, but in the latter part of that time the aspiration entirely ceased despite the fact that the stopcock on the tube between the two aspirator bottles was finally opened wide. The stopcock was then closed, the apparatus removed to the laboratory for analysis, and the back pressure was noticed to be large (by the inrush of air into the first aspirator when the stopcock was removed). This, of course, caused a figure much too large for the volume of gas aspirated, as shown by the fact of a 12 per cent. drop in volume percentage of $CO_3$ when the final results were obtained. The remedy, where the aspiration does not run smoothly and stoppage occurs, unless very near the end of the aspiration, is to throw out the determination, wash out the apparatus and begin over again. If desired, the back pressure can be measured, the error calculated, and the determination finished.

To prevent stoppage of apparatus, the tubes connecting the absorption flasks should be carefully washed out (between determinations) two or three times with dilute HCl. This dissolves the precipitated $BaCO_3$, and the excess of HCl must then be carefully washed out with distilled

water, or results for $CO_2$ will be too high (the HCl acting on the $Ba(OH)_2$ during aspiration). Eight to a dozen washings are required to eliminate the HCl.

*Arsenic.*

The analysis for arsenic vapor in smelter flues offers no great difficulty. This constituent is present in such small amounts that rapid aspiration, to obtain large volume, and analyzable amounts, is the first requisite. Consequently large carboys are employed for aspirating bottles. Experiment proved the impossibility of escape of $As_2O_3$ through the absorbing solution (water alone is excellent) even under a speed of 25 liters per hour, for the reason that the vapor readily condenses to a solid on cooling to boiling point of water. Two of the four bottles in our apparatus box are all that are necessary, though all four may be employed. Nor are beads in the connecting tubes required, their omission causing no loss of $As_2O_3$ while at the same time the speed of aspiration and ease of washing out tubes are thus facilitated. That part of the aspirating tube between the flue and the absorption bottles was found to contain a large proportion of the $As_2O_3$ (deposited therein by cooling), and it must therefore be thoroughly washed out with warm dilute NaOH solution. Analysis of the arsenic can be run by any standard method, but we prefer the so-called "sulphate method," substantially as given by A. H. Low in *Technical Methods of Ore Analysis,* as we have found it the most accurate of several tried.

*Composition of Gases.*

By courtesy of the management of one of the copper smelters where work was accomplished, we submit the following table of analyses for various flues:

| Constituents | Roaster Flue Per Cent. | Blast Flue Per Cent. | Converter Flue Per Cent. | Reverberatory Flue Per Cent. | Base of Main Stack Per Cent. |
|---|---|---|---|---|---|
| Sulphur dioxide | 2.545 | 1.274 | 2.845 | 0.423 | 1.164 |
| Sulphur trioxide | 0.275 | 0.086 | 0.0515 | 0.0044 | 0.0395 |
| Carbon dioxide | 0.1136 | 6.493 | 0.2084 | 5.242 | 2.748 |
| Water vapor | 2.784 | 3.490 | 1.061 | 3.869 | 2.834 |
| Arsenic trioxide | 0.0073 | 0.0091 | 0.00073 | 0.0156 | 0.00465 |
| Oxygen | 14.02 | 10.18 | 12.04 | 10.37 | 11.88 |
| Nitrogen | 81.18 | 78.13 | 83.64 | 79.57 | 80.73 |
| Totals | 100.9 | 99.7 | 99.8 | 99.5 | 99.4 |

All of the above are volume percentages at standard conditions—760 mm. and 0° C.

The only flue that showed certain trace of carbon monoxide was the reverberatory.

Selenium and tellurium oxides were found in both reverberatory and main flues, but were not determined quantitively, appearing incidentally during $As_2O_3$ aspirations. Traces of antimony were found throughout in all flues.

About a year later, with changed conditions at this same smelter, analyses of the sulphur oxides were repeated. Changes were three in number: 1, (slight) in tonnage; 2, in character of feed (considerably more zinc being present); and 3, the flues, which were in a poor shape because of leaks during the first series of analyses, had been put in nearly perfect condition.

In all flues, both departmental and main, the $SO_2$ content was higher, due to differences in both tonnage and condition of flues. On the other hand, the $SO_3$ percentage was lower, due to greater neutralizing action of the zinc.

At base of stack the $SO_2$ volume percentage was 1.734 per cent., a raise of about 50 per cent. over figures previously given, indicating leakage; while the $SO_3$ gave a beneficent drop of over a half, becoming 0.0159 per cent., indicating possible entire neutralization of $SO_3$ by slightly greater infusion of zinc in feed.

In closing, it should be emphasized that probably none of us realize fully what can be done at the average smelter, in the matter of actual saving of dollars and cents, by means of intelligent and accurate gas analysis. By "intelligent" we mean a minute and exact log of furnace operations, and a careful checking of tonnages, with analyses of materials fed and produced. Without these any analysis is simply a figure to satisfy curiosity, and indicates nothing for future practice. A case in point will be illuminating. At one plant where we worked, the orders were to keep entirely away from all operations, confining ourselves entirely to analysis. The $SO_2$ in blast-furnace gases showed at different times tremendous variations, and the average of a large number of determinations was reported as the content of the blast-furnace flue, but the figure carried no illumination as to cause of variation, which had to be ascribed roughly to differences in tonnage and feed. At a second smelter, the same variation was observed, but a careful study of conditions showed that, at times when the $SO_2$ was high, there was less coke fed to the furnaces than the supposed minimum amount required for smooth running. When the $SO_2$ was at its low ebb in the flue the coke was fed either in normal or excess of normal amount, and the sulphur was volatilized, *and not utilized as fuel.* The furnaces ran smoothly under either condition. The indicated result, the regular running of the furnaces

with less coke, with correct utilization of the sulphur fuel sometimes thrown away, and a consequent saving in good hard cash by the management, should forever remove "intelligent" gas analysis from the schedule of luxuries at the second plant.

Aside from considerations of indicated changes in practice, a minute log of operations often serves to clear up results of a contradictory character. We remember working in a converter flue on two succeeding days, when with a larger number of converters blowing the first day the flue content of $SO_2$ was considerably lower than that of the second day, when fewer converters were blowing, with conditions of feed and products identical, which certainly looks absurdly incorrect. But we fortunately had included in our log the stage of the blow of each converter while sample was being drawn from the flue, and a study of this confirmed the result obtained, for on the second day (higher $SO_2$ content) a majority of the converters were in the stage "white metal to copper;" while on the first day the preliminary, or "slagging," stage of operations predominated. We had previously (sampling every 15 min. of the blow of about 2.5 hr. at a single converter mouth) found that the preliminary stage (oxygen used in iron silicate formation) gives in round numbers 1.0 per cent. of $SO_2$ while the succeeding blow of white metal to copper evolves as high as 6.0 per cent. of $SO_2$, the oxygen being utilized in sulphur oxidation. Without a perfect log, flue results would in this instance have been discarded with disgust, but with the log for guidance, results at first glance impossible were predicated.

Many other examples might be cited as to the importance of analyses of gases at the average smelting plant, but space does not permit. In addition to value for determining leakages between furnaces and stack emanations, or for checking up metallurgical balances of various parts of the plant, one other actual experience in "detection of crime" will be mentioned. Recently, at Anaconda, balances of the arsenic plant showed loss of some $As_2O_3$, back into the main flue with which both sets (first and second refining furnaces) are connected after the products are settled in two separate lines of kitchens. The product settled in both sets of kitchens differed by only a few per cent. in $As_2O_3$ content. Which set of furnaces and kitchens was causative of loss and what is the remedy?

Analysis for $As_2O_3$ at exhausted end of both sets of kitchens (either one would do it when comparing its loss with the total) caught the culprit, and a careful log of the two operations showed (draft the same for both) too high a heat and too great an admission of air through the refining furnace and kitchens, indicating changes in practice necessary to prevent loss.

Three final words: 1, Accuracy of analysis; 2, Minute log of all operations; and 3, Diligent study of results for contraindications of practice

observed; first, chemistry; second, bookkeeping; third, metallurgy; these three form a combination too strong to be neglected.

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## PART II.—NOTES ON DETERMINATION OF DUST LOSSES AT THE WASHOE REDUCTION WORKS, ANACONDA, MONT.

In the winter of 1910–11 an exhaustive investigation was carried on by us, with the purpose of determining as closely as possible, incidentally the velocity of our smoke stream, with accurate figures for the daily amount of stack emanation, and ultimately the amounts and character of dusts carried in this emanation. Could the gross amount of smoke passing the stack be handled by any of the processes (more or less at that time in experimental stages) either in use or being installed at various smaller smelters? The gross amount being approximately, of course, known to us at that time, if such installations, with known enormous initial expense, were made, would the amount of dust and fume, if recovered successfully, pay anything toward this large installation cost; and, if not, would the values be sufficient to guarantee upkeep and daily operation of this smoke plant? Or would dust values be so small as to render both items of expense largely chargeable to smoke alone, tending to wipe out the margin of profit on smelting costs at the prevailing price of copper? In a nutshell, how efficient is our flue system as a value catcher?

Notes on the work as turned over to the management will be later submitted, but as a preliminary, a brief description of our flue system is illuminating. It is taken from a booklet, *Brief Description of the Washoe Smelter*, compiled by members of the A. C. M. staff: "There is an elaborate flue arrangement, especially noted for its immense size. The principal flues: viz., the blast, roaster and reverberatory, are 20 ft. wide and 15 ft. high, and are of brick and steel construction. The converter flue consists of two 7 by 7 ft. flues. The blast, roaster and converter flues connect with their respective dust chambers; the reverberatory flue with the furnaces direct (after passage of gases from each furnace through two Stirling boilers in tandem for recovery of waste heat). The flues are of the following lengths:

| | *Feet.* |
|---|---|
| Blast flue | 1,653 |
| Roaster flue | 488 |
| Converter flue | 703 |
| Reverberatory flue | 1,253 |

These flues all merge into one main flue, the rough plan of which is shown in the accompanying diagram [Fig. 2]. For the first 1,234 ft. this main flue is 60 ft. wide; side walls 20 ft. high; the bottom being excavated at

an angle of 30° from the horizontal. The roof is of I-beam and brick arch construction. The remaining distance to the stack is 995 ft. of 120-ft. flue (width, this last 995 ft. being practically twin flues of 60-ft. width for further slowing velocity of gases). This portion of the flue has a roof of No. 9 sheet steel (to enhance cooling effects by greater radiation). The stack is 300 ft. high, with an inside diameter of 30 ft. The top of this stack is 932 ft. above the surrounding valley." Cross-sections of the 60-ft. and the 120-ft. flues are shown in Figs. 3 and 4.

As a result of this elaborate flue system, the distances traveled by gases of various departments between furnaces and top of stack are as follows:

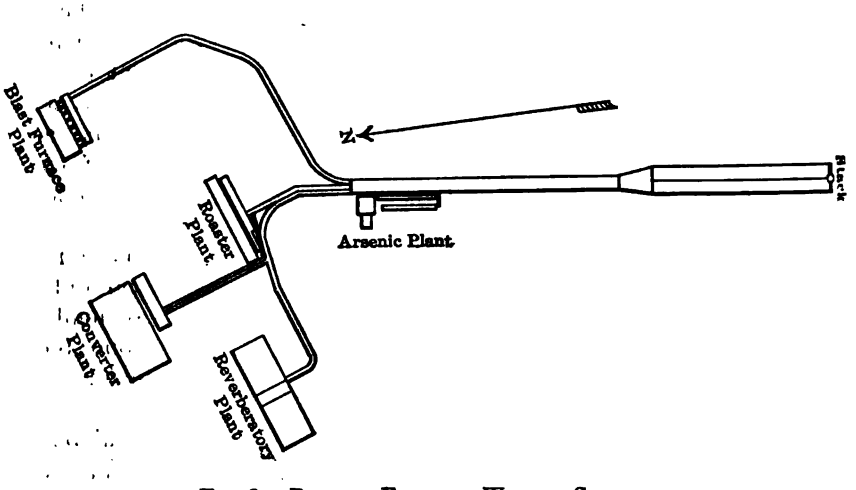



FIG. 2.—PLAN OF FLUES AT WASHOE SMELTER.

Blast gases: 1,653+1,234 at lower velocity+995 at still lower velocity and greater radiation+300 at increased velocity and reduced radiation =4,182 ft.

Roaster gases: 488+1,234+995+300=3,017 ft.

Converter gases: 703+430 (in roaster flue)+1,234+995+300=3,662 ft.

Reverberatory gases: 1,253+1,234+995+300=3,782 ft.

These figures are here inserted to emphasize three effects of the Anaconda flue system, the combination of which is unique; namely: immense distance that must be traveled by dust before chance of escape is given; lowering of velocity of smoke stream of gases from all flues twice before escape; and great relative increase of radiation (with greater cooling effect) during from 20 to 25 per cent. of the distance (covered at the lowest velocity).

Its efficiency can be judged by the results contained in the notes here-

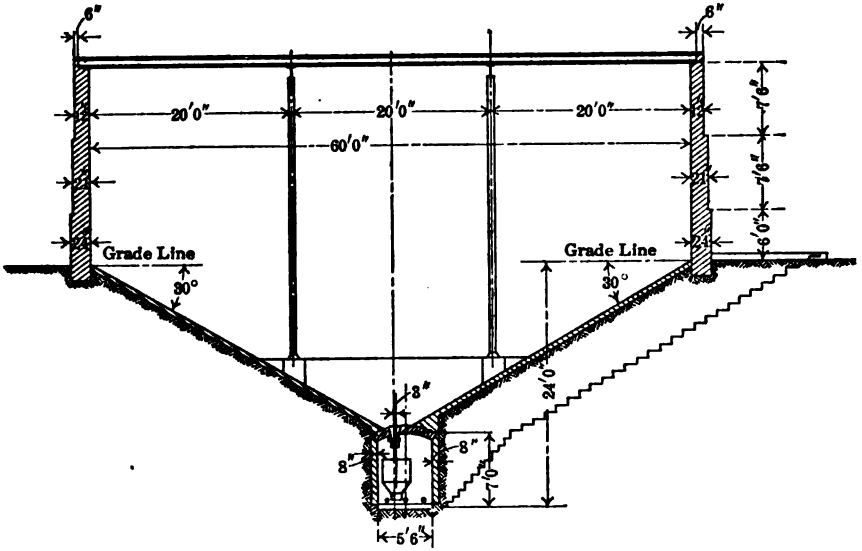



FIG. 3.—SECTION OF 60-FT. FLUE.

with submitted, covering fully methods of procedure, results obtained, and conclusions therefrom.

As the temperature of the smoke stream escaping plays a part in selecting method for analysis, that temperature is here given. The testing department records showed near base of stack for summer (1909–10) 377° F., or 192° C.; and for winter same year, 318° F., or 159° C. We ourselves checked up the temperatures 20 ft. from the base of stack in August, 1910, and obtained an average from a long series of readings of 375° F., practically that given by the testing department.

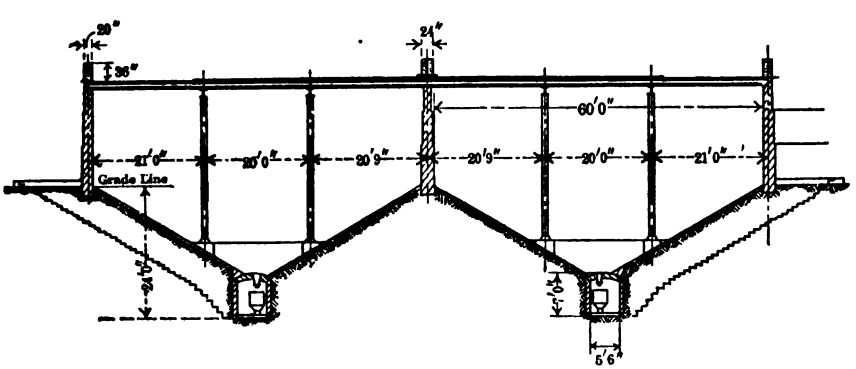



FIG. 4.—SECTION OF 120-FT. FLUE.

### *1. Various Places and Methods for Determinations.*

In looking over the flue and stack preparatory to starting tests for velocities and volumes, four places for doing the experimental work were considered.

(*a*). Stack at point previously used in 1905.

(*b*). Uptakes into stack used in 1907.

(*c*). Cross-section of 120-ft. flue, 30.7 ft. below the stack, where six holes had already been cut for dust soundings.

(*d*). The cross-section of 120-ft. flue, about 70 ft. below (*c*), where ten holes had been previously cut for dust soundings.

(*a*). The opening in the stack was discarded for two reasons. First the height from the ground (53 ft.) and the very small distance above, the point at which the gases enter the stack (13 ft. above the old baffle in stack—now down) seemed to point unmistakably to varying eddies; second, the small diameter of the hole in the stack (5.75 in.) makes the insertion of a large type, well-supported Pitot tube impossible.

(*b*). The two side uptakes into the stack were discarded because the quick sweep of a large volume of the gas around the end baffle of the flue rendered the position of flow of the gas problematical, even without the chance of eddies.

(*d*). The cross-section of the 120-ft. flue about 100 ft. below the stack seemed the best point for determinations, but on taking soundings from the top of the flue, the space above the hoppers was found to be very nearly free from dust, necessitating at some places in the cross-section Pitot tubes 35 ft. long. Of course, the mechanical difficulties in using tubes of this type would require a very large set of tubes of varying lengths, with a tremendous number of readings in order to cover the open area satisfactorily.

(*c*). The cross-section of the flue 30.7 ft. below the stack wall was therefore chosen, soundings showing that but two Pitot tubes would be needed to cover the open area successfully. The bottom of the flue at this point is above the regular line of hoppers, and therefore the area as determined by soundings would be subject to no change, by removal of dust during progress of the test. Consequently a line of 18 holes (nine to each side of the flue) was cut at this cross-section. The depth of the flue at this cross-section is about 27 ft. (on the slope up to the stack wall), and a depth of dust was found by soundings (Nov. 9, 1910) of from 9.4 ft. near middle of flue to 18.45 ft. on the sides. Later soundings during progress of the test (Nov. 25 and 26, 1910) confirmed these first soundings, the greatest variation at any point being 0.35 ft.

### 2. *Instruments Used.*

The instruments used in the test are shown in Figs. 5 and 6.

*Pitot Tubes.*—The Pitot tubes employed in both determinations were two in number, one having an extension of 8 ft. down, the other 14 ft. They are of the same construction and of the same general type as those outlined in Fig. 5, though the impact arm is considerably larger in dimensions. They were carefully checked against each other and gave true readings.

*Bristol Pyrometer.*—The temperature determinations were taken with the Bristol pyrometer, which had been previously checked in several of the flues against mercury thermometers, and read correctly within very narrow limits (a few degrees under widely different temperatures). By means of this instrument, temperatures were taken at every point of Pitot readings.

*Barograph and Standard Mercury Barometer.*—The pressures were taken for each hole at finish of reading, by means of the barograph in the testing department, which is standardized daily by means of the United States

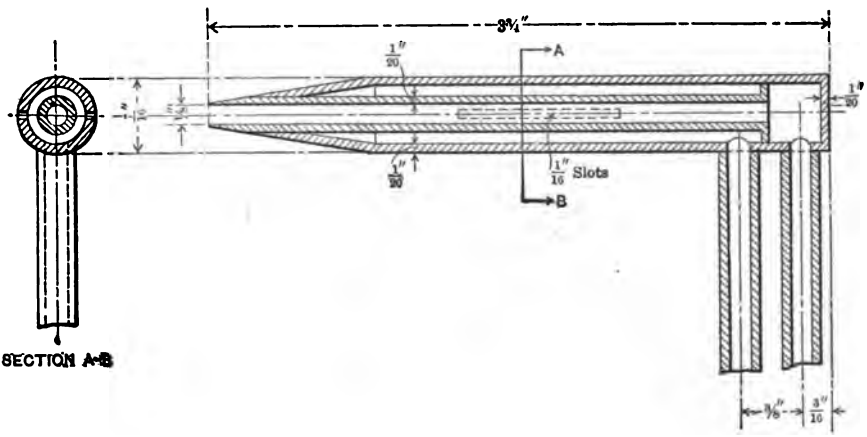



FIG. 5.—PITOT TUBE.

standard mercury barometer, also in the testing department. As a matter of fact, these pressures are slightly too high, but it was deemed more accurate to use them than to trust to an aneroid barometer at the stack, which would fluctuate too greatly during the exceedingly variable weather of the determinations.

*Ellison Differential Draft Gauge.*—The Ellison draft gauge proved a distinct advance over any other form of U-tube so far known for taking the Pitot readings. The gauge is made to read directly in inches of water when the Ellison gauge oil is used therein. Consequently before starting the determinations at the main flue, the two gauges (but one of which,

No. 1, was used on the main flue) were taken to the MacDougall flue and were compared with each other, using the Ellison gauge oil as a reading medium. The two gauges checked absolutely. They were then checked by means of T's inserted in the connecting tubing against each other, using gauge oil in each, and also against water in a delicate reading U-tube. Again they checked as closely as the U-tube could be read. Gauge No. 1 was then measured at various points and checked up with both oil and water, and found to read directly in inches of water when Ellison gauge

1. Government Mercury Barometer.
2. Barograph, Checked against Mercury Barometer.
3. Bristol Pyrometer, connected with 6.
4. Ellison Gauge, boxed, and on base with lag screws for leveling; connected with 5.
5. Large Size Pitot Tube.
6. Thermo-Couple in Iron Sheath.

FIG. 6.—INSTRUMENTS USED IN DUST DETERMINATION.

oil was used as a reading medium. Gauge No. 1 was then taken to the main flue and tried. The air temperature was nearly 0° C., and under these conditions the oil became too viscous for accurate work, and was therefore perforce discarded as a reading medium; absolute alcohol and petrolic ether were next in line, and the latter was adopted as a reading

medium, because of its lower specific gravity, and consequent wider range of reading velocity head minus static pressure, which remainder is small at some points in the cross-section of the main flue where work was done. Readings were tabulated directly on the scale, which had been already proved to be inches of water when using gauge oil for a reading medium. On finishing the determinations, the specific gravities of the Ellison gauge oil and the petrolic ether were determined, and each reading on the gauge converted to inches of water when using petrolic ether as a reading medium. Petrolic ether gave a specific gravity of 0.640254; Ellison oil, of 0.828625. Consequently the factor for conversion was

$$\frac{\text{Petrolic ether}}{\text{Ellison oil}} = 0.773.$$

The manufacturer's figure for specific gravity of Ellison oil is 0.834, but we found 0.829 correct. However, the small error could not be read on that part of the scale we used.

### *3. Procedure in Making Determinations.*

In making determinations every possible precaution to guard against error was taken. The two Pitot tubes used were 8 ft. and 14 ft. long, respectively. The average height above the dust at which initial readings were taken in all 18 holes was 2 ft., succeeding readings being taken at 2-ft. distances to within 2 ft. of the top of the flue. The reading at each point is an average of three readings taken at intervals of from 1 to 2 min., so that each point reading is approximately an average of about 5 min. Each time before insertion in a hole, each Pitot tube was blown out to insure accuracy, and in each hole at a point 8 ft. below the top of the flue the Pitot tubes were checked against each other, three readings being taken with each.

To guard against the possibility of change in the specific gravity of the petrolic ether used in the Ellison gauge by absorption of dust through the Pitot tube or any other source, the petrolic ether was changed daily, the gauge being emptied every morning, the old ether thrown away, and a fresh supply used for refilling from the large, air-tight, stock bottle. Specific gravity determination of the petrolic ether was made at the end of the test, from the same stock bottle.

During readings in all holes the Pitot tubes were turned at various angles to determine the point of greatest velocity head. In all holes except two, it was found to coincide with the run of the flue. In two holes (one in each side of the 120-ft. flue) it was found necessary to turn the point of the Pitot tube toward the inside center wall of the flue, showing that gases were sweeping out toward the open stream on the outsides of the flue.

*Soundings for Depth used in Area Determination.*

| First Determination | | | | | | No. of Holes Used in Making Determinations[b] | Second Determination | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Date and Place | Free Space | Date and Place | Free Space | Average Free Space[a] | Variations from Average | | Date and Place | Free Space | Date and Place | Free Space | Average Free Space[c] | Variations from Average |
| Nov. 9, 1910 East Main Flue | 9.50 | Nov. 15, 1910 East Main Flue | 9.85 | 9.68 | 0.18 | 1 | Nov. 15, 1910 East Main Flue | 9.85 | Nov. 26, 1910 East Main Flue | 9.50 | 9.68 | 0.18 |
| | 11.65 | | 11.40 | 11.52 | 0.13 | 2 | | 11.40 | | 11.70 | 11.55 | 0.15 |
| | 14.20 | | 14.50 | 14.35 | 0.15 | 3 | | 14.50 | | 14.40 | 14.45 | 0.05 |
| | 15.92 | | 16.30 | 16.11 | 0.19 | 4 | | 16.30 | | 16.60 | 16.45 | 0.15 |
| | 16.95 | | 16.80 | 16.88 | 0.08 | 5 | | 16.80 | | 16.70 | 16.75 | 0.05 |
| | 17.60 | | 17.30 | 17.45 | 0.15 | 6 | | 17.30 | | 17.35 | 17.32 | 0.03 |
| | 15.15 | | 15.00 | 15.08 | 0.08 | 7 | | 15.00 | | 15.20 | 15.10 | 0.10 |
| | 11.95 | | 11.70 | 11.82 | 0.13 | 8 | | 11.70 | | 11.70 | 11.70 | 0.00 |
| | 8.70 | | 9.00 | 8.85 | 0.15 | 9 | | 9.00 | | 8.80 | 8.90 | 0.10 |
| Nov. 9, 1910 West Main Flue | 8.80 | Nov. 15, 1910 West Main Flue | 8.55 | 8.68 | 0.13 | 10 | Nov. 15, 1910 West Main Flue | 8.55 | Nov. 26, 1910 West Main Flue | 8.55 | 8.55 | 0.00 |
| | 11.10 | | 11.10 | 11.10 | 0.00 | 11 | | 11.10 | | 11.00 | 11.05 | 0.05 |
| | 13.60 | | 13.40 | 13.50 | 0.10 | 12 | | 13.40 | | 13.75 | 13.58 | 0.18 |
| | 14.63 | | 14.55 | 14.59 | 0.04 | 13 | | 14.55 | | 14.20 | 14.38 | 0.18 |
| | 14.90 | | 14.70 | 14.80 | 0.10 | 14 | | 14.70 | | 14.45 | 14.58 | 0.13 |
| | 13.90 | | 13.65 | 13.78 | 0.13 | 15 | | 13.65 | | 13.60 | 13.62 | 0.03 |
| | 13.20 | | 13.10 | 13.15 | 0.05 | 16 | | 13.10 | | 12.95 | 13.02 | 0.08 |
| | 11.00 | | 11.15 | 11.08 | 0.08 | 17 | | 11.15 | | 10.80 | 10.98 | 0.18 |
| | 8.60 | | 8.55 | 8.58 | 0.03 | 18 | | 8.55 | | 8.60 | 8.58 | 0.03 |

[a] Used as "depth" in first determination. [b] Reading east to west. [c] Used as "depth" in second determination.

The Ellison gauge was protected in every instance from the wind by a three-sided shelter (wood and corrugated iron), shown in Fig. 7, and was placed on a shaded shelf therein, to escape heat effects of flue and sun on the petrolic ether. At every point in each hole the Ellison gauge was set true by a double level, and read to zero, a drop or two of ether from a small stock bottle being occasionally added when necessary, and after taking the three readings at this point, the gauge was again checked to zero. It was found that change of position of occupants of shelter would of course throw the gauge off the level slightly, so that, after start-

FIG. 7.—SHELTER FOR GUAGE IN DUST DETERMINATION.

ing readings at any point, absolute quiet was insisted upon. If level of gauge did not read zero, both before and after taking the readings at each point, readings recorded were thrown out, and new ones taken.

During readings, the Pitot tubes were clamped firmly in place by a

screw device, and the holes in the flue tamped with waste to eliminate any possible error from outside air currents.

In each of the two determinations, the soundings (from which the areas were figured) were made at start and end of test. Some of the 18 holes checked absolutely; others varied somewhat, the greatest variation being 0.35 ft. The averages of soundings Nos. 1 and 2 were taken in figuring the area of the first determination, and the averages of soundings Nos. 2 and 3 in figuring the area of the second determination. These soundings are appended, as they may be of interest in showing the amount of dust in flue at cross-section of soundings.

### 4. *Calculation of Results.*

*Velocities.*—For the calculation of velocities each hole was taken by itself in each determination, and as the points of reading were practically equidistant from each other as well as from the top and bottom of the flue, the average of the readings was taken as the average Pitot reading for each hole. The velocities were calculated by the formula $V = \sqrt{2gh}$, in which V equals velocity; g equals the acceleration due to gravity; h equals the height in feet of a column of gas equal to the displacement of the reading medium by the velocity pressure minus the static pressure. The Pitot tube is of such construction as to read the velocity pressure minus the static pressure direct. The petrolic ether displacement was first calculated to inches of water and finally to feet of water for greater ease of later calculations. In calculating, h, the specific gravity of the gas, was taken as based on former gas analyses made in previous years. The specific gravity calculated therefrom is 1.013, or a molecular weight (H = 2) for the gas of 29.2146. Or in other words, at 760 mm. pressure and 0° C. temperature, 1 liter of gas weighs 1.307689 g. or 1.308 g. (1 liter of H under same conditions weighing 0.089523 g.). Or 29.2146 g. of gas divided by 1.308 equals 22.33 liters of gas, at standard conditions. The following formula was next employed for calculating the number of liters of gas at flue conditions which are equal to 22.33 liters at standard conditions:

$$V^F = V^S \frac{(273 + t)\ 760}{273\ B} = V^S \frac{(1 + 0.00367\ t)\ 760}{B}$$

$V^F$ equals number of liters of gas (volume) sought at flue conditions of temperature and pressure; $V^S$ equals volume of gas at standard conditions (a constant 22.33 liters); t equals the temperature of the gas; B equals the barometric pressure. As the figures for t and B vary in the different holes, $V^F$ was figured for each hole separately.

*Areas.*—In figuring areas, each side of the 120-ft. flue was taken separately, and subdivided into nine sections. The depth of the free space at each of the nine holes was determined, the figures taken being average depths of two soundings, one taken before and one after each determination of velocity. The basis of widths of section areas was the measured distances between holes and the measured distance between the outside holes and the sides of flue. The widths taken for each section area are the distances midway from hole to hole, with the exception of sections Nos. 1, 9, 10, and 18. The width of section area No. 1 is one-half the distance between holes Nos. 1 and 2, plus one-half distance to flue wall from hole No. 1. Widths of section areas Nos. 9, 10, and 18 are similarly obtained.

The remaining one-half distance between hole No. 1 and east wall of east flue was thrown out when calculating areas, for the velocity decreases in a more or less steady ratio, from the maximum at the hole to zero at the wall. This distance is of small account in figuring area, as in the four sides where it is discarded it averages about 8 ft. high (clearance above the dust), and never more than 1 ft. in width.

Width times average depth of course equals area for each section in square feet.

Pitot readings were taken at several of the 18 holes at a distance of 1 ft. from the top of the flue. In every case the Pitot reading at 1 ft. was very nearly a half of that found at 2 ft., in the same holes. This figures to a decreased velocity of approximately three-fourths as much at 1 ft. as at 2 ft.

Volume figures from areas and velocities previously calculated are certainly too high, as they assume full area of the flue and take no account of either any decrease in velocity above 2 ft. from top of flue or below 2 ft. from top of dust. Nor do they take cognizance of retarding of velocity by steel work in flue. In an attempt to ascertain the actual volume, these three things were figured on and approximated by taking off the top foot from area, assuming full velocity from 2 ft. below top of flue to 1 ft. below top; and also full velocity from 2 ft. above dust to dust; and not figuring on any retarding of velocity by steel supports of flue.

*Daily Volume of Gas.*—In calculating the daily amount of gases passing through the flue, each area is multiplied by its own velocity, giving cubic feet per second. Adding together the results obtained for all nine holes of each side gives the volume of gases per second in cubic feet passing each side of the flue. Multiplying by 86,400 (number of seconds in 24 hr.) gives volume of gases per 24 hr. at flue conditions of temperature and pressure.

For calculating to 0° C. temperature and 760 mm. pressure (to form

a basis for comparison with other results) the following well-known formula is employed:

$$V^s = V^r \frac{273\ B}{(273 + t)\ 760} = V^r \frac{B}{(1 + 0.00367\ t)\ 760}$$

For B, the nine barometric readings taken during testing the nine holes to a side were averaged, and for t the temperatures taken while reading each point in the nine holes to a side were also averaged. Working out formula, this gives the daily volume of gas passing each side of the flue reduced to standard conditions of temperature and pressure.

In an attempt to approximate the daily amount of gas at all times of the year at the temperature and pressure of the atmosphere surrounding the stack, some averages of temperature and pressure for two and one-half years (February, 1906, to September, 1908) which were figured in the testing department, were taken. Figures are: temperature average, 4.6° C.; pressure average, 624.84 mm. Using the formula

$$V^a = V^s \frac{(273 + t)\ 760}{273\ B} = V^s \frac{(1 + 0.00367\ t)\ 760}{B}$$

where $V^a$ equals volume at atmospheric conditions; $V^s$ equals volume at standard conditions; t = 4.6° C.; and B = 624.84 mm., the result is the average daily volume of gas after it leaves the stack, and adjusts itself to surrounding conditions. These figures are given under "Results Obtained."

*Results Obtained.*

| | | Cu Ft. per Day. |
|---|---|---|
| November, 1910 (full area) | No. 1 | 1,470,223,000 |
| | No. 2 | 1,430,367,000 |
| November, 1910 (area less 1 ft.) | No. 1 | 1,359,564,000 |
| | No. 2 | 1,322,369,000 |

After calculating velocity and area and volume for each hole separately, the volume was divided by the area to obtain the average velocity (feet per second). The velocity readings for all nine holes to each side were then added together and divided by nine, to see how closely this method would check the other for velocity feet per second per side of flue. Figures follow:

| | av ÷ a = Average Velocity Feet per Sec. | 9v ÷ 9 = Average Velocity Feet per Sec. |
|---|---|---|
| East Side main flue: | | |
| First determination | 21.19 | 21.07 |
| Second determination | 21.94 | 21.70 |
| West Side main flue: | | |
| First determination | 22.01 | 21.83 |
| Second determination | 21.26 | 21.12 |

The maximum velocity was found at hole No. 2 on the east side in both determinations; and at hole No. 17 on the west side in both determinations; while in both determinations, the minimum velocity was at the holes nearest the center dividing wall of the 120-ft. flue—hole No. 9 on the east and hole No. 10 on the west, all holes being numbered from east to west across the entire 120-ft. flue.

| | East Side—Main Flue Feet per Second | West Side—Main Flue Feet per Second |
|---|---|---|
| Maximum Velocity: | | |
| First determination | Hole No. 2—28.90 | Hole No. 17—29.72 |
| Second determination | Hole No. 2—31.27 | Hole No. 17—29.23 |
| Minimum Velocity: | | |
| First determination | Hole No. 9—11.84 | Hole No. 10—12.37 |
| Second determination | Hole No. 9—12.42 | Hole No. 10—12.30 |

Over 700 Pitot readings were taken in the two determinations.

The volumes of gases emanating from stack were, for 24 hours:

| | | Cu. Ft. |
|---|---|---|
| Flue conditions | 169° C. and 622.9 mm. | = 2,649,530,000 |
| Standard conditions | 0° C. and 760 mm. | = 1,340,960,000 |

The average velocity in one 120-ft. flue was (assuming 80 per cent. clear space) 12 ft. per second.

*Plant Operations During Tests.*—During the first determination, the following units of the plant were in operation: Five blast-furnace units; seven reverberatory furnaces; 37.46 MacDougall furnaces; eight converters. During the second determination, blast furnaces and converters were the same, while six reverberatory and 29.43 MacDougall furnaces completed the operation.

Summarized on a basis of maximum capacity:

| | First Determination | Second Determination |
|---|---|---|
| Blast furnaces | 5/7 | 5/7 |
| Reverberatory furnaces | 7/8 | 6/8 |
| MacDougall furnaces[a] | 37.46/64 | 29.43/64 |
| Converters | 8/13 | 8/13 |

[a] MacDougall furnaces given on basis of 64 furnaces—the *full* equipment.

## Dust Determinations at Stack.

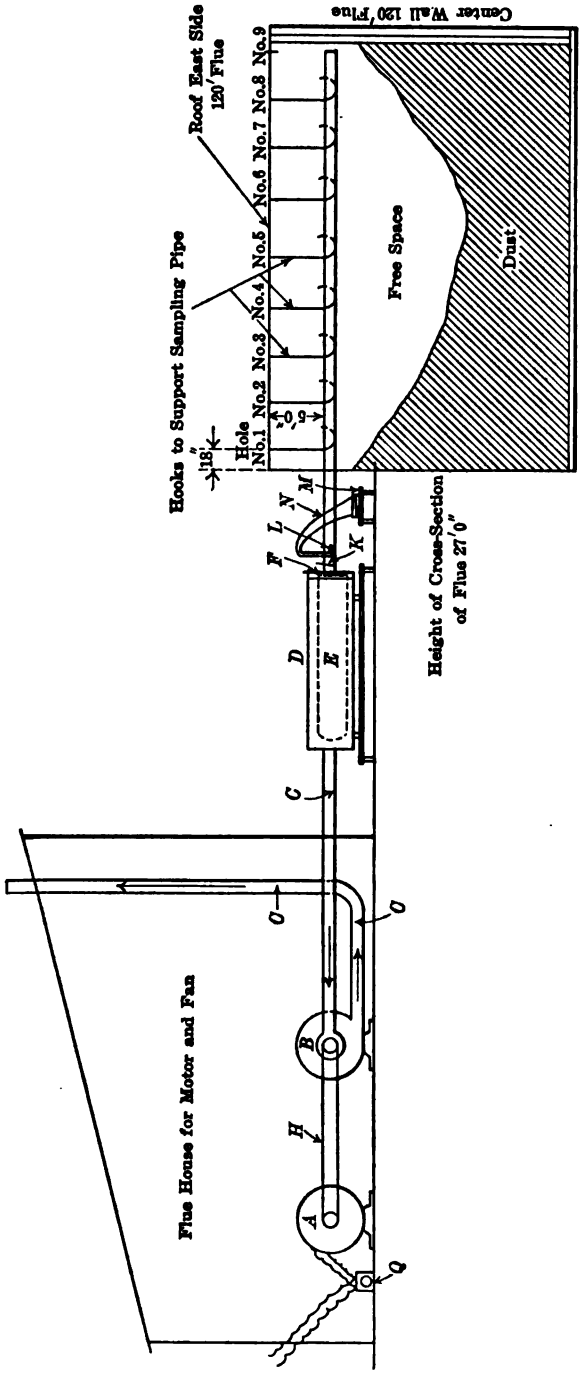
### 1. *Place, Apparatus, and Methods of Sampling.*

*Place.*—The place chosen for sampling the dust and fume emanating from the Washoe Smelter flue was the same as that selected for velocity determinations, a line across the large 120-ft. flue, 30.7 ft. below the base

of the stack wall. Fig. 8 is a sketch showing the dust-catching apparatus. This place was chosen in preference to the two others considered (stack and arches into the stack) largely for the same reason for which it was given the preference in former velocity work: namely, to escape swirling eddies, which has been found to occur largely in both places aforementioned. Then, too, the apparatus employed was of such a nature as to render the use of the sampling station in the stack, at a considerable height above the ground, impossible. Of course, it must not be forgotten that some small percentage of the dust (which passes this line 30.7 ft. below the stack wall) is probably precipitated during its later passage into the arches and up the stack. But that a very small percentage, even of this precipitated dust and fume, falls *inside* the stack is shown by the relatively long time required for assembling the dust in the hoppers at the base of the stack. Reasons why so little dust and fume are caught after passing the sampling plant are two-fold. The first (based on velocities) is that the area being so soon constricted after the passage of the sampling point, there is a quick corresponding jump in velocity from approximately 21 ft. per second (average of flue) at sampling line to about 46 ft. per second in the stack, which would tend to overcome any great tendency of dust to settle by precipitation caused by greater friction on walls of stack over walls of flue, and, consequently, the greater part of such dust and fume, *if precipitated*, would largely be swept out at top of stack. Then, too, the time in flue and stack after passing sampling line is so short (about 8 sec.) that precipitation after passing stations must be slight indeed.

The second reason is arrived at by analysis of dusts. The insoluble is so low as to stamp any further dust settling after passing sampling stations as slight, and the conclusion is forced that the flue, *as now constituted*, possesses a remarkably high efficiency as a *dust* settler, nearly everything now passing the sampling stations 30.7 ft. below the stack wall being "*fume*" so called, and not precipitable under the slightly changed conditions beyond the sampling stations, neither in the flue itself nor yet in the stack. So much for possible criticisms as to the possibility of dust obtained at sampling stations 30.7 ft. below stack wall being larger in any great degree than the emanations from top of stack. Of course, there is some later precipitation, else there could be no dust in hoppers under the stack and above sampling stations, but, for all reasons just mentioned, we believe it to be far below 1 per cent., which is certainly within the errors of sampling.

*Apparatus and Methods of Sampling.*—In 1905, dust determinations were attempted in the stack, measuring the volume of gas (containing the dust obtained) by means of water aspiration, using a meter for gas readings. These samples averaged something like 70 cu. ft. of gas




*A.* 3.5-h. p. D. C. Motor.
*B.* 30-in. Buffalo Blower.
*C.* 5-in. Stove Pipe.
*D.* Sheet Iron Cylinder, 2 ft. diameter by 8 ft. long.
*E.* Asbestos Bag wired to flange F
*F.* 13-in. Diameter.
*H.* Belt connecting motor and fan.
*K.* Thermometer.
*L.* Pitot Tube.
*M.* Ellison Draft Gauge.
*N.* Rubber Tubing.
*Q.* Resistance Box.

FIG. 8.—DUST-CATCHING APPARATUS AND CROSS-SECTION OF EAST HALF OF 120-FT. FLUE, SHOWING OPEN AREA AT TIME OF DUST DETERMINATION

(standard conditions) and a fraction of a gram of dust. Such methods might be termed "chemical" from the smallness of measures obtained, and while perhaps accurate for the short time employed, are yet questionable when applied to a volume like 1,450,000,000 cu. ft. per day (standard conditions) and nearly double that amount at flue conditions. We soon gave up all such "chemical" methods, and turned perforce to some determinations made by the testing department in 1907. After a thorough trial, we were convinced that this method (with some variations) would give the best possible results. In brief, the gases were drawn from the flue by means of a 30-in. rotary blower fan, driven by a variable speed 3.5-h.p. electric motor. The dust and fume contained in the gases were filtered out by means of a very closely woven thick "blanket" asbestos bag, suspended in a large air-tight drum; and the velocity (and consequently the volume) of gases passed was obtained by readings with small Pitot tubes and thermometers placed in the sampling tube between the flue and the drum.

The filtering medium was tried out thoroughly, and proved much more efficient than the regular woolen bag of the bag house; in fact, efficiency is absolute as far as dust is concerned, and nearly the same for fume. Of course, the efficiency of the filter is aided by two things: lowering of temperature due to radiation from large drum surface, and great decrease in velocity in the bag over the velocity in the sampling pipe. The temperature in the bag is still high enough to be above the condensation point of the water vapor but below that of $As_2O_3$.

Variations from previous methods were as follows: In the first place, they made use of the arches of the stack, which we discarded at once because of the presence of eddies. Then, too, they made use of a very small sampling pipe in order that they might with a *fixed* speed motor obtain the same velocity in sampling pipe as in the flue. This necessitated taking Pitot readings either in the very small sampling tube, where the friction on the sides would be relatively enormous, and correct readings very hard to obtain; or else in the large 5.5-in. pipe between the drum and the fan, which would include any leakage around the removable head of the drum, making gas readings too high and dust readings consequently too low. Our testing engineer insisted that dust must be drawn from the flue at a velocity at least as high as that within the flue, otherwise the sample might contain less dust and fume than present in the smoke stream. He frankly acknowledged no proof in support of his opinion, so we tried it out. We made five runs in the same place, three with 5.5-in. sampling pipe, with an average velocity of about 20 ft. per second in the sampling pipe, and two with 3-in. sampling pipe and an average velocity therein of approximately 50 ft. per second. In all five instances the flue velocity was about 28 ft. per second. We experienced

greater fluctuations in Pitot readings with the 3-in. pipe, and consequently greater probabilities of inaccuracy of reading, but when we figured the dusts there was slight variation in them, the said slight variation being easily ascribable to difference in dust content on different days.

Consequently, we adopted the large 5.5-in. pipe as a sampling tube for our determinations, and took the readings between the flue and the drum, which eliminated all chance of reading in any leakage of air.

Preliminary runs were made to determine the life of the bag, and the filter was found efficient up to 20 hr. Judged by the appearance of the bag at the end of that time, its lifetime is probably about 30 hr. in the dust stream of the drum bag holder, though it would probably be less than that if used in the flue itself at slightly higher temperature.

Runs of from 100 to 120 min. at various places in the flue were found to check as closely as longer runs, and were therefore adopted as time for regular determinations. As compared with 1905 determinations, with 70 cu. ft. (standard conditions) and trace of dust, we obtained about 10,000 cu. ft. (standard conditions) and from 400 to 500 g. of dust. We experienced mechanical difficulties with the sampling pipe, and finally adopted the scheme of first running in the full 60 ft. of pipe, strongly riveted, about 58 ft. into the flue from each side, supporting it with hooks hung from the steel roof of the flue through small holes pierced therein, and pulling out and cutting off the pipe at lengths corresponding to the stations where velocity measurements had previously been taken. This gave stations on both east and west sides of flue, with a total of 18 for the entire 120-ft. flue. For these 18 stations, 28 samples were taken, besides a large number of preliminary runs in trying out the apparatus.

For the east side, all samples were taken in a line across about 5 ft. below the steel roof, while for the west side, 6 ft. below was approximated.

During each determination Pitot readings were taken every 2 min. during the entire run, and the 2-min. readings were all taken at each station, in the center of the 5.5-in. pipe. This pipe was sectioned into six areas, and in each station, with its different length of sampling pipe, the readings in the center of the pipe were corrected by a factor obtained by repeatedly reading the Pitot tube in all six areas, and applying the average reading (or rather the square roots of same, averaged and squared) to the center reading.

As the velocities vary as the square roots of the Pitot readings, the square roots of the 2-min. readings were averaged and squared to obtain the average Pitot reading for the entire run. As the temperatures were of slight variation, they were also averaged for the entire run.

As the dust on each side approached within about 8 ft. of the steel roof of the flue, the necessity of fighting shy of settled dust rendered it impossible to sample horizontally at a lower depth than 6 ft., though the

centers of both sides of the flue showed a clear space of from 16 to 17 ft. below the steel roof. But sampling perpendicularly is open to the objection of low results by loss of some of the dust that strikes and tends to precipitate on sides of sampling tube, later falling back. Dust in the gas might very well vary at a distance of 5 ft. to 16 ft. below the flue roof, because of difference in velocity of gases at the two points; but it is probable that fume does not, for we found pretty nearly the same amount of dust (and fume) all the way across the flue at stations with velocities ranging from 11 to 31 ft. per second, and it is reasonable to suppose that if no great variation occurs horizontally within such a range of velocities perpendicular variations are also slight.

It was found best to cement the bag by a preliminary run before each determination. The dust thus obtained was left in the bag and weighed *in toto* (bag and dust) and after the regular run the bag was again taken out and weighed with the contained dust. Gain in weight equals dust obtained during determination. The bag was then shaken out thoroughly and the dust bottled for screening (to take out asbestos particles) and analysis.

Before putting in the bag and drum, a preliminary run of a few minutes at high speed with rapping and shaking of pipe was made, and the pipe was shaken and rapped at intervals of the regular run, and finally just before the finish.

The Pitot tube was read, as in the regular velocity determinations, by means of the Ellison differential gauge, containing petrolic ether.

To sum up, low dust results might be obtained by reading in air leaks or by taking the velocity readings in the center of the pipe without correction for areas of pipe where velocity is lower because of friction. Both of these cause low results by giving too high a volume of gas for the dust obtained. Low results again may be obtained if the small amount of dust collecting in sampling pipe is not drawn out into the bag by increasing the velocity at end of run, and shaking and rapping the pipe. All of these errors were carefully guarded against. High dust results might occur from taking the Pitot readings in an outside area of the sampling tube, without a correction for the increased velocity in the center.

Either high or low dust results might occur from incorrect gauges for reading, or imperfect determinations of the specific gravity of the petrolic ether used as a medium. So far as known, every possible error has been carefully eliminated from the work.

### 2. *Results Obtained.*

The figures for the dusts here submitted were obtained by two methods of figuring: First and simplest, by taking the arithmetical average of the dusts obtained at all stations, based on two standards (for which see

previous report of velocity determinations). Assuming the figure 1,450,000,000 cu. ft. for the daily volume of gas at standard conditions gives the first figures; while the second (more nearly correct—for reasons see previous report of velocity determinations) are based on a daily discharge of 1,341,000,000 cu. ft. at standard conditions.

After figuring the arithmetical average, the dust figures were also obtained by a more correct method of calculation, each dust being calculated by its own area and velocity, using the figures obtained from previous velocity report. These results checked the arithmetical average within much less than 1 per cent.

*Dusts Obtained in Main Flue.*

*East Side (Arithmetical Average).*

| Date of Sample, 1911 | Station No. | Velocity in Pipe Feet per Second | Dust Tons per Day Based on 1,450,000,000 Cu. Ft. | Dust Tons per Day Based on 1,341,000,000 Cu. Ft. |
|---|---|---|---|---|
| Jan. 16 | 1 | 19.325 | 87.000 | 80.460 |
| Jan. 20 | 1 | 15.186 | 79.469 | 73.495 |
| Jan. 21 | 1 | 50.560 | 85.672 | 79.232 |
| Jan. 30 | 2 | 21.176 | 85.927 | 79.458 |
| Jan. 25 | 3 | 17.678 | 83.239 | 76.982 |
| Jan. 26 | 4 | 18.722 | 83.784 | 77.486 |
| Feb. 17 | 4 | 19.147 | 84.482 | 78.131 |
| Feb. 14 | 5 | 19.823 | 81.111 | 75.014 |
| Feb. 13 | 6 | 19.138 | 88.276 | 81.641 |
| Feb. 11 | 7 | 19.138 | 82.927 | 76.694 |
| Feb. 7 | 8 | 17.197 | 78.828 | 72.904 |
| Feb. 6 | 8 | 20.200 | 79.594 | 73.609 |
| Feb. 4 | 8 | 18.170 | 82.675 | 76.461 |
| Feb. 4 | 9 | 17.657 | 84.151 | 77.826 |
| Feb. 1 | 9 | 17.818 | 83.431 | 77.160 |
| Arithmetical average | ..... | ..... | 83.371 | 77.104 |

*West Side (Arithmetical Average).*

| | | | | |
|---|---|---|---|---|
| Mar. 1 | 10 | 17.333 | 72.68 | 66.43 |
| Mar. 2 | 10 | 18.256 | 75.02 | 69.38 |
| Mar. 2 | 11 | 18.292 | 73.06 | 67.57 |
| Mar. 3 | 12 | 18.278 | 80.01 | 73.99 |
| Mar. 4 | 13 | 18.606 | 93.45 | 86.44 |
| Mar. 6 | 13 | 17.695 | 85.80 | 79.35 |
| Mar. 6 | 14 | 19.247 | 93.20 | 86.20 |
| Mar. 7 | 14 | 18.018 | 95.91 | 88.69 |
| Mar. 7 | 15 | 19.080 | 91.51 | 84.63 |
| Mar. 9 | 16 | 19.675 | 77.03 | 71.23 |
| Mar. 10 | 16 | 17.251 | 77.17 | 71.37 |
| Mar. 10 | 17 | 18.362 | 81.79 | 75.65 |
| Mar. 11 | 18 | 16.416 | 78.54 | 72.63 |
| Arithmetical average | ..... | ..... | 82.713 | 76.428 |
| Arithmetical average (both sides) | ..... | ..... | 83.066 | 76.790 |

Figuring the dust by the roundabout but exact way aforementioned (ascribing to each sample its own flue velocity and area for each separate sample and station) gives correct figures as follows: Using 1,450,000,000 cu. ft. as daily discharge at standard conditions from both sides of flue, gives 83.015 tons of dust per day against 83.066 as the arithmetical average; while employing the figure that we believe to be more nearly correct, viz., 1,341,000,000 cu. ft. per day, gives 76.777 tons per day against 76.790 arithmetical average.

In the east side of 120-ft. flue there is small variation in dusts at different stations; while in the west side the greater variation is probably due to greater velocity in the west flue. There is more nearly a swirling eddy at stations 13, 14, and 15 (west side) than at stations 5, 4, and 3 (east side), the velocity in the west side being slightly higher, with greater, bounding back from baffle 30 ft. above stations. The greater amount of fume on the east side is probably due to imperfect admixture of gas and fume from the sectional flues in the 60-ft. flue before entering the 120-ft. flue. Slight differences in previous gas analyses showed similar evidence.

*Analysis of Dusts.*—Eighteen samples were made up for analysis, all being screened through 40 mesh for removal of asbestos from bags. They represent the dusts at the eighteen different stations of sampling

*East Side of Flue.*

Nos. 1 and 8. Composite 3 determinations
Nos. 2, 4, and 9. Composite 2 determinations
Nos. 3, 5, 6, and 7. One determination only

*West Side of Flue.*

Nos. 10, 13, 14, and 16. Composite 2 determinations
Nos. 11, 12, 15, 17, and 18. One determination only

These 18 samples were analyzed, with the following results:

| Number | Ag Oz. per Ton | Au Oz. per Ton | Cu Per Cent. | $As_2O_3$ Per Cent. |
|---|---|---|---|---|
| 1 | 3.8 | 0.005 | 1.05 | 40.3 |
| 2 | 4.0 | 0.005 | 0.92 | 40.6 |
| 3 | 3.9 | 0.005 | 1.03 | 40.7 |
| 4 | 3.4 | 0.005 | 0.96 | 39.9 |
| 5 | 4.1 | 0.006 | 0.95 | 31.0 |
| 6 | 4.2 | 0.006 | 0.90 | 32.8 |
| 7 | 4.4 | 0.006 | 1.15 | 31.1 |
| 8 | 4.3 | 0.006 | 0.92 | 31.6 |
| 9 | 4.3 | 0.006 | 1.10 | 35.7 |
| 10 | 4.9 | 0.006 | 0.95 | 29.7 |
| 11 | 5.5 | 0.007 | 0.82 | 33.0 |
| 12 | 5.0 | 0.006 | 1.13 | 30.8 |
| 13 | 4.5 | 0.006 | 1.01 | 33.2 |
| 14 | 4.7 | 0.006 | 1.02 | 30.8 |
| 15 | 4.7 | 0.006 | 0.91 | 39.4 |
| 16 | 4.1 | 0.005 | 0.82 | 32.2 |
| 17 | 4.8 | 0.006 | 0.93 | 32.4 |
| 18 | 5.3 | 0.006 | 1.18 | 31.3 |
| Average..... | 4.44 | 0.0058 | 0.986 | 34.2 |

The methods used for copper (color and electrolytic) gave results far too high, as shown by our own later analysis. A composite sample of 10 g. of each of the 18 samples was made up and thoroughly mixed, and run in the fall of 1911. Results are herewith submitted:

*Analysis of Main Flue Dust Passing Base of Stack.*

| | Per Cent. | Ounces |
|---|---|---|
| $SiO_2$ | 4.19 | ..... |
| Insol | 5.68 | ..... |
| $Al_2O_3$ | 3.14 | ..... |
| $Fe_2O_3$ | 2.78 | ..... |
| CaO | 0.50 | ..... |
| MgO | 0.69 | ..... |
| Total S | 10.05 | ..... |
| Total $SO_3$ | 23.25 | ..... |
| Free $SO_3$ | 3.60 | ..... |
| Copper colorimetric | 0.91 | ..... |
| Copper electrolytic | 0.84 | ..... |
| Copper iodide (Sept. 6, 1912) | 0.85 | ..... |
| Gold | ..... | 0.005 |
| Silver | 0.02 | 4.9 |
| Zinc | 9.50 | ..... |
| $As_2O_3$ | 34.34 | ..... |
| Pb | 10.80 | ..... |
| $Sb_2O_3$ | 1.17 | ..... |
| $Bi_2O_3$ | 1.15 | ..... |
| $Na_2O$ | 0.49 | ..... |
| $K_2O$ | 0.42 | ..... |
| Carbon | 0.8 | (Method used |
| Tellurium | 0.28 | not accurate to |
| Selenium | Nil | hundredths.) |

Gold and Silver by Hunter. Tellurium and Selenium by Ware. Analysis by Dunn.

Sample taken for analysis was a mixed one of the dusts taken from 18 stations across the 120-ft. flue, 30.7 ft. below the base of stack. The dust was drawn out by means of a motor and suction fan, and was caught in asbestos bags. Evidences of the bags are seen in the analysis, particularly in $SiO_2$, Insol., MgO, $K_2O$ and $Na_2O$.

*Flue Dust Analysis—Totaled.*

| | *Per Cent.* |
|---|---|
| $SiO_2$ | 4.19 |
| $Al_2O_3$ | 3.14 |
| $Fe_2O_3$ | 2.78 |
| CaO | 0.50 |
| MgO | 0.69 |
| Free $SO_3$ | 3.60 |
| Combined $SO_3$ | 19.65 |
| Sulphur free and as sulphide | 1.88 |
| CuO electrolytic | 1.05 |
| ZnO | 11.84 |
| $As_2O_3$ | 34.34 |

| | | Per Cent. |
|---|---|---|
| PbO | | 11.63 |
| $Sl_2O_3$ | | 1.17 |
| $Bi_2O_3$ | | 1.15 |
| $Na_2O$ | | 0.49 |
| $K_2O$ | | 0.42 |
| Carbon | (Method used | 0.8 |
| $TeO_2$ | not accurate | 0.35 |
| $Ag_2O$ | to hundredths) | 0.02 |
| | | 99.69 |

Fine asbestos is plainly present (from the bags used for catching dust), vitiating the analysis somewhat.

Six months standing (between sampling and analysis) may also have increased the combined $SO_3$ at the expense of the free $SO_2$.

Tellurium may be present as $TeO_2$ in combination as tellurate, which would increase total slightly.

Our thanks are due to H. S. Ware, S. D. Hunter, and A. Austin (of the Washoe Laboratory force of 1911) for various analyses; and to J. O. Elton, H. H. Goe, and P. A. Haines (of the testing department and laboratory force of that time) for valued assistance in work on the flues.

As to commercial side of dust recovery, we append portion of report of Apr. 8, 1911, as follows:

"A partial analysis of the dusts taken during the determinations at the stack gives the following values (arithmetical averages being employed):

| | Ag Ounces | Au Ounces | Cu Per Cent. | $As_2O_3$ Per Cent. |
|---|---|---|---|---|
| East side flue | 4.04 | 0.0056 | 0.998 | 36.0 |
| West side flue | 4.83 | 0.0060 | 0.974 | 32.5 |
| Average, two sides | 4.44 | 0.0058 | 0.986 | 34.2 |

"Expressed in values per ton of dust the average figures give about $2.25 Ag; $0.12 Au; $2.40 Cu; or about $4.75 per ton for what one might call 'fixed values.' Figured to 77 tons per day (daily dust and fume emanating from stack) shows a daily loss from stack of $365.75, a figure which would hardly pay for smelting the refractory material, should it be recovered by any process. Add to smelting cost the initial cost of installation of dust and fume device, and the daily cost of labor and power for same, and the recovery of our solid stack emanations becomes

a matter outside the realms of commercial processes, and means a big charge against smoke expense.

"Nor will the recovery of the arsenic aid the matter materially, for this substance is what might be called a 'variable value.' At present the market is 'dull and hard to move' at $1.875 per 100 lb. white arsenic (*Engineering and Mining Journal*, Apr. 1, 1911). While this gives a figure of $37.50 per ton for $As_2O_3$ (with about 26 tons daily, or $975 daily value), yet the value of the $As_2O_3$ is a far different thing from the values of the other commercial elements. Its recovery means special treatment at the arsenic plant, with probable installation of bag house at the end of the kitchens before re-entrance of waste gases into main flue. And its recovery and entrance into an already dull and sagging market means breaking of present prices to a vanishing point, or at least to a point where the price would not pay for delivery. Consequently, that $975 per day loss of arsenic is a problematical loss; that is, one where recovery might mean an additional loss, and an added charge to smoke expense. Commercially, there is nothing in the line of dust and fume now emanating from the stack of the Washoe Smelter which would pay for recovery, treatment, and delivery of product.

"The efficiency of present flue system is shown by a screening test of the material recovered in our experiments. All of the 18 samples employed in analysis were screened through 40 mesh to eliminate (in the oversize) the asbestos from the bags used as filters. Within 10 min. the free $SO_3$ and $H_2SO_4$ in all samples chokes a screen of any size, but by brushing the dusts with a large camel's hair brush, everything except the asbestos was put through the 40 mesh. In some cases there was a trace remainder from the 400–500 g. of sample. A mixed sample of 200 g. was then made up from approximately equal amounts of all samples used in the analysis, and by means of the camel's hair brush, 100 g. of this composite was put through a 200-mesh screen. The oversize was astonishingly small—0.046 g., or about 0.05 per cent. If we add to this the trace obtained in some instances as oversize 40 mesh in the first screening the percentage of what one might call dust still remains below 0.1 per cent., giving about 99.9 per cent. fume, which would require widely different methods (from those of our present flue system) for recovery.

"The oversize on 200 mesh was too small for analysis, but a powerful glass betrayed the presence of siliceous particles and pyrite. A magnet showed a considerable percentage of magnetic material, probably roasted pyrite."

Later analyses checked the silver, gold, and arsenic figures, but showed the actual amount of copper present as shown by first results to be too high. The discovery of relatively high amounts of Zn and Pb aids conditions somewhat, though they are in such a finely divided state as to

require special methods of treatment for their smelting, if the stack loss could be recovered.

There can be no question that the installation of any device for catching the *fume* (*dust* being at present well settled) on the enormous scale required by the daily volume of stack emanation is not a commercial proposition; a large part of the cost of such installation and daily upkeep of same, being of necessity an additional charge against present smelting cost.

### *References on Pitot Tube and Calculations.*

1. Jager and Westby: Determination of the Velocity of Gas with the Pitot Tube. *Engineering and Mining Journal*, Vol. LXXXVIII, p. 468.
2. C. E. McQuigg: Pitot Tube in Gas Measurement, *Engineering and Mining Journal*, Vol. CXV, p. 649.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.
[SUBJECT TO REVISION].

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Butte meeting, Aug. 18 to 21, 1913, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1913, when Vol. XLVI. of the *Transactions* will go to press. Any discussion offered thereafter should preferably be in the form of a new paper for publication in Vol. XLVII. (with suitable cross references in both volumes).

# Some Recent American Progress in the Assay of Copper-Bullion.

BY EDWARD KELLER, PERTH AMBOY, N. J.

(Butte Meeting, August, 1913.)

## THE ASSAY FOR COPPER.

SOMEONE some time ago remarked that some chemists still insist on telling us how to determine copper by the electrolytic method. The writer must confess that he believes that everything is not known definitely as yet as to how the exact amount of copper is determined in such material as purest commercial electrolytic copper. Some of us are not yet convinced that the pure copper atom is at all times deposited from an acid solution and that no oxygen or hydrogen will, under certain conditions, accompany it. Difficulties with impurer material are frequent, but erroneous results are not always apparent, since these do not speak so readily for themselves as those in the first case. By this is meant that if a chemist finds 100 per cent. of copper in electrolytic copper it is pretty certain that he perceives the result to be wrong, while if he finds 0.1 per cent. of copper too much in a very impure material he is far more likely to be unconscious of his error. These, in a wide experience, are frequent happenings and the writer has always looked with interest to publications on this subject. In turn, he feels justified in giving to others a few of his own observations.

Up to a few years ago there were two methods of electrolytic copper assay in technical use, which had in common, that the amount of copper deposited on the cathode did not exceed 2 g., and they differed in that, for the one a large sample[1] (20 to 80 g.) was taken from the general sample by a splitting device, dissolved, and from the solution a small portion (to contain 1 or 2 g. of copper) measured out for the electrolytic deposition; in the other method the copper was determined, generally in 1-g. portions of the separated coarse and fine portions of the sample, all of which passed a 16 or 20-mesh screen, being accomplished by a 40-mesh screen,

[1] W. C. Ferguson gave a thorough description of the details and precautions necessary in this method. *Journal Industrial and Engineering Chemistry*, vol. II, No. 5, p. 187 (May, 1910).

and the average assay was figured according to the weight-ratio of the parts. To use larger quantities (5 g.) for electro-deposition on such copper as Lake or commercial electrolytic had been recommended some years ago,[1] but this had never been attempted in a systematic way with crude coppers.

When in 1909, John T. Stoddard[2] published a paper in which he pointed out the feasibility of rapid electro-deposition of metals, with analytical accuracy, on stationary gauze cathodes and without special stirring device for the electrolyte, it occurred to the writer that this idea, in properly modified form, could be taken up in the laboratories of the copper smelteries and refineries for the purpose of more accurate copper determinations in the metallic materials; however, not by reducing the time usually required for deposition, but by multiplying the quantity of sample taken for the assay. It was reasoned that gauze cathodes, for much work, would be too flimsy and changeable in weight through abrasion and that perforated platinum sheet would answer the purpose better. For certain work, cathodes with larger perforations had been recommended and used before by G. A. Heberlein[3] of Great Falls, Mont. It developed after a few trials that with the new perforated cathodes[4] 5 g. of copper, from crude-copper solution, could be as quickly and cohesively deposited as 1 g. on the smooth cathode, with equal gross surface, within the customary time of 18 to 20 hr. With the perforated cathode a current of 0.5 ampere was applied, while with the smooth cathode 0.15 ampere had been found to be the upper limit for good work. It was found impossible to deposit, in good form, 5 g. of copper on a smooth cathode within 24 hr. Most important with the 5-g. method is the fact, that the general limit of agreement between duplicate determinations was moved from the second percentage decimal to the third, whatever the absolute accuracy might be. This in itself is sufficient proof of the fairness of each of the 5-g. charges taken from the general sample. Besides, it is the writer's opinion that there is far more simplicity in this method than in the one in which a much larger sample is dissolved and an aliquot portion of the solution, with a small quantity of copper (1 to 2 g.), measured out.

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[1] George L. Heath, The Electrolytic Assay as Applied to Refined Copper. *Trans.* XXVII, p. 390 (1897).

[2] Rapid Electro-Analysis with Stationary Electrodes. *Journal American Chemical Society*, vol. XXXI, p. 385 (1909).

[3] Frank Klepetko, Discussion of Heath's Paper on the Electrolytic Assay, etc. *Trans.* XXVII, 967 (1897).

[4] NOTE: As no priority controversy is desired, the writer would state that he placed his order for perforated cathodes with Baker & Co., Inc., Newark, N. J., on March 20, 1909, and that he exhibited them before a meeting of the New York Section of the American Chemical Society on March 11, 1910.

The operation of this method for Anaconda copper, is now carried out in the following manner:

*Stock solutions:*—Nitric acid (sp. gr. 1.42) one part to one part of water; sulphuric acid (sp. gr. 1.84) one part to two and one-half parts of water; sodium chloride, 5.5 g. to the liter. The proper ratio of the fine portion of the sample, $\frac{5}{\frac{C}{F}+1}$ grams (C=weight of coarse; F=weight of fines in grams), is weighed and the remaining part of the 5 g. made up with the coarse portion. This is dropped into a 300-cc. lipless Jena beaker (tall form) which seems to be the smallest size that can safely be used for the solution of 5 g. of copper. Tests made by placing a filter paper over cover glass and beaker and snugly folded down over the edge of the latter

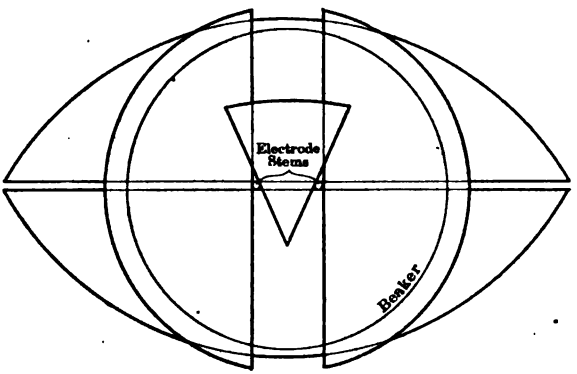


FIG. 1.—SYSTEM OF COVERING ELECTROLYTIC BEAKER.

revealed no loss of copper mechanically carried away. Ten cubic centimeters of stock sulphuric acid is first added to the copper, followed by 25 cc. of stock nitric acid; the purpose of the initial sulphuric acid is to retard the action of the nitric acid but afterwards to aid in the solution of the oxides present. The beaker, covered with a watch glass, is placed in a cool part of the hood until the copper is dissolved and is then placed on a steam plate, carefully avoiding boiling, until complete disappearance of nitrous fumes; 2 cc. of the sodium chloride solution is now added and the hot solution gently shaken until the silver chloride is well coagulated. Filtration is then performed through a 7-cm. filter into a beaker of the same size and shape as the original. To make up the full acidity of the electrolyte 30 cc. more of the stock sulphuric acid and 3 cc. of the nitric is necessary, all of which is run through the filter to insure complete recovery of the copper. The whole is diluted to 225 cc. with distilled water.

When any difficulty in depositing the copper is experienced it may be attributed to insufficient dilution of the electrolyte; addition of more water will overcome it.

The perforated platinum sheet constituting the closed cathode cylinder is 6 x 11 cm. and has seven perforations of about 0.5 mm. to the linear inch. The total height of the cylinder and stem is 23 cm.; they are somewhat unnecessarily heavy, weighing about 23 g. Those cylinders that have been in daily use for four years on Anaconda material lost on the average 1.7 milligrams per year. Others, which are used for copper high

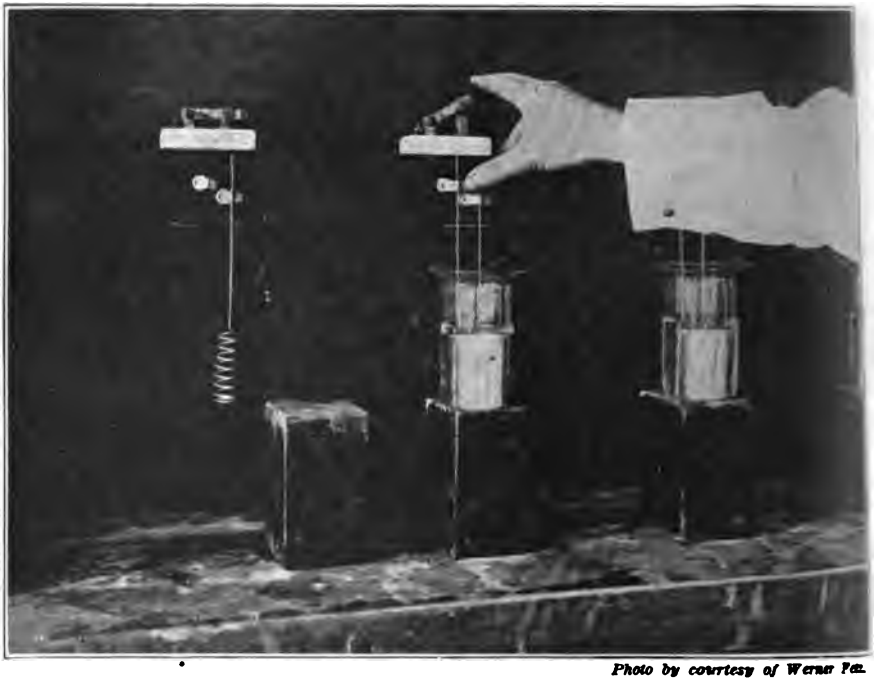

*Photo by courtesy of Werner Fax.*

FIG. 2.—ELECTROLYTIC COPPER DEPOSITION STAND.

in gold, gain very appreciably in weight after every deposition; this being due to the deposition, with the copper, of that portion of the gold which is soluble in nitric acid. In course of time these cylinders become gold-plated. With the heavy current used, it is important that the beakers are well covered, so that there can be no loss of solution by spraying. Fig. 1 illustrates the scheme to accomplish this with the aid of several sets of split watch glasses. Fig. 2 shows the construction of an electrolytic stand with series connections. The ammeter is not visible. It is here shown chiefly to call attention to the knife switches on the top of the bar which

enable the operator to remove the cathode and simultaneously to close the circuit, which is not possible with the generally used plug. It is of the utmost importance to transfer the cathode with its copper deposit as quickly as possible from the electrolyte into pure water, so as to lose a minimum of copper by re-solution. It is done as follows: The covers are first removed from the beaker, then the base block from under the latter with the right hand, holding the beaker with the left. Up to this moment the current passes uninterruptedly. Now the forefinger of the right hand closes the switch and simultaneously the thumb presses the spring releasing the cathode; the left hand at the same time draws the beaker and cathode down and away from the anode (the usual corkscrew spiral). By this time the right hand is free to transfer the cathode to several clean waters and to absolute alcohol, after which it is dried over a Bunsen burner.

As already stated, the current employed for each assay is 0.5 ampere, the time (over night) for complete deposition about 18 hours. The latter point is arrived at when polarization (bubbles) at the cathode has taken place for about one-half hour. The beaker is then filled with distilled water, so that the electrolyte touches the cover-glasses and the current permitted to continue for another half-hour. Each electrolyte is tested for the presence of copper by saturation with hydrogen-sulphide gas.

The question of the accuracy of results should in all cases be subjected to analysis. It has already been hinted that after all the copper has been deposited on the cathode, there is a possibility of re-solution when removing it from the acid electrolyte. Tests made on numerous individual determinations showed that the electrolytes contained 0.0092 per cent. of the original copper. Table I shows, on the other hand, that

TABLE I.—*Impurities Contained in Copper-Bullion and Deposited with Electrolytic Copper.* (*Analysis of 100 Grams.*)

| | Lead, Per Cent. | Bismuth, Per Cent. | Antimony, Per Cent. | Arsenic, Per Cent. | Tellurium and Selenium, Per Cent. |
|---|---|---|---|---|---|
| Impurities in converter copper | 0.0103 | 0.0040 | 0.0630 | 0.0211 | 0.0072 |
| Impurities in electrolytic copper | trace | 0.0037 | 0.0031 | 0.0009 | 0.0016 |
| Portion of impurities deposited [a] | ? | 92.50<br>[b] 100.00 ? | 4.92 | 4.26 | 22.22 |

[a] Gold in this case was not dissolved and silver is completely eliminated.

[b] Bismuth is probably all deposited, the difference between the two figures of analysis being well within the limits of experimental accuracy.

the copper which is deposited is not entirely pure, that in this case it contained 0.0093 per cent. of its weight of impurities deposited from the electrolyte. These two figures balance each other very closely and in commercial work usually no corrections are made.

In many laboratories it has been customary to precipitate the silver electrolytically with the copper on the cathode and then to subtract from the gross percentage that of the silver as determined by separate assay. This works well up to a certain silver content; 0.3 per cent. or 100 oz. per ton probably being the limit. When under this method the first difficulties with a good copper deposit are experienced, it is well to eliminate the silver from the electrolyte. On a number of occasions this was the only precaution necessary to bring about an excellent deposit of electrolytic copper.

When certain impurities in crude copper, or copper-bullion, exceed a certain quantity, it becomes impossible to electro-deposit from its solution a copper of sufficient purity for commercial analytical requirement. We have already seen that bismuth is all deposited with the copper, but this metal is rarely met in disturbing quantity in converter copper. When met in larger quantity in black coppers it must be eliminated before the electro-deposition of the copper; a method for which will be described further on.

Selenium is an element which is more readily deposited from solution than copper and we meet it in appreciable quantity in the converter coppers from our Southwest. When the copper on the cathode is dissolved in dilute nitric acid the selenium remains, at least in part, as a red coating, and some chemists weigh this and use the weight as a minus correction in their copper assay. This, no doubt, is but an approximation. The following is a very simple method to remove quantitatively the selenium from any quantity of copper, the silver being removed at the same time.

Selenium and copper cannot be completely separated by the alkaline sulphide method, as there is always some insoluble selenide formed with the heavy metal. Selenium is also precipitated with the copper when the latter is separated as sulphocyanate from a number of other elements. In fact, however, any method that would necessitate the precipitation of 5 g. of copper as a chemical compound for the purpose of its purification would be unsatisfactory. The precipitation of the selenium from the electrolyte by means of sulphur dioxide naturally suggested itself, but it was soon found that here also the two elements combined to form copper selenide.

Upon further investigation it was learned that in the presence of copper and silver, in sulphate solution, the selenium, when liberated by sulphur dioxide, had a strong preferential affinity for silver and if the latter was

present, in sufficient excess, the selenium would be precipitated quantitatively as silver selenide, $Ag_2Se$, without a trace of copper. The excess of silver is then precipitated as chloride and with the silver selenide is filtered and washed with dilute sulphuric acid and water. In case too much of an excess of salt solution has been used, it is best now to evaporate to sulphuric acid fumes, in order to drive off the chlorhydric acid and thus avoid its deleterious effect on the character of the copper deposit.

The presence of an appreciable quantity of tellurium will also interfere with the accuracy of the electrolytic copper determination and it must, accordingly, be eliminated before the electro-deposition of the copper. However, it is not often present in disturbing amount. Tellurium is not very readily precipitated from the copper sulphate solution by sulphur dioxide, but whatever portion of it is reduced combines with copper to form copper telluride. It is no rival of selenium in the latter's affinity for silver. A complete elimination method will be described with that of the bismuth.

Antimony and arsenic are not very readily electro-deposited from nitric-acid solution and only small percentages of the original quantities are found in the electrolytic copper; yet they become troublesome at times, because they are generally the elements found in largest quantity associated with copper. It has been claimed that certain compounds, "dopes,"[1] added to the electrolyte will entirely prevent the deposition of these elements. In tests made in the Anaconda laboratory with commercial electrolytic copper it was found with the addition of such dopes the results were very noticeably high.

To eliminate antimony and arsenic when they become disturbing factors, the writer prefers the double-deposition method, which has proved to be very satisfactory and in proof of which the data of Table II are illustrative. To execute this method the original deposit on the cathode is treated with the usual precautions and is then placed, together with the anode, in a regular depositing beaker, the complete electrolyte added and the whole covered as per scheme of Fig. 1. Only gentle heat is applied to insure the re-solution of the deposit and the solution is digested long enough to expel the reduction products of the nitric acid. When, in these cases, the copper deposit has a slight coating of arsenic or antimony, there is no re-solution of the copper during the time of removal of the cathode-cylinder from the electrolyte, since the former elements are first subject to attack.

Lead is one more element which is usually present in copper in small quantity, but which, when present even in larger quantity, may be readily eliminated in the course of the regular routine of preparing the electro-

[1] George A. Guess. The Electrolytic Assay of Lead and Copper. *Trans.* XXXVI. 605 (1905).

analysis. In the first place a major part of it may be separated as sulphate from sulphate solution and the small remaining part can be electro-deposited from nitric acid solution as peroxide on the anode; it being only necessary that the latter presents sufficient surface for the adhesion of the peroxide.

There is yet to be described a method by which bismuth and tellurium may be eliminated and which will, at the same time, include the elements for which individual methods have been given. For this the writer be-

TABLE II.—*Arsenic in Original Copper-Bullion and in Copper of First and Second Electrolytic Deposition.*

Arsenic in Original Bullion, 2.609 per cent.

| | Average, Per Cent. | Arsenic in First Deposit, Per Cent. |
|---|---|---|
| First Electrolytic Deposits, Copper, Per Cent.: | | |
| 94.632, 94.678, 94.682..... | 94.671 | 0.0534; 2.05 of original |
| 94.652, 94.690, 94.692..... | | |
| First Electrolytic Deposit, Copper Dissolved for Second Deposition, Per Cent.: | | |
| 94.594, 94.626, 94.576..... | 94.6698 | |
| 94.590, 94.652, 94.666..... | | |
| 94.652, 94.594, 94.724..... | | |
| 94.824, 94.770, 94.770..... | | |
| Second Electrolytic Deposit, Copper, Per Cent.: | | Arsenic in Second Deposit, Per Cent. |
| 94.580, 94.550, 94.526..... | 94.5607 | 0.00146; 0.056 of original. 2.73 of first deposit. |
| 94.528, 94.558, 94.544..... | | |
| 94.596, 94.572, 94.594..... | | |
| 94.600, 94.526, 94.554..... | | |

lieves the simplest process to be the precipitation with ferric hydroxide in slightly ammoniacal solution with an addition of ammonium carbonate to precipitate the bismuth. To carry out this method the copper is dissolved and the silver eliminated as usual, 0.1 g. of ferrous sulphate having been added to the 5-g. copper charge. The solution is made slightly ammoniacal amd a little ammonium carbonate added; it should then be kept near the boiling point for about 15 minutes. The ferric hydroxide carries down with it all of the selenium, tellurium, antimony and arsenic, but unfortunately, and this is the drawback of the method, it also carries with it an appreciable quantity of copper, so that after filtering and washing the precipitate with ammoniacal water it must be re-dissolved, re-precipitated, filtered and washed, and the whole procedure repeated four or five times to insure the recovery of all of the

copper. In each re-precipitation the same conditions are to be observed as in the first one. All the filtrates are united and the regular quantity of sulphuric acid is added. Should the volume of solution be too great it must be evaporated to the desired mark.

The marked progress in rapid electro-deposition of metals in recent years, by means of high current density and the use of rotating electrodes or the rapid circulation of the electrolyte, is well known. These methods, undoubtedly, have their high merit for special purposes, where time is the chief factor. They do not seem to have appealed to the chemists of copper works where large numbers of accurate determinations are required and in which the work of the electric current during the night requires no attention whatever, while rotating electrodes, motors, and violently stirred electrolytes would be sources of much concern.

## The Assay for Silver and Gold.

From time immemorial the methods of assaying any materials for silver and gold were in reality nothing but laboratory smelting methods. When in 1885, the writer arrived in Butte, Mont., he found that these metals, contained in copper bullion, were still determined by the all-fire assay, both in the same charge. There seems to be no record as to when and where the so-called combination method (dissolving the copper in nitric acid and precipitating the silver as chloride, etc.) was first introduced; but in the early nineties of the past century it became evident that for the sake of accuracy of the gold assay this metal must be determined by the all-fire method which, on the other hand, was quite unsatisfactory for silver. The combination method became the standard method for silver and the all-fire the standard for gold. From that time on probably most assayers were longing for a reliable single method for both metals, since the simplicity of the combination method, with its accurate results for silver, contrasted strongly with the unchemical, tedious and expensive all-fire method for gold.

To many, the comparative cost of the all-fire and sulphuric-acid assay-methods may be of interest. For the Anaconda laboratory, of Perth Amboy, it has been calculated with accuracy, that the materials per assay, exclusive of those used in the parting for gold and for the correction-assay (these being approximately the same for both methods), amount to 8.5 cents for the sulphuric-acid method and 75.5 cents for the all-fire method; a saving of 67 cents per assay. The saving in labor, too, is considerable, although its value in our case has not been established.

The first recorded attempt to modify the combination method in a way to obtain accurate gold results was made by L. D. Godshall,[1] who

[1] Assay of Copper Materials for Silver and Gold. *Trans.* XXX., 529 (1900).

suggested to reprecipitate any gold that might have gone into solution[1] with the copper by conducting hydrogen-sulphide gas into the solution in sufficient quantity to precipitate all the silver with no copper or only a minimum quantity thereof. A. R. Ledoux and Cabell Whitehead[2] pointed out the defects of such a method. About the same time W. R Van Liew[3] showed that accurate gold results could be obtained by modifying the combination method in the way that dilute nitric acid was added in several portions to the metallic copper and the reaction conducted under ice-cooling. The time required for this method was a factor very much against its becoming popular. A new idea was introduced by Thomas B. Swift,[4] who first amalgamated the surface of the copper by means of mercuric nitrate before dissolving the metal with nitric acid. In this method, the surface of each copper particle would remain coated with mercury during the whole process of solution and the mercury would absorb the gold and protect it from any solvent action. The author gave no data regarding silver.

During the period in which the afore-described efforts for a simplified gold assay were made, attempts along the same line were also carried on in the Anaconda laboratory. The idea was to find a reducing medium which would quickly precipitate the gold from the nitric acid solution without precipitating copper in any form. Good results were obtained by boiling the copper nitrate solution (one assay ton of copper), which contained little nitric acid in excess and no nitrous products, with 25 cc. of formaldehyde, strength about 37 per cent. A few comparative results by methods of that time are given in Table III.

A peculiar phenomenon was observed in trying cane sugar for the same purpose; instead of reducing and precipitating the gold which had been dissolved by the nitric acid, the greater portion of the total gold would be dissolved when the solution was boiled; Table IV gives the data.

It was at first thought that chlorine was present in the sugar and that it was responsible for the solution of the gold, but a test of the sugar solution as well as of the burnt residue revealed but an indistinct trace of that element, so that the following facts may satisfactorily explain the phenomenon. When the copper nitrate solution is boiled with the addition of sugar the latter is quantitatively converted into oxalic acid and this forms, with the copper, an insoluble oxalate. The nitric acid

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[1] Edward Keller, The Solubility in Nitric Acid of Gold Contained in Copper Bullion, *Trans.* XLIII., 582 (1912).

[2] Discussion of L. D. Godshall's paper, *Trans.* XXX., 1121 (1900).

[3] Losses in the Determination of Gold and Silver in Copper Bullion, etc. *Engineering and Mining Journal*, vol. LXIX., No. 16, p. 469 (April 21, 1900).

[4] The Assay of Copper Bullion, *Engineering and Mining Journal*, vol. LXXIV., No. 20, p. 650 (Nov. 15, 1902).

TABLE III.—*Comparative Gold Assays by Various Methods.*

*Gold, milligrams or ounces per ton.*

| SAMPLE NO. | All-Fire Method | Swift's Method Hg = 0.75g. | Combination Method | Combination Method, Formaldehyde = 25cc. |
|---|---|---|---|---|
| 1 | 5.82 | 5.88 | 5.70 | 5.82 |
| 2 | 6.03 | 6.08 | 5.88 | 6.02 |
| 3 | 5.78 | 5.80 | 5.58 | 5.78 |
| 4 | 5.78 | 5.81 | 5.60 | 5.79 |
| 5 | 5.66 | 5.68 | 5.51 | 5.64 |
| 6 | 5.56 | 5.60 | 5.48 | 5.55 |
| 7 | 5.41 | 5.50 | 5.42 | 5.45 |
| 8 | 5.63 | 5.66 | 5.52 | 5.71 |
| 9 | 5.44 | 5.56 | 5.32 | 5.47 |
| AVERAGE | 5.6789 | 5.7300 | 5.5567 | 5.6922 |

is reduced to free nitrous acid and this will, the oxalic acid being inert, dissolve the gold.

Experiments on these lines were discontinued when, early in 1908, it was learned that Frederic F. Hunt,[1] of New York, had devised a new combination method which yielded good results for silver and for gold. This method consisted in adding to 1 assay ton of copper 10 cc. of water and 100 cc. of sulphuric acid of 1.84 specific gravity; first heating on a hot-plate and finally on an open flame until complete sulphatizing of the copper was accomplished. There was no solution of the gold and when the silver, or a part of it, was dissolved it was precipitated as chloride and the remaining procedure was the same as that well known in the old combination method. Some difficulty was experienced in sulphatizing the copper to completeness when the sample was too coarse and when

TABLE IV.—*Gold-Solubility in Copper-Nitrate Solution in the Presence of Cane Sugar.*

(Gold present = 4.16 mg.)

| Grams, sugar | 1 | 2 | 3 | 4 | 5 |
|---|---|---|---|---|---|
| Gold after boiling, mg | 0.69 | 0.31 | 0.30 | 0.39 | 0.33 |

the copper contained appreciable amounts of sulphur. A little later in the same year Albert M. Smoot,[2] of Ledoux & Co., New York, informed the writer that copper first amalgamated superficially and heated with

[1] Private communication; since published with modification, *Engineering and Mining Journal*, vol. LXXXVII., No. 9, p. 465 (Feb. 27, 1909).

[2] Private communication.

80 cc. of sulphuric acid (sp. gr. 1.84) per assay ton of copper was an improvement over Hunt's method. Difficulties in the complete sulphatizing of the copper in certain bullions continued and led to numerous tests in the Anaconda laboratory which resulted in the final adoption of the following routine, by which even coarse drillings and copper shot may be completely sulphatized and of which the resulting gold and silver chloride may be scorified with a total of 20 g. of lead in 2-in. scorifiers and cupelled in 1-in. cupels; the lead buttons weighing 5 g.

*Stock solutions.*—Mercuric nitrate, 25 g. of mercury per liter; sulphuric acid, sp. gr. 1.84; sodium chloride, 19 g. per liter (10 cc. will precipitate 350 mg. of silver). The copper bullion drilling sample has been ground to pass a 16-mesh screen and if by test it has been found that the coarse and fine parts, separated by a 40-mesh screen, differ appreciably in precious-metal content the parts are weighed separately in their proper ratio. The fine, $\frac{29.166}{\frac{C}{F}+1}$ grams (C = weight of coarse; F = weight of fine portion of sample), is weighed first and the remainder of the assay ton made up with the coarse. This is now placed in an 800-cc. Jena beaker, Griffin shape, 30 cc. of water and then 10 cc. of the mercuric-nitrate solution added (Hg = 0.25 g.). The beaker is shaken until all the copper appears amalgamated over its surface, then 100 cc. of sulphuric acid is added; the beaker is covered with a 5-in. watch glass and placed on an electric hot-plate. The time necessary to complete the reaction depends on the state of division of the sample, as also on the temperature of the plate. Table V. gives the time and temperature record of two samples.

For about one hour the liquid appears to boil, which, however, is only a bubbling, due to the evolution of sulphur-dioxide gas, from the reduction of the sulphuric acid and the oxidation of the copper. This completed, the supernatant liquid assumes a very dark green color, finally changing to a light grayish blue which, as experience has taught, is the indication of the finishing point. Boiling over an open flame has been abandoned, the excessive heat having been found unnecessary and the act of boiling being bound to entail some loss. The beaker is next removed from the plate and placed to cool on an asbestos sheet. Complete cooling is unnecessary as there is very little free sulphuric acid present. Four hundred and fifty cubic centimeters of water with the necessary amount of sodium chloride solution, the latter depending on both the amount of the silver and the amount of mercury present, are used. With 80 to 100 mg. of silver and 0.25 g. of mercury for Anaconda material 30 cc. of the stock solution or 0.57 g. of sodium chloride is always a safe quantity. The beaker is again placed on the hot-plate and the solution brought to a

TABLE V.—*Time and Temperature-Record of Sulphatizing Amalgamated Copper-Bullion Samples (One Assay Ton).*

| Sample I | | | Sample II | | |
|---|---|---|---|---|---|
| Time, p. m. | Temperature, F. | C. | Time, p. m. | Temperature, F. | C. |
| Start.....2.35 | 302 | 150 | Start.....1.50 | 224 | 107 |
| 2.38 | 406 | 208 | 1.57 | 336 | 169 |
| 2.45 | 412 | 211 | 2.05 | 350 | 177 |
| 3.05 | 412 | 211 | 2.15 | 358 | 181 |
| 3.25 | 378 | 192 | 2.25 | 364 | 184 |
| 3.45 | 386 | 207 | 2.35 | 378 | 192 |
| 3.55 | 384 | 206 | 2.45 | 374 | 190 |
| 4.05 | 386 | 207 | 2.55 | 389 | 198 |
| 4.15 | 408 | 209 | 3.05 | 390 | 199 |
| 4.25 | 415 | 213 | 3.15 | 386 | 197 |
| Finish.....4.35 | | | 3.25 | 389 | 198 |
| | | | Finish.....3.27 | | |
| Total time 2.00 | | | 1.37 | | |

boil, which dissolves the copper sulphate and coagulates the silver chloride. On removal of the beaker 150 cc. more water is added, the total amount now being 600 cc., which is capable of keeping in solution all copper sulphate after cooling. It is then a matter of expediency to filter immediately or the day after.

Details as to the furnace operations of this method are not deemed to be within the scope of this paper; they are much a matter of individual taste. For the description of mechanical and labor-saving devices the reader is referred elsewhere.[1]

It should be noted that with comparatively pure copper the amount of mercuric nitrate may be reduced, while with copper high in sulphur content an increase in the amount of the mercuric nitrate will be required.

It has been noted as a peculiarity of this method that with certain low-grade bullions none of the silver is rendered soluble and that upon addition of sodium chloride solution no silver chloride is formed. Table VI. gives a variety of data on this subject. It is shown that all of the arsenic and nearly all of the antimony is dissolved. It was also qualitatively proven that no selenium or tellurium remains in the residue. The insoluble silver, therefore, is not combined with any of these elements.

It was found that with all of the insoluble silver residues there remained some mercury and with sample No. 4 always a little copper; the latter probably being in the form of sulphide, as this sample contained

[1] Edward Keller, Labor-Saving Devices in the Works Laboratory, *Trans.*, XXXVI., 3 (1906); XLI., 786 (1910).

TABLE VI.—*Soluble and Insoluble Silver by Sulphuric-Acid Assay-Method: with and without Mercury.*

| SAMPLE | 1 | | | 2 | | | 3 | | | 4 | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Soluble | Insoluble | Total | Soluble | Insoluble | Total | Soluble | Insoluble | Total | Soluble | Insoluble | Total |
| Silver,[a] oz. per ton | 579.33 | 131.30 | 710.63 | 84.33 | 129.83 | 214.16 | 53.66 | 35.95 | 89.61 | 6.90 | 28.39 | 35.29 |
| Arsenic, per cent | 2.506 | 0.00 | 2.506 | 0.0444 | 0.00 | 0.0444 | 0.0569 | 0.00 | 0.0569 | 0.1558 | 0.0 | 0.1558 |
| Antimony, per cent | 0.0885 | 0.0060 | 0.0945 | 0.1600 | 0.0054 | 0.1654 | 0.0355 | 0.0060 | 0.0415 | 0.0344 | 0.0062 | 0.0406 |
| Mercuric Nitrate, 20 cc.; $H_2SO_4$, 100 cc. | | | | | | | | | | | | |
| Silver, oz. per ton | 477.14 | 236.01 | 713.15 | 164.62 | 48.47 | 213.09 | 71.69 | 17.46 | 89.15 | 23.83 | 10.85 | 34.68 |
| Mercuric Nitrate, 10 cc.; $H_2SO_4$ 100 cc.; heated 18 hr.; 15 min. over open flame | | | | | | | | | | | | |
| Silver, oz. per ton | 621.50 | 89.35 | 710.85 | ..... | ..... | ..... | 86.19 | 3.88 | 90.07 | 23.30 | 11.60 | 34.90 |
| No Mercuric Nitrate; $H_2SO_4$ 100 cc.; heated 18 hr.; 15 min. over open flame | | | | | | | | | | | | |
| Silver, oz. per ton | 475.04 | 235.97 | 711.01 | 209.28 | 6.44 | 215.72 | 1.22 | 88.75 | 89.97 | 0.38 | 35.18 | 35.56 |
| Mercuric Nitrate, 10 cc.; $H_2SO_4$ 200 cc.; heated 18 hr.; 1 hr. over open flame | | | | | | | | | | | | |
| Silver, oz. per ton | 703.82 | 8.86 | 712.68 | [b]209.30 | 6.36 | 215.66 | 88.62 | 1.42 | 90.04 | [b]37.60 | 0.48 | 38.08 |
| No Mercuric Nitrate; $H_2SO_4$ 200 cc.; heated 18 hr.; 1 hr. over open flame | | | | | | | | | | | | |
| Silver, oz. per ton | 285.66 | 425.98 | 711.64 | 5.98 | 211.12 | 217.10 | 1.92 | 88.38 | 90.30 | 0.18 | 37.90 | 38.08 |

[a] Assayed by our regular sulphuric acid method; 30 cc. water, 10 cc. mercuric nitrate solution, 100 cc. $H_2SO_4$ (sp. gr. 184). The volume of water used in all of the above assays was constant, 30 cc.

[b] Nos. 2 and 4 samples changed.

TABLE VII.—*Gold and Silver in Copper-Bullion by the Sulphuric-Acid Method. With Varying Amounts of Mercury for the Amalgamation of the Copper.*

*Weight, Milligrams or Ounces per Ton.*

Volume of mercuric-nitrate solution and water constant (35 cc.).

| | | | | | | | |
|---|---|---|---|---|---|---|---|
| Mercury, grams | None | 0.125 | 0.250 | 0.500 | 1 | 2 | 4 |
| Water, cc | 35 | 30 | 25 | 15 | 25 | 25 | 25 |
| $H_2SO_4$ (1.84), cc | 100 | 100 | 100 | 100 | 100 | 100 | 100 |
| [a]Salt solution, cc | 12 | 12 | 12 | 15 | 40 | 80 | 120 |
| | 66.68 | 67.06 | 67.20 | 66.66 | 67.16 | 67.22 | 66.96 |
| | 66.96 | 67.16 | 67.20 | 66.54 | 67.32 | 67.18 | 66.82 |
| | 67.16 | 67.22 | 66.92 | 66.48 | 67.16 | 67.22 | 66.92 |
| | 67.32 | 67.02 | 66.96 | 66.72 | 67.42 | 67.02 | 66.90 |
| | 66.88 | 67.04 | 67.34 | 66.76 | 67.14 | 67.00 | 66.92 |
| | 66.50 | 66.78 | 66.98 | 66.76 | 67.24 | 67.06 | 66.92 |
| | 66.64 | 66.94 | 67.26 | 67.24 | 67.18 | 67.20 | 67.10 |
| | 66.90 | 66.92 | 67.12 | 66.83 | 67.64 | 66.98 | 67.14 |
| | 66.72 | 67.06 | 66.96 | 66.72 | 67.32 | 67.36 | 67.12 |
| | 66.50 | 67.00 | 67.16 | 66.74 | 67.02 | 67.20 | 66.80 |
| | 66.70 | 67.32 | 67.26 | 66.10 | 67.36 | 67.14 | |
| | 66.98 | 67.02 | 67.18 | 66.58 | 67.22 | 67.04 | |
| | 67.14 | 66.84 | 67.28 | 66.66 | 67.16 | 67.20 | |
| | 66.84 | 66.90 | 67.14 | 66.76 | 67.42 | 67.34 | |
| | 66.76 | 66.68 | 67.18 | 66.84 | 67.50 | 67.30 | |
| | 67.06 | 66.96 | 67.56 | 66.78 | 67.30 | 67.34 | |
| | 67.20 | 66.86 | 67.42 | 66.70 | 67.42 | 67.14 | |
| | 66.94 | 66.76 | 67.18 | 66.58 | 67.36 | 67.18 | |
| | 67.26 | 67.26 | 67.04 | 66.64 | 67.38 | 67.34 | |
| | 67.38 | 66.68 | 66.68 | 66.72 | 67.36 | 66.82 | |
| Average, silver-gold | 66.926 | 66.974 | 67.151 | 66.690 | 67.304 | 67.164 | 66.960 |
| Less gold | 0.476 | 0.476 | 0.479 | 0.479 | 0.483 | 0.476 | 0.483 |
| Silver | 66.450 | 66.498 | 66.672 | 66.211 | 66.821 | 66.688 | 66.477 |
| Silver correction | 0.906 | 1.128 | 0.999 | 1.222 | 1.009 | 1.069 | 1.130 |
| Corrected silver | 67.356 | 67.626 | 67.671 | 67.433 | 67.830 | 67.757 | 67.607 |

GOLD ASSAYS.

| | | | | | | | |
|---|---|---|---|---|---|---|---|
| Uncorrected | 0.4760 | 0.4765 | 0.4785 | 0.4795 | 0.4830 | 0.4755 | 0.4830 |
| Correction | 0.0010 | 0.0015 | 0.0020 | 0.0015 | 0.0010 | 0.0010 | 0.0010 |
| Corrected gold | 0.4770 | 0.4780 | 0.4805 | 0.4810 | 0.4840 | 0.4765 | 0.4840 |

[a] One cubic centimeter of salt solution contained 0.019 *gr.* sodium chloride.

by far the most of that element. We may conclude that, although the presence of mercury facilitates the solution of the silver as it does that of the copper, all copper and all mercury must be dissolved before all silver can go into solution. In the ordinary process of the assay, there is not enough sulphuric acid present and the heating is not continued sufficiently long to meet these conditions; therefore, with little silver present none is dissolved, and with much silver present only a portion is rendered soluble.

Table VII. demonstrates that an increased amount of mercury has no marked effect on the silver results if only the necessary increase in the amount of sodium chloride be provided. Neither of the two substances can influence the gold results.

Table VIII. demonstrates that immediate filtration of the silver chloride may be chosen for ordinary purposes, while when great accuracy is desired it is safer to leave it for settling and complete cooling over night.

TABLE VIII.—*Experiment on Sulphuric-Acid Method Silver Determinations. Filtration Test.*

*Milligrams or ounces per ton.*

| Sample No. | Filtered Next Day | Filtered At Once | Difference | | Sample No. | Filtered Next Day | Filtered At Once | Difference | |
|---|---|---|---|---|---|---|---|---|---|
| | | | Over | Under | | | | Over | Under |
| 1 | 84.92 | 84.89 | 0.03 | ..... | 19 | 87.13 | 87.18 | ..... | 0.05 |
| 2 | 81.87 | 81.79 | 0.08 | ..... | 20 | 85.44 | 85.41 | 0.03 | ..... |
| 3 | 80.28 | 79.97 | 0.31 | ..... | 21 | 84.39 | 84.66 | ..... | 0.27 |
| 4 | 84.46 | 84.13 | 0.33 | ..... | 22 | 87.82 | 87.63 | 0.19 | ..... |
| 5 | 82.91 | 82.73 | 0.18 | ..... | 23 | 85.35 | 85.00 | 0.35 | ..... |
| 6 | 82.86 | 82.60 | 0.26 | ..... | 24 | 90.01 | 90.07 | ..... | 0.06 |
| 7 | 90.01 | 90.06 | ..... | 0.05 | 25 | 87.39 | 87.06 | 0.33 | ..... |
| 8 | 83.97 | 83.78 | 0.19 | ..... | 26 | 86.72 | 86.43 | 0.29 | ..... |
| 9 | 82.46 | 82.40 | 0.06 | ..... | 27 | 90.81 | 90.59 | 0.22 | ..... |
| 10 | 90.62 | 90.72 | ..... | 0.10 | 28 | 85.62 | 85.25 | 0.37 | ..... |
| 11 | 89.10 | 88.63 | 0.47 | ..... | 29 | 84.04 | 84.51 | ..... | 0.47 |
| 12 | 87.61 | 87.50 | 0.11 | ..... | 30 | 86.14 | 85.83 | 0.31 | ..... |
| 13 | 87.42 | 87.56 | ..... | 0.14 | 31 | 89.15 | 89.39 | ..... | 0.24 |
| 14 | 84.76 | 84.50 | 0.26 | ..... | 32 | 85.86 | 85.59 | 0.27 | ..... |
| 15 | 84.12 | 84.30 | ..... | 0.18 | 33 | 86.28 | 85.91 | 0.37 | ..... |
| 16 | 86.40 | 85.83 | 0.57 | ..... | 34 | 87.22 | 86.77 | 0.45 | ..... |
| 17 | 85.14 | 85.09 | 0.05 | ..... | 35 | 86.51 | 86.34 | 0.17 | ..... |
| 18 | 86.95 | 86.54 | 0.41 | ..... | | | | | |
| | | Average..... | | | 86.0497 | 85.9040 | | 0.1903 | 0.0446 |
| | | Difference..... | | | 0.1457 | | | 0.1457 | |

Table IX. gives comparative results for silver and gold; the former by two and the latter by four methods.

In Table X. are given probably somewhat more convincing, comparative figures for gold as determined by the once standard all-fire method

TABLE IX.—*Comparison of Several Methods for Assaying Copper-Bullion.*

*Milligrams or ounces per ton.*

| SILVER | | | GOLD | | | | |
|---|---|---|---|---|---|---|---|
| COMBINATION | | | COMBINATION | | | | |
| Sample No. | Nitric-Acid Method | Sulphuric-Acid Method | Nitric-Acid Method | | Sulphuric-Acid Method | | All-Fire Method 0.1 Assay Ton |
| | Water | Mercury | Water | Mercury | Water | Mercury | |
| 1 | 78.3 | 78.4 | 0.22 | 0.26 | 0.26 | 0.28 | 0.27 |
| 2 | 71.8 | 71.9 | 0.27 | 0.30 | 0.30 | 0.31 | 0.30 |
| 3 | 73.7 | 73.7 | 0.28 | 0.31 | 0.30 | 0.30 | 0.31 |
| 4 | 72.6 | 72.9 | 0.28 | 0.31 | 0.31 | 0.32 | 0.33 |
| 5 | 71.9 | 71.8 | 0.28 | 0.30 | 0.29 | 0.30 | 0.31 |
| 6 | 72.7 | 72.9 | 0.27 | 0.30 | 0.30 | 0.30 | 0.31 |
| 7 | 80.8 | 80.7 | 0.30 | 0.34 | 0.34 | 0.33 | 0.35 |
| 8 | 73.7 | 73.7 | 0.28 | 0.30 | 0.31 | 0.32 | 0.31 |
| 9 | 81.7 | 81.7 | 0.31 | 0.32 | 0.34 | 0.34 | 0.35 |
| 10 | 85.1 | 85.3 | 0.27 | 0.30 | 0.30 | 0.30 | 0.29 |
| 11 | 78.6 | 79.0 | 0.22 | 0.26 | 0.27 | 0.27 | 0.27 |
| Average | 76.445 | 76.545 | 0.2709 | 0.3000 | 0.3018 | 0.3064 | 0.3091 |
| Average weighing all gold buttons together | | | 0.2655 | 0.2955 | 0.2955 | 0.2991 | 0.2973 |

TABLE X.—*Gold Assay Test.*

*Weight, milligrams.*

| NUMBER OF ASSAYS | METHOD | |
|---|---|---|
| | Sulphuric Acid | All-Fire |
| 10 | 3.911 | 3.881 |
| 10 | 4.162 | 4.122 |
| 20 | 7.195 | 7.155 |
| 29 | 10.384 | 10.344 |
| 53 | 23.697 | 23.646 |
| Total 122 | 49.349 | 49.148 |
| | Ounces Per Ton | Ounces Per Ton |
| Average uncorrected | 0.40450 | 0.40285 |
| Correction (slag and cupel losses) | 0.00180 | 0.00380 |
| Average corrected | 0.40630 | 0.40665 |

and the now adopted sulphuric acid combination method. Clearly, the figures for the all-fire method are lower than those for the sulphuric acid method when the scorification and cupellation losses are not taken into account; but when corrections are made for these losses they may justly be deemed identical for all practical and commercial purposes.

This leads to the consideration of the corrected versus the uncorrected silver and gold assays. Nearly 20 years ago Carl Stetefeldt published a paper[1] in which he pointed out that, in assays for silver in silver and silver-lead ores the losses entailed in the commercial assay methods amounted to from 5 to 20 per cent., depending on the grade and composition of the ore, of which no account was taken. There can be little doubt that the silver losses in assaying copper ores are equal, if not greater, than those in Stetefeldt's ores. Unless such losses are carefully deter-

TABLE XI.—*Silver Losses in Scorification and Cupellation. Silver-Chloride Precipitated from Solution of Copper-Bullion.*

| Mg. or Oz. of Silver per Ton found after Scorification and Cupellation | Mg. or Oz. of Silver per Ton found in Slag and Cupels (Correction) | Percentage of Silver Retained in Slag and Cupels |
|---|---|---|
| 15.71 | 0.61 | 4.02 |
| 29.00 | 0.82 | 2.83 |
| 35.74 | 0.81 | 2.27 |
| 55.57 | 1.41 | 2.54 |
| 86.18 | 1.72 | 2.00 |
| 91.15 | 1.62 | 1.78 |
| 163.02 | 2.62 | 1.61 |
| 193.91 | 2.98 | 1.54 |
| 220.17 | 2.90 | 1.32 |
| 317.64 | 4.52 | 1.42 |
| 447.34 | 6.11 | 1.37 |
| 596.73 | 7.68 | 1.12 |
| 725.48 | 8.38 | 1.16 |
| 755.72 | 8.77 | 1.16 |

mined and the proper corrections made on all materials from the mine to the refinery, the statistics of milling, smelting and refining-efficiency must be a deception; they are unable to tell us the truth on conservation. In order to understand this it must be remembered that percentage-losses in assaying are by far the lowest in the high-grade or final products of the metallurgical establishments. For an example, let us take the loss for silver in these materials at 2 per cent. and that in the ore as probably not less than 10 per cent.; accordingly, the mill receiving the ore would be charged with only 90 per cent. of the silver which it has actually received. Suppose that in its highest product it shows, by the same assay

[1] The Inaccuracy of the Commercial Assay for Silver and of Metallurgical Statistics in Silver Mills, etc., *Trans.* XXIV, 530 (1894).

methods, a yield of 95 per cent., the apparent loss through its whole process would then be 5 per cent., while the true loss would be made up of the factors 10+5—2, or 13 per cent.

There are many data given in chemical and metallurgical literature on the assay losses of silver and gold, but Table XI., giving data on silver only, taken from actual practice in our own laboratory and derived from uniform methods, gives a wider range than anything the writer can now recall.

It clearly shows the slow increase in percentage loss toward the high-silver end, or the rapid increase of these losses toward the low-silver end of the table. Fig. 3 shows the same data plotted and the general trend illustrated by a curve. There are no corresponding data for gold available, but it may be stated that the loss percentages for that metal, in the materials and by the methods under discussion, are far smaller than for silver. A similar relation holds true as regards value losses of the two metals, which is illustrated in Table XII, the figures being taken from a full month's run of Anaconda copper-bullion. While these values look small when considered for the single ounce, they are of commercial importance when the great quantities produced per year are taken.

TABLE XII.—*Scorification and Cupellation Losses, Weight Percentage and Value Percentage for One Month's Run of Anaconda Copper Bullion.*

Value per ounce, silver = $0.60; gold = $20.67.

| | Average of Month Corrected Assay, Oz. per Ton | Weight Correction, Oz. per Ton | Correction, Per Cent. | Correction Value per Oz., Cents |
|---|---|---|---|---|
| Silver | 92.32 | 1.761 | 1.917 | 1.15 |
| Gold | 0.522 | 0.0016 | 0.031 | 0.64 |

Corrected assays should not only be made for commercial reasons on high-grade products, but they should be universally advocated for the sake of accuracy itself, for the establishment of correct efficiency records and true conservation statistics.

In making these assay-loss corrections for such uniform material as the Anaconda copper or that of other large producers it is perfectly permissible to collect assay slags and cupels for a whole month's run and to apply the correction found to the individual results of the month following. The mode of operation is as follows: The slag and cupels are crushed in a small jaw-crusher and thoroughly mixed. The whole or an aliquot part may be taken for the reduction-fusion in G crucibles, each of which is charged, in grams, with 200 slag and cupels, 70 borax, 70

bicarbonate of sodium and 10 flour. The resulting lead buttons are scorified and cupeled; the resulting silver-gold buttons by weighing and parting yield the corrections sought.

In order to obtain full recovery of the silver and gold from slags and cupels it is necessary to reduce practically all the lead from each crucible charge. For this 10 g. of flour with a salt cover suffice; the salt cover serves no other purpose than the prevention of combustion of the reducer.

Inquiry into the absolute accuracy of the heretofore-described assay methods naturally suggests itself. We have already seen the slag and cupel losses for silver and gold and have noted that corrections are made for them. There are, however, other losses, such as by volatilization, by solution of the silver chloride, and in mechanical ways; their values are

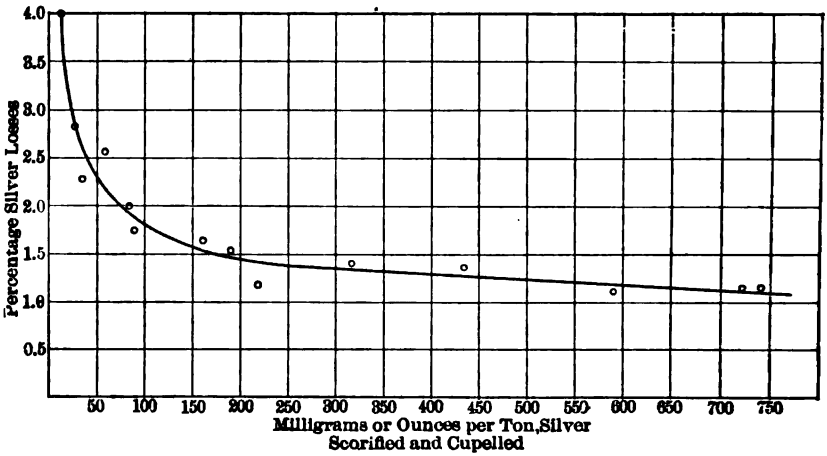


FIG. 3.—SILVER LOSSES IN SCORIFICATION AND CUPELLATION. SILVER CHLORIDE PRECIPITATED FROM SOLUTION OF COPPER BULLION.

of very difficult determination by direct methods. These losses are entirely or in part offset by the amount of impurities which remain with the buttons and are weighed as precious metals. It is a simple matter to analyze a sufficient number of silver buttons. Thirty grams of these, each weighing between 80 and 100 mg., were found to contain 0.16 per cent. lead and 0.15 per cent. bismuth, total 0.31 per cent., or the buttons to be of a fineness of 996.9. Table XIII. shows a test in which fine silver was subjected to exactly the same process as the silver in the assay of copper bullion, *i. e.*, the fine silver was dissolved in a copper solution free from silver and precipitated as chloride, etc. It will be seen that the final and corrected result shows a small deficit, amounting to 0.098 oz. per ton, or to 0.12 per cent. of the total silver. More extended tests were undertaken with more silver and in comparison with higher grade materials,

anode residues or slimes. Here, too, the fine silver was subjected to the same treatment as the slimes, a sulphuric acid combination assay, and through the furnace process slimes and silver alternated in position, in scorifiers as well as in cupels. One-tenth of an assay ton of slimes was taken; all of the same sample. The fineness of the buttons was determined by titration by Volhard's method, with potassium sulphocyanate. The results of such a test are given in Tables XIV. and XV.

In these tables it will be observed that the large silver buttons are of slightly lower fineness than that which was given for those of lighter weight. Of chief interest is the fact, that when to the fine silver of the assay buttons of the slimes is added the slag and cupel correction and the volatilization and miscellaneous loss correction, as determined by the assay treatment of fine silver, the results correspond very closely to the usual results when the slag and cupel correction is added to the assay button of the slimes. Repeated tests confirmed this and the conclusion follows, that impurities in assay buttons and volatilization and miscellaneous losses balance each other. It follows that the assay of anode residues for silver by this assay method is correct in every case to within about plus or minus 0.1 per cent. of the total silver, an accuracy which could hardly be attained by any other chemical method. Of course, this degree of accuracy is only obtained with a sufficient number of determinations, averaged to one result, in our case 10, with 0.1 assay ton each of the material.

The contract assay methods for such materials are still of an archaic kind, of the all-fire nature, elastic beyond reason, in which results differing several per cent. are often obtained. It is to be hoped that concise methods will soon take the place of the old.

For the western (smeltery) and the eastern (refinery) end of the Anaconda Company, sampling has been established on a scientific basis and the most accurate assay methods are employed to determine the values of the copper-bullion shipped from one to the other. Any inaccuracy in sampling will offset the accuracy of analysis, or *vice versa;* an equal degree of accuracy is requisite for both. The concordance of results obtained in monthly averages at the two places is illustrated in Table XVI.

The data given in this paper were, in the greater part, determined and accumulated in the interest of the Anaconda Copper Mining Co., in its laboratory at the Raritan Copper Works, Perth Amboy, N. J. In this work the laboratory staff must naturally have contributed its share and the writer herewith especially acknowledges the services of K. W. McComas and W. L. Raup, Jr.

TABLE XIII.—*Experiment to Show Losses in Silver Assaying by the Combination Method.*

| Fine Silver Weighed Out Milligrams | Silver Recovered Milligrams | Silver Recovered, Corrected Milligrams |
|---|---|---|
| 81.62 | 80.28 | 81.61 |
| 83.74 | 82.34 | 83.67 |
| 83.44 | 82.18 | 83.51 |
| 93.62 | 92.24 | 93.57 |
| 83.56 | 82.18 | 83.51 |
| 81.04 | 79.40 | 80.73 |
| 85.20 | 83.72 | 85.05 |
| 85.82 | 84.28 | 85.61 |
| 85.18 | 83.66 | 84.99 |
| 84.40 | 82.78 | 84.11 |
| 83.42 | 82.24 | 83.57 |
| 87.40 | 86.00 | 87.33 |

Average silver weighed out, mg.....84.870
Average silver recovered, mg.....83.442

Difference..... 1.428

Average silver weighed out, mg.....84.870
Average silver recovered, corrected, mg.....84.772

Difference..... 0.098

Correction for slag and cupel losses = 1.327 mg. = 1.33 mg.

TABLE XIV.—*Fine Silver Treated by Sulphuric Acid Assay Method.*
*Silver titration by Sulphocyanate Method.*

| No. | Fine Silver Weighed Out Mg. | Assay Buttons Mg. | KCNS 1 cc = 4.4907 Ag. Cc. | Fine Silver Mg. | Impurities Mg. | Difference Orig. Fine and Button Fine Mg. |
|---|---|---|---|---|---|---|
| 1 | 374.28 | 370.24 | 82.1 | 368.69 | 1.55 | 5.59 |
| 2 | 376.94 | 373.38 | 82.9 | 372.18 | 1.20 | 4.76 |
| 3 | 373.47 | 369.68 | 82.1 | 368.69 | 0.99 | 4.78 |
| 4 | 374.26 | 370.06 | 82.1 | 368.69 | 1.37 | 5.57 |
| 5 | 373.96 | 370.72 | 82.2 | 369.14 | 1.58 | 4.82 |
| 6 | 369.81 | 366.56 | 81.3 | 365.09 | 1.47 | 4.72 |
| 7 | 371.73 | 367.76 | 81.4 | 365.54 | 2.22 | 6.19 |
| 8 | 373.96 | 370.56 | 82.1 | 368.69 | 1.87 | 5.27 |
| 9 | 370.25 | 367.02 | 81.3 | 365.09 | 1.93 | 5.16 |
| 10 | 372.70 | 368.74 | 81.6 | 366.44 | 2.30 | 6.26 |
| Average... | 373.136 | 369.472 | ..... | 367.824 | 1.648 | 5.312 |
| Slag and cupel correction..... | | 4.012 | | 4.012 | 4.47 per M | ..... |
| Corrected Assays | | 373.484 | ..... | 371.836 | | |

Silver recovered by corrected assay..... 373.484
Original fine silver..... 373.136
Difference.....0.348 = 0.0933 per cent

Original fine silver..... 373.136
Total silver recovered..... 371.836

Volatilization and other losses..... 1.300
Fineness of assay buttons..... 995.53

TABLE XV.—*Silver Assay of Anode Residues by Sulphuric Acid Method.*

*Silver titration by Sulphocyanate Method.*

| No. | Assay Button Mg. | KCNS 1 cc = 4.4907 Ag. Cc. | Fine Silver Mg. | Impurities Mg. |
|---|---|---|---|---|
| 1 | 371.22 | 82.2 | 369.14 | 2.08 |
| 2 | 371.46 | 82.5 | 370.48 | 0.98 |
| 3 | 371.42 | 82.5 | 370.48 | 0.94 |
| 4 | 369.94 | 82.1 | 368.69 | 1.25 |
| 5 | 370.82 | 82.3 | 369.58 | 1.24 |
| 6 | 370.90 | 82.2 | 369.14 | 1.76 |
| 7 | 371.44 | 82.3 | 369.58 | 1.86 |
| 8 | 370.70 | 82.1 | 368.69 | 2.01 |
| 9 | 371.74 | 82.4 | 370.03 | 1.71 |
| 10 | 371.48 | 82.2 | 369.14 | 2.34 |
| Average | 371.112 | .... | 369.495 | 1.617 |
| Correction | 3.860 | .... | 3.860 | 4.36 per M. |
| Volatilization, etc. | ..... | .... | 1.300 | ..... |
| Total | 374.972 | .... | 374.655 | ..... |

Difference ..... 0.317 = 0.106 per cent.
Fineness of assay buttons, 995.64.

TABLE XVI.—*Assay-Results in Monthly Averages, East and West, Obtained on the same Copper-Bullion.*

(About 125 samples assayed per month).

| MONTH | WEST | | | EAST | | |
|---|---|---|---|---|---|---|
| | Copper, Per Cent. | Silver, Oz. per Ton | Gold, Oz. per Ton | Copper, Per Cent. | Silver, Oz. per Ton | Gold, Oz. per Ton |
| 1 | 99.3091 | 87.3093 | 0.4606 | 99.3139 | 87.2482 | 0.4592 |
| 2 | 99.3203 | 88.7305 | 0.6027 | 99.3310 | 88.7898 | 0.6057 |
| 3 | 99.2816 | 91.2517 | 0.4786 | 99.3091 | 91.1501 | 0.4817 |
| 4 | 99.2763 | 91.7445 | 0.4355 | 99.3073 | 91.5903 | 0.4382 |
| 5 | 99.2661 | 87.0271 | 0.4880 | 99.3061 | 87.0271 | 0.4877 |
| 6 | 99.2613 | 89.2657 | 0.5145 | 99.2979 | 89.1750 | 0.5143 |
| Average | 99.2843 | 89.2620 | 0.4955 | 99.3101 | 89.2006 | 0.4966 |
| West over | ..... | ..... | ..... | ..... | 0.0614 | ..... |
| East over | 0.0258 | ..... | 0.0011 | ..... | ..... | ..... |

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[SUBJECT TO REVISION].

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Butte meeting, Aug. 18 to 21, 1913, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1913, when Vol. XLVI. of the *Transactions* will go to press. Any discussion offered thereafter should preferably be in the form of a new paper for publication in Vol. XLVII. (with suitable cross references in both volumes).

# The Smelting of Copper Ores in the Electric Furnace.*

BY DORSEY A. LYON AND ROBERT M. KEENEY, PITTSBURGH, PA.

(Butte Meeting, August, 1913.)

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## I. Introduction.

In presenting this paper the writers wish to call attention first of all to the fact that the electric furnace was not developed as a competitor of the combustion furnace, but:

1. For the purpose of doing high-temperature work which it is not possible to do in the combustion furnace; and

2. For the treatment of ores from deposits which are located in regions where fuel is scarce and costly, and where hydro-electric power is comparatively cheap, as is the case in Chili, in Canada, in certain parts of this country and Mexico. This paper is not presented with the idea of trying to prove that the electric furnace should replace the reverberatory or the blast furnace as used at present in the smelting of copper ores, but that it may be substituted for them in those localities which are not favorable to them, and where either of the two following conditions may exist:

(a) A district remote from railway transportation facilities, and where there is a deposit, or deposits, containing sufficient values to warrant their being worked, but which, due to the cost of getting coke in for operating a smelter and of getting ore out to a smelter, have to remain unworked. And where, on the other hand, there is plenty of water power which can be developed at reasonable cost for the production of electric power, by the use of which for smelting the ores in an electric furnace, with subsequent Bessemerization if necessary, a concentrated product may be obtained, which can easily stand the necessary transportation charges.

(b) Or, comparatively speaking, the transportation facilities may be all right, but the cost of fuel be too great to permit of the ordinary methods of smelting, whereas, on the other hand, hydro-electric power can be developed in the district at a low cost, and the ores could be smelted by the electric furnace at a cost that would make their treatment possible.

With these possibilities in mind we have attempted to make a brief comparative study of the problem for the purpose of aiding those interested in the subject, in determining whether it would be possible metallurgically, and feasible commercially, to use the electric furnace in those localities where, by reason of excessive fuel costs, the cost of operating the furnaces for smelting copper ores is either excessive or prohibitive.

Although the smelting of copper ores in the electric furnace has received considerable attention and more or less experimental work has been done, so far as we are aware there are no electric furnaces in the United States which are being worked on copper ores. In Norway, however, trial smeltings of copper ores with an electric furnace of 1,000 h. p. and an estimated producing capacity of 2,000 tons per annum, have been

conducted at the Ilen Smelting Works, Trondhjem. A consignment of 25 tons of pure copper, the first copper ever produced by electric smelting in Norway, was exported from the Bitovaria Minea at Kaafjord, near Dyngor. Some months ago it was reported that, at Christiansand, the electric smelting and refining of nickel and copper would be undertaken by the A/s Kristianssands, Nikkelroffeneringsverk.

## II. Chemistry of Copper Smelting.

Without going into details, the objects aimed at in copper smelting may be stated as follows:

1. To concentrate the copper and precious-metal values. If a native copper or an oxide ore is smelted, the concentrated product is in the form of a metal; if a sulphide ore is treated, the product is a matte.

2. To get rid of the excess iron and gangue materials as slag.

In general we may say that three classes or ores have to be dealt with in copper smelting, namely: 1. Native copper ores; 2. Oxide and carbonate ores; 3. Sulphide ores.

### *1. Native Copper Ores.*

From the nature of occurrence of the valuable mineral content in the native state, the smelting of a native copper ore or concentrate resolves itself into a melting operation and separation of the metallic copper and gangue by the difference in specific gravity. After removal of the slag a further refining of the black copper from the melting operation is necessary. This is the simplest form of copper smelting. The electric furnace is particularly adapted metallurgically to a simple melting operation, in a neutral atmosphere, and appears to have particular application in this branch of copper smelting, especially for fine concentrates, as shown by recent experiments of the writers, to be described later.

### *2. Oxide and Carbonate Ores.*

The chemistry of smelting of oxide, carbonate or silicate ores of copper consists simply of the use of coke as a fuel and a reducing agent in the blast furnace, to produce metallic copper. Here also the separation of the slag and metal is important, and there is need for careful regulation of the coke to prevent reduction of iron by any excess carbon. The atmosphere is reducing and not neutral. This also is a condition easily obtainable in the electric furnace, as is done to-day in electric iron

smelting. The experiments of Stephan, to be described later, show this possibility.

### 3. *Sulphide Ores.*

The smelting of sulphide ores presents further complications. In order to concentrate the copper and precious metal values in a matte, one must first eliminate as much of the sulphur of the ore as may be necessary to obtain the desired results, and must cause the resultant oxides to unite with silica, which may be present in the ore or which may be added, preferably in the form of siliceous ore.

#### *a. Elimination of Sulphur.*

As pointed out by Peters,[1] "when fused in a reducing atmosphere, certain of the sulphides, such as $Cu_2S$, FeS, PbS, etc., melt down without change, while the very common and important mineral, pyrite, loses about one-half of its sulphur, which is driven off as sulphur, in the shape of yellowish fumes, which will deposit a coating of brimstone on a cool surface." As also stated by Peters,[2] "The only two sulphides of much interest to the metallurgist, which lose a portion of their sulphur by heat alone in the manner just described, are: (1) pyrite ($FeS_2$), which is assumed to lose one-half of its contents in sulphur by mere heating, without air; and (2) chalcopyrite ($Cu_2S$, $Fe_2S_3$), which (as generally assumed by metallurgists) is composed of one-third copper, one-third iron, and one-third sulphur, and loses (by melting without air) one-third of its sulphur, or one-ninth of its entire weight."

This has been quoted from Peters in order to show clearly and concisely that there would be no object in simply melting down sulphide ores in an electric furnace. That is, if we did so, we could only get an altered form of the ore, somewhat concentrated, to be sure (to the extent of the loss of the elemental sulphur), but not to the desired extent.

Such being the case, the smelting of sulphide copper ores resolves itself into one of the three following methods, or a combination of them, in order to effect the object named above.

1. The mixing of oxides and sulphides in such proportions as to get the proper reaction between them to effect the degree of concentration that is desired.

2. The roasting of the sulphide ores, thus converting them into oxides, or partially so, depending upon the degree to which the roasting is carried out, and then melting down the roasted ore in a suitable furnace. In this way the excess iron and gangue materials present may be elim-

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[1] *Principles of Copper Smelting.* [2] *Loc. cit.*

inated, for sulphur has a greater affinity for copper than it has for iron, and any sulphur present in the charge during the melting down of the latter will combine first with the copper, and only the excess over this amount will unite with iron. The remainder of the iron will be present as iron oxide (FeO). This will unite with silica and form a slag, and thus the concentration of the copper and precious metals into a matte is brought about.

3. As the roasting of the ore necessitates an additional step in the process, it is to be avoided if possible, and the desired oxidation of the sulphides accomplished in the blast furnace by means of the oxygen forced into the furnace through the tuyeres. This is what the modern copper blast furnace does, namely, combines the functions of roasting and smelting. In other words, it oxidizes and smelts simultaneously, and also makes use of the heat that is generated by the oxidation of the sulphur and iron. The utilization of the heat received from the oxidation of the iron and sulphur in this manner is called pyrite smelting, although, as will be seen later, true pyrite smelting means the smelting of ores by heat arising solely from their own oxidation and without the addition of any carbonaceous fuel.

Having thus outlined briefly the objects aimed at and the method used in the smelting of copper ores, let us now turn our attention to the manner in which the smelting of copper ores is carried on.

## III. Reverberatory Copper Smelting.

Inasmuch as copper ores are smelted in one of two ways, that is, either in a reverberatory or in a blast furnace, it will first be necessary to ascertain if the conditions essential for the successful carrying out of these two methods of smelting can be attained in the electric furnace.

As stated by Peters,[1] "the one fundamental and pre-eminent duty of this type of furnace is to produce a smelting temperature so that the ore may become liquid, and thus be able to separate into slag and matte." From the standpoint of this discussion, the reverberatory furnace may be said to be used for the smelting of ores and concentrates which cannot be successfully treated in a blast furnace.

"The manner in which the heat from the fuel is applied in this method of smelting, though much improved in modern practice, is a peculiarly wasteful one, as the mere rapid passing of a flame over the surface of a layer of ore resting upon a comparatively cool hearth, is a particularly incomplete way of transferring heat from the flame to the ore particles,

[1] *Principles of Copper Smelting.*

especially as the latter are usually poor conductors of heat. . . . The very large surface of walls, arch, flue, stack-lining, etc., is in just as favorable a position for being smelted as is the ore itself, and in spite of its refractory composition, needs frequent repairs and renewals. . . . Such a method could only originate in a district possessing cheap flaming coal, and inexpensive refractory materials, and Swansea (where the process originated has these in abundance. . . . The reverberatory process is but little complicated by the two active extraneous agents which exert so marked an effect upon the chemistry of the blast furnace smelting of roasted ores, and the pyrite smelting of sulphide ores; namely, carbon in the one case, and oxygen in the other. . . . Speaking in a broad general sense, the ore in the reverberatory smelting furnace is subjected to neither of these influences to any marked degree, and is thus enabled to work out its own salvation under the influence of heat alone. Such chemical reactions as take place in the ore mass during the smelting result almost entirely from the behavior of substances already contained in the ore, and affected by the mutual action of the various constituents of the charge itself. This renders the chemical part of the operation unusually simple, and enables the smelter to concentrate his efforts upon developing the high temperature essential to the complete fusion of the ore."

## IV. Substitution of the Electric Furnace for the Reverberatory Furnace.

If, then, as Peters states: "The proper function of the reverberatory furnace, in smelting copper ores, is to generate heat as rapidly as possible," there is no apparent reason why copper ores cannot be smelted equally as well in an electric furnace, and perhaps even more advantageously, for, as pointed out by Peters, and as is well known, the thermal efficiency of the average reverberatory furnace is only from 5 to 8 per cent., although about 20 per cent. is now obtained in the large Anaconda reverberatories, whereas, that of an electric furnace which would be used for smelting copper ores suitable to refractory smelting, would probably have an efficiency of as much as 70 per cent. Chemically the only ill effects that might possibly result from the use of such a furnace would be the effect of the carbon of the electrodes upon the iron oxides which might be contained in the charge and which would result in:

1. The loss of electrodes;
2. The production of metallic iron, which would contaminate the metallic copper and increase the cost of refining; or, if a matte was the final product, might necessitate re-concentration.

3. The loss of electric energy, due to the additional heat consumed by the reduction of iron.

Since the reverberatory furnace is essentially a means of melting, whether used for sulphide, oxidized or native ores, the results obtained in the electric smelting of native copper ores will serve for the sulphide and oxide ores also, for comparison with the reverberatory in order to clear up the three points referred to above. The slags would be somewhat similar in each case. The atmosphere of the electric smelting furnace charged with sulphide ore would not be as neutral as in the furnace charged with native copper ore, because of the tendency of the elemental sulphur and any sulphur dioxide formed to dilute the air so as to make the furnace practically reducing as far as the consumption of electrode by oxygen from the air was concerned. If the electric furnace was not kept closed, considerable sulphur dioxide might be formed, which would possibly be reduced in part by the carbon electrode. Even with such reduction of sulphur dioxide, the electrode consumption would probably be less when working on sulphide ores than when working on native copper ores. This is shown by Bureau of Mines experiments, when, with a charge of native copper concentrate, the maximum electrode consumption in fourteen experiments was 48.0 lb. of graphitized carbon per ton of charge, the minimum, 22.1 lb. and the average 34.5 lb. In the same furnace, with charges of sulphide, oxide and siliceous ores mixed, a matte being the product, the maximum electrode consumption for 20 runs was 60.0 lb., the minimum, 10.8 lb., and the average, 23.3 lb.

## V. Smelting of Fine Michigan Native-copper Concentrates in the Electric Furnace

For the purpose of investigating the feasibility of smelting fine Michigan native-copper concentrates in the electric furnace, instead of the reverberatory furnace, experiments were conducted by the writers, the detailed results of which will appear later in a Bureau of Mines Technical Paper. The objects of these experiments were:

(1) To note the effect of variation of the three main slag constituents, silica, ferrous oxide and lime, upon the power consumption, the operation of the furnace and the purity of the copper produced;

(2) To ascertain the extent of iron oxide reduction caused by the graphite electrode;

(3) To compare intermittent smelting with continuous smelting.

Two grades of concentrates were smelted: No. 3 and No. 4, of which No. 3 was much the coarser; of the former, 69.73 per cent. passed through

a 60-mesh sieve, as compared with 88.12 per cent. of No. 4. The chemical analyses of the ore were as follows:

| | No. 3 Per Cent. | No. 4 Per Cent. |
|---|---|---|
| Cu | 37.35 | 25.35 |
| $SiO_2$ | 33:33 | 35.92 |
| FeO | 15.40 | 21.60 |
| $Al_2O_3$ | 8.78 | 12.40 |
| CaO | 5.18 | 5.98 |
| MgO | 1.09 | 1.09 |
| S | 0.056 | 0.096 |
| Fe (metallic) | 0.30 | 0.50 |

Fifteen experiments were performed in a Siemens type of electric furnace, having a capacity of 50 kilowatts. The furnace was operated as a resistance furnace. The charge and products of a typical experiment were the following:

| Charge | | Black Copper | | Slag | |
|---|---|---|---|---|---|
| | *Lb.* | | *Per Cent.* | | *Per Cent.* |
| Concentrate No. 3 | 22.0 | Cu | 98.59 (by difference) | Cu | 0.15 |
| Hematite | 1.1 | Fe | 1.10 | $SiO_2$ | 44.88 |
| | | S | 0.29 | CaO | 6.92 |
| | | As | 0.02 | MgO | 2.32 |
| | | | | FeO | 29.80 |
| | | | | $Al_2O_3$ | 15.34 |

*1. Conclusions.*

(1) Fine Michigan native-copper concentrates can be smelted in the electric furnace with the production of a good grade of black copper and without excessive losses of copper in the slag or by other means.

(2) The percentage of copper in the slag need not exceed 0.25 per cent., with a slag of proper composition. The loss in the slag should not exceed 0.50 per cent. of total copper charged. Other losses should not exceed 1.0 per cent.; giving a total loss of 1.5 per cent. of all the copper charged.

(3) The loss of copper by volatilization is high, if the slag is much more acid than a mono-silicate, and if it contains much alumina. The lowest losses were obtained by addition of limestone to form a slag containing 35 per cent. $SiO_2$, 22 per cent. CaO and MgO, and 25 per cent. FeO.

(4) The difference between limestone and hematite as a flux is not marked, one being about as efficient for this purpose as the other, if the slag has not too high a specific gravity. No more iron was found in the black copper product when hematite was used than when limestone was used.

(5) With a furnace operating at a low temperature on a mono-silicate slag, the reduction of iron by the carbon electrode should not be excessive or sufficient to result in a product containing less than 95 per cent. metallic copper. With care a product containing 98 per cent. copper could probably be produced regularly.

(6) The total electrode consumption greatly exceeds the amount of carbon necessary for reduction of the iron found in the metal. An average of 9.35 per cent. of the total was used in iron reduction. The electrode consumption should not exceed 10 lb. per ton of charge.

(7) The electric furnace operates satisfactorily with the high siliceous alumina slags, produced by the natural gangue without flux, but the losses by volatilization are high because of the high melting point of the slag. There is no copper oxide in the product, as was shown by microscopic examination.

(8) The power consumption for smelting fine Michigan native copper concentrates of between 25 and 40 per cent. copper in an electric furnace of 750 kw. or greater capacity should not exceed 640 kw. hr. per ton of ore smelted.

## VI.—Proposed Process for Electric Smelting of Fine Native-copper Concentrates.

The commercial smelting of fine native copper concentrates in the electric furnace to attain the greatest economic success would be carried out in a continuous operation, consisting of charging the mixture of ore and flux at intervals into the top of an electric furnace having a short stack above the smelting crucible for preheating the charge before it reached the smelting zone, and tapping off the metal at intervals, the slag being permitted to run continuously from the furnace through a slag notch above the metal hole. The possibility of operation in this manner was shown in an experiment where there was an average of 1.10 per cent. iron in the product and 1.37 per cent. copper in the slag. The total loss of copper was as low as in the intermittent runs, and was largely due to the small scale operation. The slag loss was high because of imperfect settling, and because of the tapping of partially fluid material with the slag; a condition that would not occur in a large furnace operated properly. The power consumption was lower than in the other experiments.

The essential conditions for the successful performance of this process are a crucible, where the electrical energy is introduced and where the metal settles from the slag, and a short shaft above the crucible for preheating the charge before it reaches the smelting zone. As there is no reduction to be performed, this shaft need not be higher than to give plenty

of head room between the top of the crucible and the charging floor for the moving of the electrode.

A furnace of the type shown in Figs. 1 and 2 satisfies these conditions. It consists of a smelting chamber lined with fire brick or silica brick, having three carbon electrodes suspended vertically through the roof with a

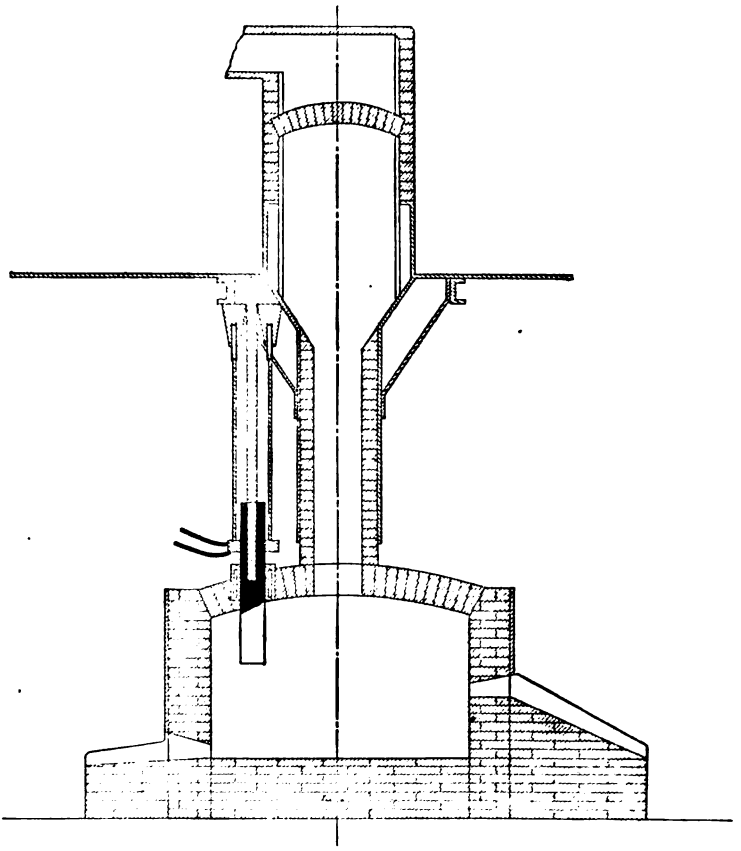


FIG. 1.—ELEVATION OF ELECTRIC FURNACE FOR SMELTING NATIVE COPPER CONCENTRATE.

feeding shaft lined with fire brick over the crucible. A 750-kw. furnace would treat, per 24 hr., about 23 tons of native copper concentrate, containing 25 to 40 per cent. copper. This furnace would be operated with three-phase current at from 50 to 100 volts. The general internal dimensions of the crucible are 14 ft. 6 in. in diameter, by 9 ft. high, with a shaft 1 ft. 6 in. in diameter by 18 ft. 6 in. high. With a furnace of this type regulation of electrodes would not be necessary after regular operation was established, so that the electrical connection is made close

to the roof of the furnace and regulation is by hand. There is less difficulty in handling the electrodes if they are suspended vertically; and, as the shaft is low, it is made vertical, thus eliminating the necessity of suspending them at about 55 degrees.

The slag from such a furnace should not contain over 0.25 per cent. copper, and could be discarded at once without further treatment. It could be granulated with water as discharged from the furnace. The metal would require further refining for elimination of iron and other impurities, which could be done by charging the hot metal directly into a reverberatory furnace, where it would be refined as previously described.

Briefly, the present process for treating these concentrates consists of the following steps: first melting and refining the coarse concentrate

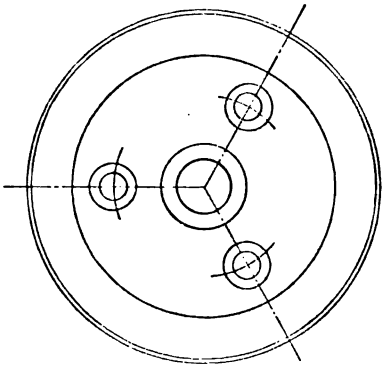


FIG. 2.—PLAN OF ELECTRIC FURNACE FOR SMELTING NATIVE COPPER CONCENTRATE.

in either one or two reverberatory furnaces, from which a slag is produced containing from 10 to 30 per cent. copper; second, agglomerating the fine concentrate in a reverberatory furnace, or briquetting it with lime; third, smelting the slags of the reverberatory furnaces and the agglomerated or briquetted fine material, with coke in a blast furnace, the slags from which contain 0.6 to 0.8 per cent. copper and the metal, 90 to 95 per cent. copper; and fourth, retreatment of this metal in a reverberatory furnace.

The use of the electric furnace for treatment of these concentrates gives the following advantages: first, no agglomeration or briquetting would be necessary; second, no slags produced by this furnace would require retreatment; third, a large proportion of the higher-grade concentrates, containing 75 per cent. copper, could be mixed with the fine concentrates and treated directly in the electric furnace, thus reducing the amount of slag necessary for re-smelting; and fourth, if cheap power was available, the electric furnace could be used for reducing the copper from the reverberatory slags, and the resulting metal would probably be lower in iron

and the slag lower in copper, because of the greater ease of regulating the reducing atmosphere, as only sufficient coke is present for the reduction of combined copper.

The limit to which the higher-grade concentrates could be mixed with the fine material would be determined by the electric conductivity of the mixture in the furnace. In the experiments performed with a charge containing 37 per cent. metallic copper, no difficulty from short circuits or other causes was experienced. The point to which the amount of copper in the charge could be increased would have to be ascertained by experiment.

Technically the electric furnace does away with the necessity of briquetting or nodulizing, and, in general, facilitates handling the materials. The commercial factors influencing the cost would largely depend upon local conditions as to cost of power and fuel. Opposed to the cost of power would be the cost of fuel and briquetting or nodulizing, with possibly a higher labor cost for the present practice. If the cost of power was equal to these three factors, and no less, the electric furnace might still have the advantage of facility of operation. Also, in the case of a mine remote from smelting plants, it might be cheaper to concentrate the low-grade product of wet concentration by smelting in a small electric furnace, and thus reduce the cost of freight. Finally, to save losses of wet concentration, the ore might not be concentrated to so high a degree in this manner, but a low-grade concentrate made, which would be smelted directly in the electric furnace and the high-grade black copper shipped to the custom smelter.

We have obtained figures from a reliable source showing that the cost of treatment of these fine concentrates by agglomeration in a reverberatory furnace, refining of this copper produced, retreatment of slags in the blast furnace, and refining and casting the cupola copper into marketable shapes, is $8.64 per ton of 35 per cent. native copper concentrate. This includes overhead charges, all repairs and maintenance of plant, but not amortization.

The treatment of the same material by the proposed electric furnace process to produce marketable shapes is estimated as follows:

| Cost of Electric Power | | Cost of Treatment Per Ton of Concentrate |
|---|---|---|
| Per Kilowatt Hour | Per Kilowatt Year | |
| 0.5 | $43.80 | $7.18 |
| 0.625 | 54.80 | 8.08 |
| 0.75 | 65.80 | 8.87 |

A rate of one-half cent per kilowatt hour with a high load factor is possible in many districts with the use of steam turbine plant, if cheap hydro-electric power is not available. The load factor in this case would be above 90 per cent., and the power factor not below 0.85. In practice the cost of refining "cupola" block of the electric furnace would probably be lower than in the case of blast furnace cupola blocks, because of the higher percentage of copper in it. In the reverberatory process, 81.3 per cent. of the material charged must be retreated as slag in the blast furnace. In the electric furnace process, the only material to be retreated would be about 5 per cent., as slag from the refining of the cupola blocks in the reverberatory. This slag could be retreated in the electric furnace. The great saving in retreatment charges accounts for the possibility of use of the electric furnace in this connection, even with expensive electric energy.

## VII.—Ordinary Copper Blast Furnace Smelting.

Ordinary copper blast furnace smelting, that is, smelting the ore with carbonaceous fuel, differs widely from pyritic smelting, which consists in smelting the ore with heat derived from the oxidation of its own sulphur and iron contents.

As stated by Peters[1], "The main characteristic of ordinary blast furnace smelting is the employment of carbonaceous fuel, mixed with ore, as the principal source of heat." Such being the case, the problem that presents itself is to determine whether it is metallurgically feasible to substitute electricity for carbonaceous fuel as a source of heat, and whether in so doing, we will obtain as favorable results chemically as it is possible to obtain in ordinary blast furnace smelting.

### 1. *Prerequisites.*

In the smelting of copper ore, sulphide ores in the main have to be dealt with, and the problem consists in getting rid of the sulphides other than copper, and of concentrating the latter into a matte containing anywhere from 20 to 60 per cent. copper. The first step is, therefore, to free the ore from its sulphur. We will first consider how this is done in ordinary copper blast furnace smelting. As pointed out by Peters[2], due to the constant presence of a large amount of glowing carbon in the furnace shaft, extending from the tuyeres for several feet up the shaft of the furnace, and often to the charging door itself, the oxygen of the air blown in through the tuyeres is consumed with extraordinary rapidity and thoroughness, and there is consequently but a poor chance for the oxidation of any substances in the ore, which possess less affinity for oxygen than carbon does.

[1] *Principles of Copper Smelting.* [2] *Loc. cit.*, p. 109.

"Therefore, in ordinary blast furnace smelting with carbonaceous fuel, the sulphides of iron and copper, and such small quantities of other metallic sulphides as may be present, stand little chance of obtaining any of the oxygen to combine with, to form sulphur dioxide and metal oxide. The atmosphere is almost always strongly reducing, and the sulphides tend to melt down with almost the same result that they would in a closed crucible, with complete exclusion of air. . . . It will thus be seen that, owing to the reducing atmosphere which prevails, we can, when smelting sulphide ores, look for the removal of but little sulphur in the blast furnace beyond the portion that will be driven off by heat alone without the presence of oxygen."

Of the two sulphides in the charge which have to be taken into consideration in this connection (pyrite, $FeS_2$, and chalcopyrite, $Cu_2S$, $Fe_2S_3$), the former loses one-half of its contents in sulphur by mere heating, without air, while the latter, similarly heated, loses one-third of its sulphur and one-ninth of its entire weight.

Continuing, Peters states, "It is obvious, then, that there would be little object in smelting unroasted ores carrying a high percentage of metallic sulphides in a blast furnace running on carbonaceous fuel."

From what has been stated, it would appear that electric heat could be used as well as heat derived from the combustion of coke for the melting down of the ore. However, such melting would not, in either case, give us the concentration we desire, and so we next turn our attention to that phase of the problem, namely, the concentration of the copper and accompanying precious metal values into a matte, by the removal of sulphur and the subsequent oxidation of the iron.

The removal of sulphur in ordinary blast furnace smelting is brought about in two ways: (1), by heat alone, which drives off a part of sulphur as elemental sulphur, and (2), from the reaction of oxides and sulphates present in the ore as such, upon the sulphides. The presence of oxides in the charge may be due to the use of oxidized ores, while, if roasted ores are used, both oxides and sulphates may be present. These compounds react with the sulphides present, a partial oxidation of the latter takes place, and some of the sulphur is removed as sulphur dioxide ($SO_2$). It is to be remembered, that the atmosphere of the ordinary blast furnace is reducing and hence these reactions in this instance have to take place in a reducing atmosphere, and for this reason do not take place as freely as they would if in a neutral atmosphere. In general, it may be stated that, with no oxides present in the ore charge, practically no sulphur (aside from the elemental sulphur removed by heat, as above stated), would be driven off, except, as stated by Peters, "the small amount that would be oxidized by the blast, in spite of the generally reducing atmosphere of the furnace." He also states, judging from his

own experience, that this "loss of extra sulphur in the blast furnace cannot be placed higher than 5 per cent. of the sulphur remaining in the ore after deducting the sulphur which the pyrite and chalcopyrite lose by direct volatilization."

Summarized then, we may say that the ordinary blast furnace smelting of copper ores consists simply in a melting-down process, and the sulphur removed is that which is removed when a sulphide is heated without air, and that any concentration that takes place is due to the reactions which take place between the oxides and sulphates present, and which would take place better in a neutral atmosphere than they do in a blast furnace, the atmosphere of which is reducing. Such being the case, a blast furnace operating in this manner is only a melting-down furnace and gives the same results which would be obtained by melting such an ore charge in an open crucible. The only advantage which the blast furnace has in this instance over the reverberatory furnace is that it is more efficient as a melting furnace, because it affords intimate contact between the fuel of the charge and the resultant gases, and permits a continuous operation; whereas, the reactions between the oxides and sulphates do not take place as readily as they would in the comparatively neutral atmosphere of the reverberatory furnace, or of the still more neutral atmosphere of the electric furnace. So far then, aside from the question of costs, there is apparently no reason why the electric furnace could not be substituted for the ordinary blast furnace when operated on the class of ores mentioned above.

### 2. *Losses.*

Let us now turn our attention to the subject of the losses which occur in ordinary blast furnace work, and, as far as possible, determine their causes. These losses are, according to Peters:

1. The loss of small particles, in the form of dust, which contain valuable metals, and which loss is brought about every time the ore is moved.
2. Flue dust losses, due to blast.
3. Volatilization losses.
4. Losses in slag due to "particles of matte or metal which have failed to separate properly from the slag, and to a lesser degree from the oxidation of the oxides of the valuable metals—especially lead and copper—that have entered into combination with the silica, and have thus become chemically part of the slag."

Would these same losses occur if an electric furnace (with no blast) were used instead of a blast furnace, operated in the manner above described?

Loss 1. This would be common, of course, to both the blast furnace and the electric furnace.

Loss 2. Although there would be more or less fine particles carried away as dust by the gases escaping from the electric furnace, the amount of the same, of course, would not be as great as would be the case with the blast furnace.

Loss 3. An electric furnace used for this purpose would be operated more in the man-

ner of a resistance furnace (the charge acting as the resistor), than as an arc furnace, and hence the heat adjacent to the electrodes would not be as great as would be the case if an arc were used, and probably not greatly in excess of that which is generated at the tuyeres of such a blast furnace as we have been considering.

If an electric furnace of the type shown in Fig. 2 be used, the volatilization would take place at the base of the shaft of the furnace, and hence the volatilized metals would have a chance to be deposited on the cold part of the charge in the upper part of the stack, and, as it would not be necessary to use a blast when operating the electric furnace for the purpose we have been discussing (as the blast of the blast furnace in this particular instance is used only for furnishing air for burning the coke to produce the heat necessary for the process), the volatilization loss should not, and probably would not, be as great as in the case of the blast furnace.

Loss 4. As to the losses in the slag due to particles of matte or metal, which have failed to separate properly from the slag, such losses would occur just the same if an electric furnace were substituted for the blast furnace, but there is no reason to believe that they would be any greater and on the other hand they should not be as large, for, due to the fact that it is possible to so control an electric furnace as to obtain the temperature desired, a sufficiently high temperature could be given to the matte and slag before it left the furnace to insure its being fluid enough to cause perfect separation in the settler, which would thus obviate this loss. As to the oxidation of valuable metals, this loss should be reduced to a minimum in a furnace operation without a blast, for the loss by oxidation is doubtless due to the action of the blast upon the products of the furnace as they pass the region of the tuyeres.

Having thus made a brief comparative study of the possibility of using an electric furnace for the smelting of mixed sulphide and oxide ores (it being understood, of course, as was stated at the beginning of this paper, that we are not trying to prove that the electric furnace should replace combustion furnaces, but to determine, if possible, whether it can be substituted for the same in those localities which are not favorable to the latter), it may be well at this point to consider some of the experimental work which has been done along the lines above indicated.

## VIII. Experimental Data on the Electric Smelting of Raw Sulphide, Oxidized and Carbonate Ores.

### *1. Vattier's Experiment.*

One of the best known of the experiments which have been conducted upon the smelting of copper ores in the electric furnace is that which was made by Vattier at the Livet works of the Compagnie Electrothermie Keller et Leleux, France, April 23. 1903.[1] These tests were made by Vat-

[1] For a full account of this work see Report of the Canadian Commission, 1904.

tier for the Chilean government, which wished to secure such data as it would be possible to obtain in making experiments on a commercial scale. These tests were made before a Commission, which was composed of some of the best known metallurgists of England and Europe.

As stated by Vattier, the main object of these experiments was to determine, if possible, whether electric heat could be substituted for the heat derived from coal, either wholly or in part. This is especially important to Chile, for it has plenty of valuable ores, and, so it is stated, an abundance of water power, but coal is costly.

The experiments were conducted on two different kinds of ore:

1. An ore the copper contents of which was approximately 7 per cent., present principally as copper pyrites. It also contained 8 to 9 per cent. of sulphur, and the gangue was made up mainly of micaceous copper oxide, together with silicates and a little carbonate of lime.

2. Low-grade copper ore from the mining regions in the vicinity of Santiago, Chile, mixed with a small proportion of manganese and lime.

The composition of the ore, matte and slag was as follows:

| | Ore Per Cent. | Matte Per Cent. | Slag Per Cent. |
|---|---|---|---|
| $SiO_2$ | 23.700 | 0.80 | 27.20 |
| $Al_2O_3$ | 4.000 | 0.50 | 5.20 |
| CaO | 7.300 | ..... | 9.90 |
| MgO | 0.330 | 1.40 | 0.39 |
| $CO_2$ | 4.310 | ..... | ..... |
| S | 4.125 | 22.96 | 0.57 |
| Fe | 28.500 | 24.30 | 32.50 |
| Mn | 7.640 | ..... | 8.23 |
| P | 0.046 | ..... | 0.06 |
| Cu | 5.100 | 47.90 | 0.10 |
| As | Trace | ..... | ..... |

## 2. *Wolkoff's Experiments.*

In some experiments that were made by Wolkoff[1] on the smelting of copper ores, the ore experimented with was a sulphide, containing 8.20 per cent. of copper, with an acid gangue. Several experimental runs were made, but only the results from one or two of them will be mentioned in this connection. For instance, using 12 kg. of ore (25.5 lb.) and with the addition of 6.2 per cent. of hammer scale to furnish iron oxide for the siliceous gangue, he obtained a thoroughly fused product, the matte containing practically the whole of the copper, the slag retaining only 0.15 per cent. of the same, and he also states that the volatilization losses were very light.

In another experiment, he smelted 10 kg. of ore (22 lb.) with 1.25 kg.

[1] *Metallurgie*, Vol. VII, pp. 99 to 102.

(2.74 lb.) of roasted matte, which contained 75 per cent. CuO, 8 per cent. $Cu_2S$, and 15 per cent. $Fe_2O_3$, the object being to determine the extent of the reaction, $Cu_2S+{}_2CuO={}_2Cu_2+SO_2$. On smelting this charge with a current of 400 amperes and 75 volts for half an hour, Wolkoff obtained 1.78 kg. of crude copper with 92 per cent. Cu, 3 per cent. Fe, and 1 per cent S. He also obtained 10.9 kg. slag with 0.10 per cent. Cu. In this experiment 96 per cent. of the metal was extracted; the slag contained 0.6 per cent. of the total quantity of copper; therefore, as stated by Wolkoff, the copper balance is as follows:

| | | |
|---|---|---|
| The charge contained | | 1.706 kg. Cu |
| 1.78 kg. crude copper with 92 per cent. Cu = | 1.638 kg | |
| 10.9 kg. slag with 0.1 per cent. Cu = | 0.011 kg. | |
| Losses | 0.057 kg. | |
| | | 1.706 kg. Cu. |

### 3. *Experiments of M. Stephan.*

At the first general meeting of the recently formed "German Metallurgical Engineers," M. Stephan, Superintendent of the Girod Electric Steel Works at Ugine, France, gave an account of some experiments made by him on the electric smelting of non-ferrous ores, one of which was copper. The following is an abstract of Stephan's paper:[1]

"The ore came from the Belgian Congo, being mined by a Belgian-English concern. In five different analyses given, the CuO varies from 21.01 per cent. to 5.73 per cent.; $SiO_2$ from 28.48 per cent. to 78.55 per cent., with from 4 to 13 per cent. $Al_2O_3$; from 4 to 16 per cent. $Fe_2O_3$, and, besides smaller amounts of impurities, from 2 to 7 per cent. CoO; no nickel. . . . The moisture in the ore varied from 7 per cent. to 32 per cent. It was not removed, in order to meet the conditions of practical operation and to carry the experiments out under most unfavorable conditions. . . . Charcoal, coke and anthracite were used successfully as reducing agents. Charcoal would probably be the cheapest in this particular instance. . . . Electric furnaces, similar to the Girod ferro-alloy furnace type, were used and the dimensions changed in several runs within wide limits, in order to secure enough data for the construction of a large furnace for the same work. Electrodes were used suspended from the top and inserted in the bottom, also in the sides for heating by radiation, and a system was adopted of heating with a smothered arc mainly by the resistance of a thick layer of slag over the conducting metallic charge. The temperatures were measured with LeChatelier, Wanner and Fery pyrometers.

[1] Einiges Über die Erzeugung von Metallen in elektrischen Ofen, von M. Stephan, *Metall. und Erz*, Oct., 1913; abstract in *Metallurgical and Chemical Engineering*, p. 22 (Jan., 1913).

"A slag of the composition—

| $SiO_2$ | $Al_2O_3$ | CaO | MgO | $Fe_2O_3$ | MnO | CuO | CoO |
|---|---|---|---|---|---|---|---|
| 51.90 | 11.31 | 16.83 | 13.71 | 3.55 | 0.94 | 0.46 | 0.87 |

begins to melt at 1250° C.; begins to flow in a pasty condition at 1350° C., and is liquid enough at 1400° to allow the little copper balls to settle completely. At 1550° C. the slag is liquid enough to flow off freely; 1920° C. were necessary to render the slag liquid when the highly acid ore was smelted without any fluxes. The pig copper (*Schwarzkupfer*) produced analyzed in six different runs from 65 per cent. to 95 per cent. Cu, containing from 1 to 21 per cent. Fe, and from 1 to 11 per cent. Co. The purity of the product will be higher the lower the smelting temperature. The lower the temperature the smaller will be the chance for reducing any of its impurities, but, at the same time, more copper will also be retained in the slag, so that the output is decreased."

### A.—*Conclusions of Stephan.*

"It will be a matter of commercial calculation, according to the composition of the ore and other special conditions, whether a high-quality product or a maximum output will be desirable. . . . A continuous run for several days with the same ore, aiming at slags of about the composition mentioned above, required 1000 to 1200 kw.-hr. per ton (2205 lb.) of ore. . . . These figures are high, as a result of the amount of heat required to keep the very viscous and abnormal slags in liquid condition. With an easily fluxible ore, the specific power consumption was only about 500 kw.-hr. The electrode consumption averaged 8 kg., or 17.6 lb., per ton of ore. It was operated with 4 amperes per sq. cm. (25.8 amperes per sq. in.). Coal for reduction was used at the rate of 25 per cent. of the copper in the charge. The best lining to withstand the severe conditions of the furnace was tamped from fire-clay with 80 per cent. $SiO_2$ and 15 per cent. $Al_2O_3$."

## IX.—Smelting of Sulphide Ores in the Electric Furnace.

To study conditions in the smelting of sulphide ores in the electric furnace, the writers have recently conducted a series of experiments at the Bureau of Mines Laboratory. The objects in view were: (1) to determine if there are any conditions arising which make electric smelting without air anything but a simple melting operation; (2) to note the percentage of concentration and the sulphur removal; (3) to study the possibility of condensation of the elemental sulphur as such; (4) to get

general figures on power consumption with varying charges; (5) to determine gold, silver and copper losses, and (6), to study the use of a low-copper matte as a collecting agent for gold and silver.

### *1. Ores Treated.*

The ores used in the experiments were: A heavy sulphide low-grade copper ore; a gold; a silver, siliceous ore, and some roasted ore of the analyses shown below. The limestone contained 63.2 per cent. CaO and 5.7 per cent. MgO.

| | Pyrite Per Cent. | Nodule Per Cent. | Siliceous Ore Per Cent. |
|---|---|---|---|
| $SiO_2$ | 2.50 | 4.00 | 74.53 |
| Fe | 44.07 | 65.60 | 10.71 |
| S | 48.20 | ..... | 8.36 |
| $Al_2O_3$ | 0.24 | 1.76 | 0.40 |
| CaO | 0.08 | 0.45 | ..... |
| MgO | 0.73 | 0.46 | 0.26 |
| P | 0.02 | ..... | ..... |
| As | 0.20 | ..... | ..... |
| Cu | 1.30 | 0.07 | ..... |
| Au | 0.01 oz per ton | 0.03 oz. per ton | 1.28 oz. per ton |
| Ag | 0.14 oz per ton | 0.05 oz per ton | 3.20 oz. per ton |

The ores were treated in the electric furnace described under the native copper experiments. The top was roofed and kept closed tightly to prevent escape of sulphur and admission of air. A condenser, consisting simply of three rectangular, horizontal chambers with several baffles in it, was attached to the furnace to condense the sulphur and catch any escaping dust.

Twenty experiments were made with these ores. A typical charge was as follows:

| Charge | | Matte | | Slag | |
|---|---|---|---|---|---|
| | *Pounds* | | *Per Cent.* | | *Per Cent.* |
| Pyrite | 8.8 | Cu | 1.22 | Cu | 0.05 |
| Nodule | 13.2 | Fe | 64.18 | $SO_2$ | 35.[illegible] |
| Siliceous ore | 5.76 | S | 22.38 | FeO | 41.3[illegible] |
| Limestone | 2.73 | | *Oz. per Ton* | $Al_2O_3$ | 4.[illegible] |
| | | Au | 0.72 | CaO | 10.[illegible] |
| | | Ag | 0.96 | MgO | 1.6[illegible] |
| | | | | S | 3.[illegible] |
| | | | | | *Oz. per Ton* |
| | | | | Au | 0.[illegible] |
| | | | | Ag | 0.[illegible] |

### 2. *Conclusions.*

(1) Smelting in an electric furnace treating sulphide ores consists simply of melting the charge, volatilizing about 60 per cent. of the sulphur as elemental sulphur, and separating the slag and matte.

(2) The ratio of concentration is simply that possible from a separation of matte and slag, together with a reaction of oxides and sulphates upon sulphides, and the volatilization of 10 per cent. more than one atom of sulphur, but with no formation of iron oxide to enter the slag, because there is nothing present to oxidize the iron sulphide.

(3) Qualitatively, it is possible to condense some of the elemental sulphur driven off.

(4) The loss of copper, by volatilization and in the slag, is low.

(5) A matte containing about 1 per cent. copper, from a charge containing 0.30 per cent. copper, makes a good collecting agent for gold and silver, if a clean separation of slag and matte is obtained.

(6) There is very little loss by volatilization of gold in the electric furnace, but there is some loss of silver in this way. This would not probably occur in a larger furnace with closer temperature regulation.

(7) The electrode consumption in the smelting of a sulphide ore is low, and in practice need not exceed five pounds per ton of charge.

(8) In a large commercial furnace the power consumption for most ores would be about 480 kilowatt-hours per ton of ore, or 0.055 kilowatt-years.

## X.—Electric Furnace vs. the Reverberatory Furnace and Blast Furnace as a Melting Agent.

So far, we have only considered the use of the electric furnace as a melting furnace in the treatment of copper ores. From what has been said, we believe that we are justified in making the following statements as to the possibility of using the electric furnace for this purpose:

(1) The melting of native copper oxide or sulphide ore of copper can be done just as efficiently in the electric furnace, and perhaps more so, than in either the reverberatory furnace or the blast furnace.

(2) The reactions desired in reverberatory smelting, or ordinary blast furnace smelting, can be obtained in the electric furnace as well, and perhaps better, than in either of those furnaces.

(3) The loss of electrodes is small, varying from 5 to 10 lb. per ton of ore smelted, and the presence of the carbon electrode does not cause enough reduction of iron to be troublesome, or increase the consumption of electrical energy appreciably.

(4) The losses of copper, gold and silver by volatilization and in the slag would be no greater, as shown in all the experimental work cited,

than they are in reverberatory smelting or ordinary blast furnace smelting.

(5) A matte containing as low as 1 per cent. copper can be used as collecting agent for gold and silver in the electric furnace, as well as in combustion furnaces.

(6) The comparison of costs would depend entirely upon the nature of the ore treated, and the relative cost of coal or coke, and electrical energy. In general, from the work done in the experiments, we may say that from 500 to 700 kilowatt-hours per ton charged would be required for smelting copper ore, depending upon the nature of the ore.

## XI.—Smelting of Sulphide Ores without Roasting, or Partial Pyritic Smelting.

In discussing the smelting of copper ores by ordinary blast furnace smelting, we noted the fact that the process consists essentially in melting down the charge, and that, during the melting, certain reactions take place between the constituents of the charge, which bring about a concentration of the copper and precious-metal values (if the latter are present) into a matte, and the excess iron and gangue material into a slag; that, in this method of smelting, the heat necessary for carrying out the process was obtained by the combustion of coke at the tuyeres, and that the blast used was simply for the purpose of supplying the necessary oxygen for combustion. In other words, the oxygen entering the tuyeres plays no part in the reactions which take place in the furnace, so far as the constituents of the charge other than the coke, are concerned. Such being the case, the process is simply a melting down process and, so far as the reactions of the process are concerned, the melting down could just as well be done in an electric furnace as in a blast furnace. Now, however, we turn our attention to those processes of copper smelting where the air entering at the tuyeres not only furnishes the oxygen necessary for the producing heat, but also for bringing about the necessary reactions which take place during the progress of the process, and which result in the production of a matte and a slag, as in ordinary blast furnace smelting. In this connection it is to be remembered that no matter what process we employ in obtaining metallic copper (as a final product) from a sulphide, the three essentials are:

(1) The removal of sulphur.

(2) The oxidation of the iron with which the sulphur was combined.

(3) The removal of this iron oxide by causing it to unite with silica to form a slag.

Hence, as stated by Peters, "the first step in the smelting of a sulphide ore of copper must be to oxidize it." In the process which we are now

to consider, the object is to bring about the required degree of oxidation and the melting, at practically the same time, and to obtain either a whole or a part of the heat required for the process from the combustion of the iron and sulphur contained in the ore. As is well known, if practically all the heat necessary for the process is obtained in this manner, we have what is known as pyrite smelting. If only a part, semi-pyrite smelting. However, Peters states that so far as he is aware, in all pyrite furnaces of the world, a small amount of fuel (usually coke) is added to the charge, and "this amount may vary anywhere from 0.5 per cent. of the weight of the charge up to a proportion that might be actually enough to melt the ore without any heat at all being derived from the sulphides." The question, therefore, that presents itself in connection with the smelting of copper ores in the electric furnace is to determine whether electric heat may be used to replace the heat which is derived from the combustion of coke in pyrite and semi-pyrite processes. In other words, whether it would be possible, and if so, whether it would be commercially feasible, to carry out these processes in an electric furnace so constructed as to use a blast, thus obtaining as much heat as possible from the oxidation of the sulphur and iron, and supplying by electric energy whatever additional heat might be needed to carry out the process successfully.

To begin with, let us assume that our furnace is similar in construction to a modern copper blast furnace, and that part of the furnace, including the tuyeres, is practically identical with the same. Below the tuyeres, the furnace could be constructed as shown in Fig. 3. By referring to this, it will be noted that there are electrodes extending down into the crucible, the arrangement of which, along the sides of the crucible, can be noted by referring to the plan of the furnace in Fig. 4. Let us now briefly consider what would happen if we attempted to smelt a charge which would be suited to "partial pyrite smelting," that is, one in which so much coke would have to be used that "its influence upon the oxidizing power of the focus begins to be plainly discernible," which, as stated by Peters, is "the division line between true and partial pyrite smelting." Inasmuch as no coke would be used in a charge smelted in an electric blast furnace, the point to be determined is how the charge would act when such a furnace was used, and if the desired results could be obtained, as before stated, in a feasible and economical manner.

The object aimed at in operating a copper blast furnace, when working under partial pyrite conditions, is to supply enough blast to completely oxidize the coke, and also to oxidize as much of the iron sulphide for slag forming purposes as can be spared from the matte. If, therefore, there were no coke to be oxidized, only enough air would be forced through the tuyeres to effect the required degree of oxidation of sulphur and iron.

In general, we may assume, as does Peters, that a partial pyrite furnace

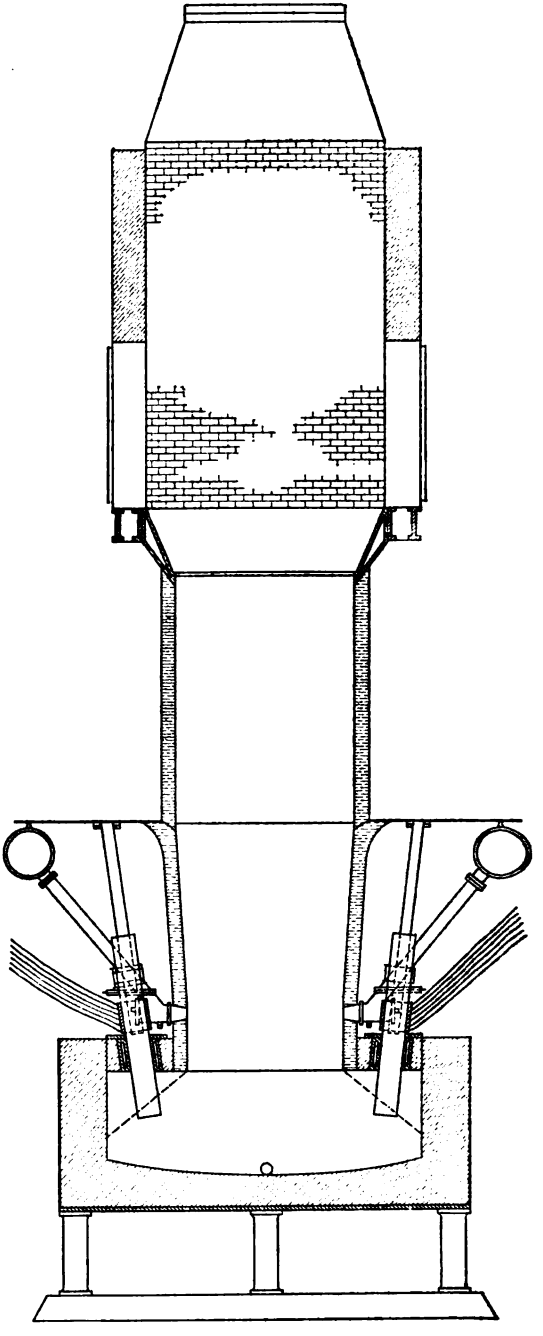


FIG. 3.—ELEVATION OF ELECTRIC BLAST FURNACE.

is "provided with but a scanty supply of pyrite," and that it produces a slag "rather high in silica and earths," and, "consequently would require a considerable amount of coke." In the electric blast furnace, as before stated, we would not use coke, but electricity instead for obtaining the equivalent heat value of the coke. If, therefore, we supply our electric blast furnace with a charge, such as indicated above, that is, a charge such as could be successfully treated in a partial pyrite furnace, what differences may we expect in the behavior of the charge as compared to

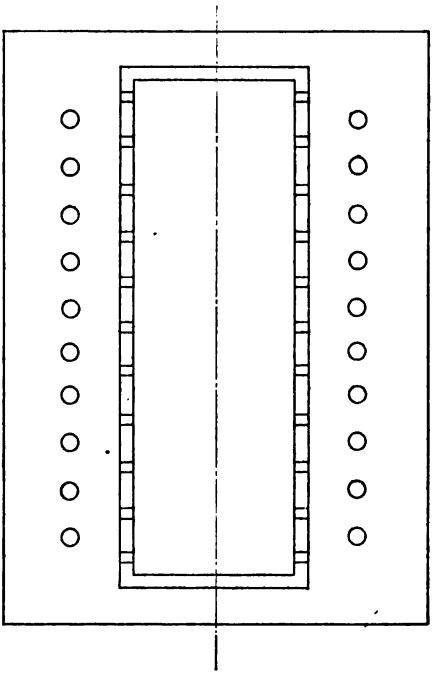


FIG. 4.—PLAN OF ELECTRIC BLAST FURNACE

the behavior of the similar charge when smelted in a copper blast furnace with coke? In making the comparison, let us first study the behavior and effect of the coke itself. Again quoting from Peters[1], he states that "the amount of coke present carries the melting process high up above the proper zone of oxidation, and to regions where there is yet no formation of FeO. Half-fused masses of acid, earthy silicates are formed, and much—in some cases, all—of the free silica is combined with the alumina, lime, magnesia, manganese, alkalies, and already oxidized iron, all of which substances are likely to be present in the ore mixture. It is not that the affinity of the silica is satisfied in forming these preliminary,

[1] *Principles of Copper Smelting*, p. 333.

temporary, refractory acid silicates; but the edge of its appetite is blunted, and the eventual formation of ferrous silicate seems to proceed somewhat sluggishly, even when ample air is blown into the furnace.

"As may be imagined, the main endeavor of the metallurgist in this type of smelting is to keep the proportion of coke to ore as low as possible; not merely because coke costs money, but still more because any excess of coke causes a lowering in the grade of the matte, due to its interference with the oxidation of the sulphides. A furnace in proper condition, and running on a suitable charge, is extraordinarily sensitive on this point. The increase of the coke from a standard charge of 60 lb. per 2,000 lb. of ore up to 65 lb. per 2,000 lb. of ore may be followed—as has come under my personal observation—by a dropping of the grade of matte from 35 per cent. copper, and a raising of the silica contents of the slag from 41 per cent. This results, of course, from the consumption by the new coke of a certain amount of oxygen, which previously had been employed in burning iron sulphide. Under this new condition, this iron sulphide entered the matte, at the same time robbing the slag of just so much FeO."

If now we smelt the charge in the electric furnace without coke, and with just enough blast to perform the necessary oxidation of the sulphur and iron, there would doubtless be no "half-fused masses of acid, earthy silicates formed," etc., as stated by Peters, and since, as shown by him, the control of the coke is a matter of great importance, due to the "extraordinary sensitiveness of the furnace on this point," and its resultant great irregularities, it would seem that it would be a great advantage if possible, to do away with coke. If so, and the smelting be done in some such furnace as shown in Fig. 3, we would expect the charge to enter the tuyere zone of the furnace practically unaltered, except, perhaps, for loss of elemental sulphur. Inasmuch as the sulphides are easily melted (950° C.) we may assume that there is enough heat present, from the oxidation of the sulphur and iron, and radiated from the crucible, to cause the iron sulphide to melt and to become oxidized to FeO, and then be instantly seized by $SiO_2$ to form a slag. However, at this temperature, the result might be the formation of a silicate, high in $SiO_2$, because, at low temperatures, the saturation point of silica for iron is low, and which, perhaps, would at once freeze, so to speak, and descend into the crucible and be gradually melted in the neighborhood of the electric current. In other words, we would have formed above the tuyere zone the artificial boshes, observed by Sticht, Freeland and others, and described by Peters, thus contracting the smelting and slag forming area "to a comparatively narrow opening, running along the middle of the furnace shaft." Sticht states that these boshes "are the contact lines between the active combustion zone and the relatively dead part of the shaft above," and that the

absence of coke (speaking of pyrite furnaces) accentuates the sharp division line between the active and stagnant regions of the furnace. Reasoning by analogy, it would seem that this division line would be as pronounced (if not more so) in operating an electric blast furnace, such as we are discussing, than would be the case in the pyrite furnace.

## XII. The Electric Blast Furnace vs. the Copper Blast Furnace for Partial Pyritic Smelting.

### *1. Metallurgical Considerations.*

To recapitulate, there does not seem to be any metallurgical reason why the chief objects of the pyrite smelter cannot be carried out in an electric copper blast furnace as well as in a coke copper blast furnace; namely, "to melt the great mass of $SiO_2$ and inert earths, to oxidize enough of the sulphides in the charge to insure a suitable matte—incidentally obtaining the heat evolved by the oxidation." On the other hand, it would seem that the difficulties ordinarily encountered in operating a pyrite furnace might be avoided when using an electric blast furnace, namely, the difficulties which arise when too much or too little coke is added to the charge, and moreover that it not only permits "introducing heat into the furnace, without at the same time robbing the combustion zone of oxygen," of which Dr. Peters states, "nothing would be so welcome to the furnace man as to do this," but would also permit of the heat being entirely under control and easily regulated, and thus avoid freeze-ups with their consequent vexations and costly delays. By this we do not mean that any sort of a charge could be put through the furnace and not freeze it up, an idea which seems to be quite prevalent in regard to electric furnaces, but that, if the charge be properly calculated, a very much wider variation would be permissible in the composition of the slag from that calculated, than would be the case in blast furnace smelting.

### *2. Mechanical Considerations.*

By referring to Figs. 3 and 4, it will be noted that the chief difference in the construction of the furnace would be in that part of it below the tuyeres,—in other words, in the crucible. Because electric furnace construction has received the attention of some of the very best mechanical and electrical engineers, and because a crucible based upon the principle shown in the design in Fig. 3 is now extensively used in the iron electric reduction furnaces of Norway and Sweden, we believe that no difficulty would be experienced in this respect. Various methods and arrangements

can be used for connecting up electrodes. If this be true, we have only to consider the matter of costs.

### 3. *Comparison of Costs.*

#### a. *General Considerations.*

From what we have said, it will be evident that a comparison of costs is made simply for the purpose of giving some idea as to the outlay that would be necessary to erect and operate an electric furnace plant for the purpose of smelting copper ores.

Naturally, the first thing to be considered is whether it would be feasible to attempt to substitute electrical energy for coke, due to the cost of the former.

#### b. *Cost of Plant.*

As a matter of fact, the cost of an electric furnace plant would be the cost of a regular plant plus the electrical installation, exclusive of the generating plant, because it is assumed that it would be possible to purchase power from some power company, and, if not, the generating plant would probably be considered as a separate organization, selling power to the smelter at so much per unit. Therefore, by electrical installation we mean the cost of transformers, bus-bars, cables, instruments, etc. In order to get some idea of this cost we will assume that our furnace is to smelt 384 tons of charge per day[1] and that the composition of the charge is:

| | *Per Cent.* |
|---|---|
| Cu | 5.21 |
| $SiO_2$ | 26.41 |
| FeO | 18.60 |
| S | 11.46 |
| $Al_2O_3$ | 4.26 |
| CuO | 17.49 |

This was the average composition of the entire blast furnace charge which was smelted at the Washoe smelter, Anaconda, for several months.[2] In this plant 8.2 per cent. of coke was used. If we should attempt to smelt a charge of this nature in an electric furnace, what amount of electrical energy would be required?

#### c. *Amount of Electrical Energy Required to Replace Coke.*

Since 8.2 per cent. of coke was required at the Washoe smelter to smelt the above charge:

[1] At the Washoe smelter, Anaconda, the furnaces were formerly 56 by 180 in. (hearth area, 70 sq. ft.) and smelted on an average a little less than 400 tons of charge daily, or 5.6 tons per square foot. From *Principles of Copper Smelting*, Peters, p. 156.

[2] *Practice of Copper Smelting*, Peters, p. 267.

$2{,}000 \times 8.2 = 164$ lb. of coke, were used. This coke contained 80.24 per cent. fixed carbon; therefore, the carbon equivalent of the coke is: $164 \times 80.24 = 131.59$ lb. If one pound of carbon be completely burned to carbon dioxide, we obtain 8,100 lb. cal. Therefore, $131 \times 8{,}100 = 1{,}061{,}100$ lb. cal. that would be obtained from 131 lb. of carbon contained in the 164 lb. of coke. As 1 kilowatt hour is equivalent to 1,897 lb. cal., it would require: $\frac{1{,}061{,}100}{1897}$ or 559 kw. hr. to replace the theoretical calorific value of the 164 lb. of coke. However, as a copper blast furnace at its best probably does not have an efficiency of over 50 per cent., the energy really obtained from the coke, so far as work performed in the furnace is concerned, is only 530,550 cal. and this would represent theoretically, $\frac{530{,}550}{1897}$ or 280 kw. hr. per ton. Inasmuch as the efficiency of electric furnaces of this type may be as much as 85 per cent., and as that of an open top ferro-silicon furnace is said to be about 60 per cent.,[1] it would be fair to assume that the efficiency of an electric furnace, such as shown in Fig. 2, would be 20 per cent. greater than that of the ordinary blast furnace, and hence we will say it is 70 per cent. If this be true, then the theoretical 280 kw. hr. needed to smelt one ton of charge would have to be increased 1.42 times ($280 \times 1.42 = 397$) or say in round numbers, 400 kw. hr. would be required to smelt one ton of charge. Considering that part of the heat in this case is supplied by combustion of sulphide the figures agree closely with the 480 kilowatt hours calculated from the experiments on melting only. As we are to smelt 10 tons of charge per hour, $400 \times 16 = 6400$ kw. hr. would be the constant load on the furnace, and, as the power supply would doubtless be three-phase, this would mean that the electrical equipment should consist of:

Three 2500-kw. variable voltage transformers, to allow ample margin for sudden overloads, etc.; instruments, bus-bars, cables, etc.

As to their costs at the plant, that would, of course, depend upon the locality, and would be the factory costs plus transportation charges. To partially offset this cost, the blower capacity would not need to be so large as in the case of the regular blast furnace, because less air is blown.

### *d.—Cost of Smelting.*

We may say that the costs of coke and electrical energy are about on a par when coke costs $7 a ton, and electrical energy 0.15 cents per kilowatt hour, or $13 per kilowatt year,—or say in the ratio of 1: 1.8. Although there may be advantages in the use of an electrical blast furnace instead of an ordinary blast furnace, these will not be taken into consideration in this connection, and we will assume that the cost of electrodes and of the electricity on the one hand, and its equivalent in heating value

[1] *Revue de Métallurgie*, vol. IX, p. 362 (1912).

of coke on the other, would be the same; in other words, that the cost of electrodes and electricity would balance the cost of the amount of coke that would be necessary to do the same amount of work. We will not take into consideration the saving in blowing by reason of not having to furnish air for burning the coke.

### *e.—Cost of Electrodes.*

The average consumption of electrodes per ton of iron produced at Trollhattan, Sweden, is about 10 lb. Assuming 1 ton of iron is equivalent to 2 tons of charge, the consumption of electrodes per ton of charge smelted in a furnace, such as the one we have been discussing, would not be over 5 lb. per ton. If we assume that the electrodes do not cost over 6 cents per pound, which allows a fair margin for transportation over their cost at the factory, the cost of electrodes would be about 30 cents per ton of charge smelted. The consumption would be probably considerably less than 5 lb. per ton, as was shown in an experiment performed by the writers in which air was blown into the furnace, and in which the electric consumption was only 2 lb. per ton of ore treated. Such being the case, if our combined electrical energy and electrode cost is not to exceed our equivalent coke cost, with coke at $9 a ton, the cost of electrical energy would be $9 × 1.8 = $16.20 a kilowatt year, minus the cost of electrodes, or about $16.00. Compared on the basis of calories, if the coke contains 80.24 per cent. fixed carbon, it would contain 1604.8 lb. of carbon per ton of 2000 lb., and this carbon would, if completely burned to $CO_2$ give a calorific value of about 13,000,000 calories. As one kilowatt year of energy is equal to 16,616,000 cal., it would require something like 1.2 tons of coke, containing 80.24 per cent. fixed carbon, to equal in calorific value the kilowatt year, but it is to be remembered that this assumes that all the carbon of the coke is completely burned to carbon dioxide, which rarely happens in practice, and so, as has been determined from actual practice, it requires from 1.5 to 1.8 tons of coke to yield the same number of calories as may be obtained from 1 kilowatt year of electrical energy.

## XIII.—The Use of the Electric Blast Furnace in the Smelting of Pyritic Ores.

As previously stated, in all true pyritic smelting, fuel (usually coke) in varying amounts is added to the charge. We will assume for the sake of argument, that the ore being treated is entirely suited to pyritic smelting, and that it is not necessary to use more than 0.5 per cent. of the weight of the charge in coke. Could such an ore be treated pure as well

in an electric blast furnace as in an ordinary blast furnace, assuming that the costs would be the same in each case. As what we have had to say in regard to the construction of the furnace and the costs of applying the process is as applicable to true pyritic as it is to semi-pyritic smelting, we will only take up the subject from the metallurgical and chemical standpoint.

In the first place let us consider the nature of the atmosphere in a pyrite furnace. As the air enters the tuyeres, practically all of it enters into immediate combination, forming $SO_2$ and FeO. Although some oxygen escapes combination, it is not sufficient to support combustion within the furnace, as has been shown by Sticht and others. Now let us briefly consider the changes which take place in the charge as it descends from the charging door to the fusion zone. They are few, and may be briefly stated as follows:

(1) The iron sulphide loses one-half of its sulphur as elemental sulphur, and becomes FeS.

(2) Chalcopyrite. It loses about one-fourth of its sulphur and becomes practically a mixture of $Cu_2S$ and FeS.

(3) The silica is unchanged.

(4) The coke is for the most part consumed in the upper zone of the furnace, but not by oxygen, as no free oxygen exists in that part of the shaft, but by the O from $SO_2$, that is, the carbon of the coke reduces the $SO_2$ giving S and CO, and this CO reacts with more $SO_2$, forming S and $CO_2$. Aside from these reactions, which are of no great importance in the chemistry of the process, the chief office of the coke as pointed out by Sticht,[1] "Is apparently to heat up the sulphides and the quartz in preparation for their active oxidation deeper in the furnace." If, therefore, coke was omitted from the charge we would not have this preparation of the sulphides and the quartz as stated by Sticht. Such being the case, how detrimental to the process would this lack of preparation be?

In order to answer this question, it must be borne in mind that in true pyritic smelting, there is "no heat to spare," and that although, as pointed out by Sticht, the heat evolved by the combustion of the coke by the oxygen of sulphur dioxide is only one-third as much as it would be if the oxygen were furnished direct by the blast, and that it is given off at a point considerably higher than the focus of the furnace, it nevertheless "assists in preparing the charge for the reactions which take place in the focus of the furnace, for with 1 per cent. of coke in the charge enough heat is generated to supply one-third of the heat required to melt the entire pyrrhotite contents of the charge." This extra heat "doubtless furnishes just the necessary aid to bridge the operation over some critical

[1] *Principles of Copper Smelting*, by E. D. Peters, p. 227 (1907).

point." From this we see that the office of the coke then is solely for the purpose of furnishing heat. Inasmuch as the nature of the charge in true pyritic smelting is such as to demand the oxidation and removal of a large excess of iron as silicate, and as it is necessary to cause the silica present to unite with as much FeO as possible (due to the fact that generally silica-bearing materials have to be added for slag-forming purposes, and often these are barren and hard to obtain), and as the saturation point of silica with iron oxide is greater the higher the temperature attainable in the pyritic furnace, it is quite necessary that the reactions in the zone of the furnace proceed as energetically as possible, in order to meet the requirements stated above, and to provide enough heat to keep the process going. If, therefore, pyritic smelting be attempted in an electric blast furnace, it would be necessary to supply the heat from the electric current in such a manner as to insure there being enough heat at the focus to carry on the reactions at that point in the same manner as if coke had been added to the charge. The writers are of the opinion that it would be possible and perfectly feasible to do this in the furnace of the type shown in Fig. 3. Moreover, that the use of the electric furnace would permit of a more uniform operation of the furnace than is now possible, for, with the electric current, it would be an easy matter to quickly increase or decrease the heat required for the successful operation of the furnace. In this way we could maintain the "degree of concentration at a constant," which, as pointed out by Sticht, is very important, but which is extremely difficult to do. Also, we could avoid the troubles arising from the "constant fluctuations" which are bound to occur, and which at present demand unbroken attention and frequent changes in the charge, especially in its proportion of silica "due to the fact that in true pyritic smelting there is no heat to spare," whereas, with the electric blast furnace, it would be possible, as just stated, to increase or decrease the heat as necessary in order to meet these fluctuations.

## *Summary.*

Summarized then we may say that the electric smelting of copper ores is nothing more than the substitution of electric heat for the heat derived from the combustion of carbon. Inasmuch as the carbon which is used either in the reverberatory furnace or in the blast furnace plays no important part in the necessary reactions which take place in these furnaces, there is no reason, metallurgically, why electric heat may not be substituted for the heat derived from the combustion of carbon. In fact, as we have tried to point out, in some cases the reactions would take place to better advantage in the neutral atmosphere of the electric furnace than in the reducing or partially reducing atmosphere of the com-

bustion furnace. Therefore, as to whether the electric furnace would be used for the smelting of copper ores would largely depend, so far as we are able to make out at the present time, upon the relative cost of coke and electric power. As the use of the electric furnace is not advocated as a competitor of the combustion furnace, but as a substitute for it in those localities where it is not advisable because of the high cost of fuel, we see no reason why the electric furnace may not be developed as a substitute for the combustion furnace, where the conditions are such as to warrant its use, especially in the treatment of copper-bearing ores. In this connection it is to be remembered that the reason why the electric furnace was developed in the iron industry, for the reduction of iron from its ores was due to necessity. As a matter of fact the field for the electric furnace in the reduction of iron from its ores is a limited one. Perhaps the same is true as regards the possible application of the electric furnace to the treatment of copper ores, but, judging from the comparative costs as shown in the preceding pages, it would seem that the chances in favor of the electric furnace for the treatment of copper ores are greater than those for the treatment of iron ores, because there is not so great a difference in the cost of coke and electric power in copper mining districts as in iron smelting centers. Also the cost of electric power is constantly becoming less, due to improvements in gas engines and steam turbines, so that, in districts where water power is not plentiful but cheap fuels unsuited to coking purposes are available, it may be found more advantageous to use electric heat than the heat derived from the combustion of coke.

It is sincerely hoped by the writers, that although it is possible at this time to present but few facts, that the comparative study herewith presented may serve to stimulate interest in the subject of the electric smelting of copper ores, and to cause others to attempt a further investigation of it. As a result of their investigations, the writers are convinced that experimental work on a larger scale should lead to the development of an electric blast furnace, which in some cases could be used to better advantage for smelting sulphide ores of copper than the combustion blast furnace.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS.
[SUBJECT TO REVISION].

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Butte meeting, Aug. 18 to 21, 1913, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Oct. 1, 1913, when Vol. XLVI. of the *Transactions* will go to press. Any discussion offered thereafter should preferably be in the form of a new paper for publication in Vol. XLVII. (with suitable cross references in both volumes).

# The Anaconda Classifier.*

BY ROBERT AMMON, GREAT FALLS, MONT

(Butte Meeting, August, 1913.)

THE purpose of this paper is to present a brief sketch of the development of this hindered-settling classifier, but primarily to show the actual results obtained in practice with the classifier working on the Butte copper ores at the Boston & Montana Concentrator at Great Falls and the Washoe Concentrator at Anaconda. The writer has endeavored to be as brief as possible, giving only the essential figures showing the efficiency of the classifier and leaving considerable interesting information to be derived by the reader from the statistics given.

The paper deals with the practical work of the classifier under the following heads:

1. Desliming 4.0-mm. primary feed, overflow from 0.07 to 0.0 mm.
2. Desliming 1.25-mm. secondary feed, overflow from 0.07 to 0.0 mm.
3. Desliming Huntington Mill discharge, overflow from 0.07 to 0.0 mm.
4. Classifying 2.5-mm. deslimed secondary feed, overflow from 1.00 to 0.07 mm.
5. Classifying 2.5-mm. deslimed primary feed, overflow from 0.75 to 0.07 mm.
6. Classifying 4.0-mm. deslimed primary feed, overflow from 0.75 to 0.07 mm.
7. Classifying middling-tailing product from roughing tables, overflow coarse tailing.
8. Classifying 2.5-mm. trommel undersize, not deslimed, overflow from 0.75 to 0.0 mm.
9. Remarks on the inner feed cone.

In the following pages all overflows are given in quartz dimensions; *i. e.*, by a 0.75-mm. overflow is meant an overflow whose maximum particle is a 0.75-mm. quartz grain. By the term "slime" is meant all material which will pass through a 200-mesh screen whose average opening is 0.07 mm.

The development of the classifier was commenced in the fall of 1910,

* U. S. Patent No. 1,058,828, issued April, 1913, to A. E. Wiggin, A. E. Wheeler, and R. H. Richards.

NOTE: In order to have printed copies of this paper available for use at the Butte meeting, it is here published without Figs. 5 and 7 of the illustrations, which have not been furnished for publication.

and was primarily the result of a suggestion by Dr. Robert H. Richards, that a conical tank equipped with a pulsating current would deslime the feed to his direct pulsator classifier then installed in No. 4 Section of the Great Falls Concentrator. The success of this arrangement in removing the slimes from ore pulp, together with the demand for a deslimed feed to the Hancock jigs,[1] led to a series of experiments initiated by A. E. Wiggin in the fall of 1910. The final result of these and later experiments has been the production of the Anaconda hindered-settling classifier, which has as its chief assets the three leading factors to be reckoned with

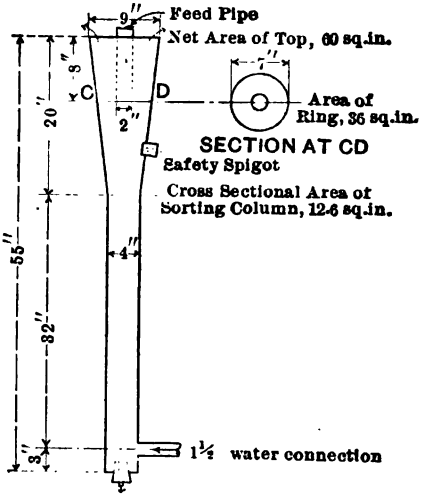



FIG. 1.—FIRST ANACONDA CLASSIFIER BUILT.

in any machine: namely, simplicity of construction, large tonnage, and high efficiency.

In criticizing the work of any classifier it must be remembered that we are dealing with grains of material of every conceivable size and shape from flat to round grains, and with varying specific gravities, from the purest and lightest gangue to the purest and heaviest mineral grains, together with the innumerable combinations of the same. If a classifier were mechanically perfect it still would be impossible to separate a large flat grain of gangue from a much smaller rounded grain of rich middling.

Fig. 1 shows the first form of the classifier constructed. The early experiments proved that the pulsating current was of no advantage in this type of machine, hence it was abandoned for a plain rising current

[1] See paper to be presented at this meeting by Albert E. Wiggin, entitled, The Great Falls System of Concentration installed in Section No. 1 at the Washoe Concentrator, p. 1801, this *Bulletin*.

The classifier consisted of merely a vertical pipe for a sorting column with a superimposed inverted truncated cone, the hydraulic water being admitted at one side near the bottom of the sorting column at right angles to the spigot discharge.

The experiments immediately following consisted chiefly in varying the dimensions of the sorting column and the conical top, which will be referred to hereafter as simply the "top."

The object sought in these earliest forms was merely a deslimer which would give an overflow practically all through 200 mesh and at the same

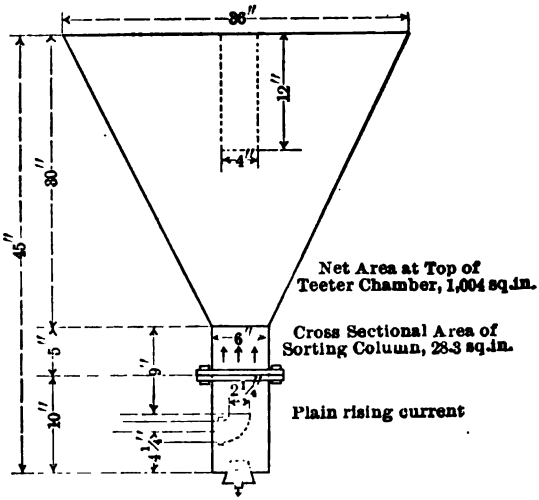



FIG. 2.—ANACONDA CLASSIFIER. ONE OF THE INTERMEDIATE FORMS.

time give a clear spigot free from slime. Many forms were tried. Fig. 2 shows one of the intermediate stages.

The hydraulic water was now admitted at 180° to the spigot discharge by means of a 90° elbow at the center of the sorting column. The work of this particular stage, while an improvement over earlier forms, was very crude when compared to the present form of deslimer.

| | |
|---|---|
| Tons treated per 24 hr | 108 |
| Per cent. total feed overflowed | 12.0 |
| Per cent. solids in overflow on 200 mesh | 45.7 |

## LABORATORY TEST ON MODEL SIZE THREE-POCKET UNIT.

Fig. 3 is a sketch to scale of a unit of three pockets in series, built for the purpose of determining in the laboratory the relative velocities of rising currents required in the sorting columns to make the proper separations other than at 0.07 mm., and particularly at 1 mm., as this point

had been decided upon as the point of division between Wilfley table feed and Hancock jig feed in the mill flow sheet. The velocities shown on the sketch are the theoretical velocities and not the actual values. The actual velocities could not be determined in the experiment because of the lack of suitable means for measuring accurately such small quantities of water as were used by the laboratory classifiers. The feed to the unit consisted of the undersize of a 5-mm. round-hole trommel screen.

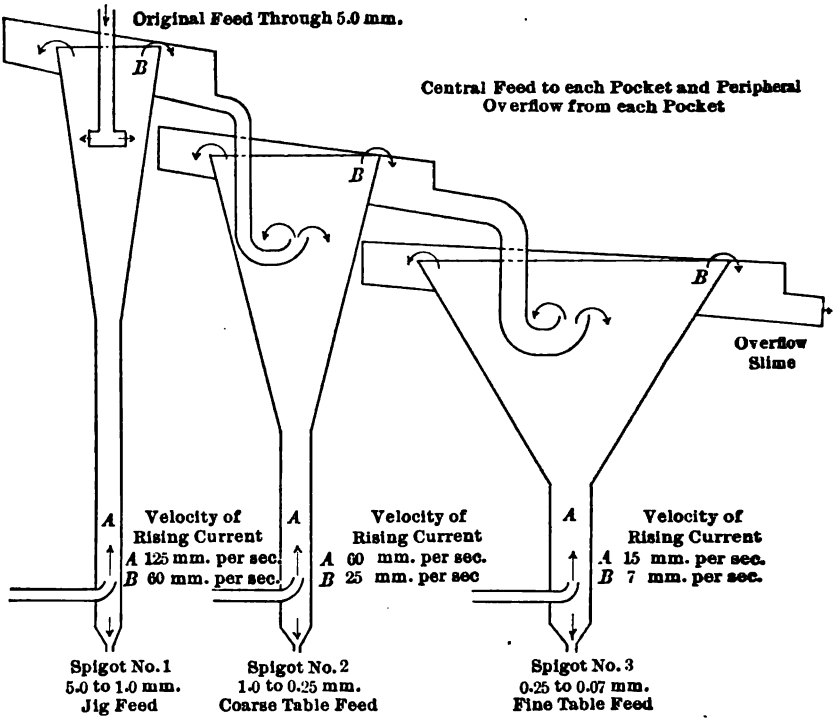



FIG. 3.—THREE-POCKET UNIT, LABORATORY SIZE.

The sand was fed dry to the first classifier by means of an automatic shaking feeder. The rising current in the sorting column of the first classifier was adjusted so that the dividing line between the first and second pockets was at about 1 mm. It was attempted to keep the overflow of the final pocket entirely through 200 mesh. The three spigot products and the overflow of the final pocket were screen sized, and each size was sorted into free mineral grains and quartz or middlings grains. The coarser sizes, from 4 mm. to 2 mm., were sorted by hand, while the finer sizes, between 2 mm. and 1 mm., were treated in a sorting tube, and the product below 1 mm. was concentrated on a laboratory size Wilfley table.

The amount of free mineral in the overflow (through 200 mesh) was estimated. The results of the test are shown in Table I and Fig. 4.

The classification chart, Fig. 4, which brings out the degree of classification quite distinctly, was worked out by C. R. Kuzell.

The free mineral overlaps on three screen sizes, 50, 60, and 90 mesh. The quartz grains in Spigot 1 which are finer than 1 mm. were fairly rich middlings grains whose specific gravities were greater than for those grains in Spigot 2 on 60 mesh, the latter being for the most part grains of quartz containing little or no mineral, and probably the bulk of these were flat grains. The feed was distributed as follows:

| | Solids in Total Feed, Per Cent. | Free Mineral in Total Feed, Per Cent. |
|---|---|---|
| Spigot No. 1 | 61.1 | 55.2 |
| Spigot No. 2 | 19.2 | 21.2 |
| Spigot No. 3 | 3.3 | 4.1 |
| Overflow | 16.4 | 19.5 |
| TOTAL | 100.0 | 100.0 |

## THE ADOPTION OF THE CONSTRICTION PLATE.

Up to this point no form of constriction had been used in the sorting column; and although a teetering mass had been observed in the column at times, there is no doubt in the writer's mind but that this was due to overcrowded free-settling condition and not to true hindered-settling. To produce strictly hindered-settling condition a constriction was adopted in the form of a thin plate of steel with a central circular opening. The result was to produce a teetering mass above the opening in the plate (sorting column), which will be called the teeter chamber. Later the depth of the sorting column was increased by using a nipple 2 in. long. The object of using a deeper sorting column was to do away with local eddy currents and to take up more or less the slight fluctuations in the hydraulic pressure. This lengthened sorting column increased the efficiency considerably, for at this point is determined whether a grain shall rise into the overflow or pass out the spigot. The teeter chamber had now been increased to 12 in. in diameter, and this diameter was adopted as a standard size.

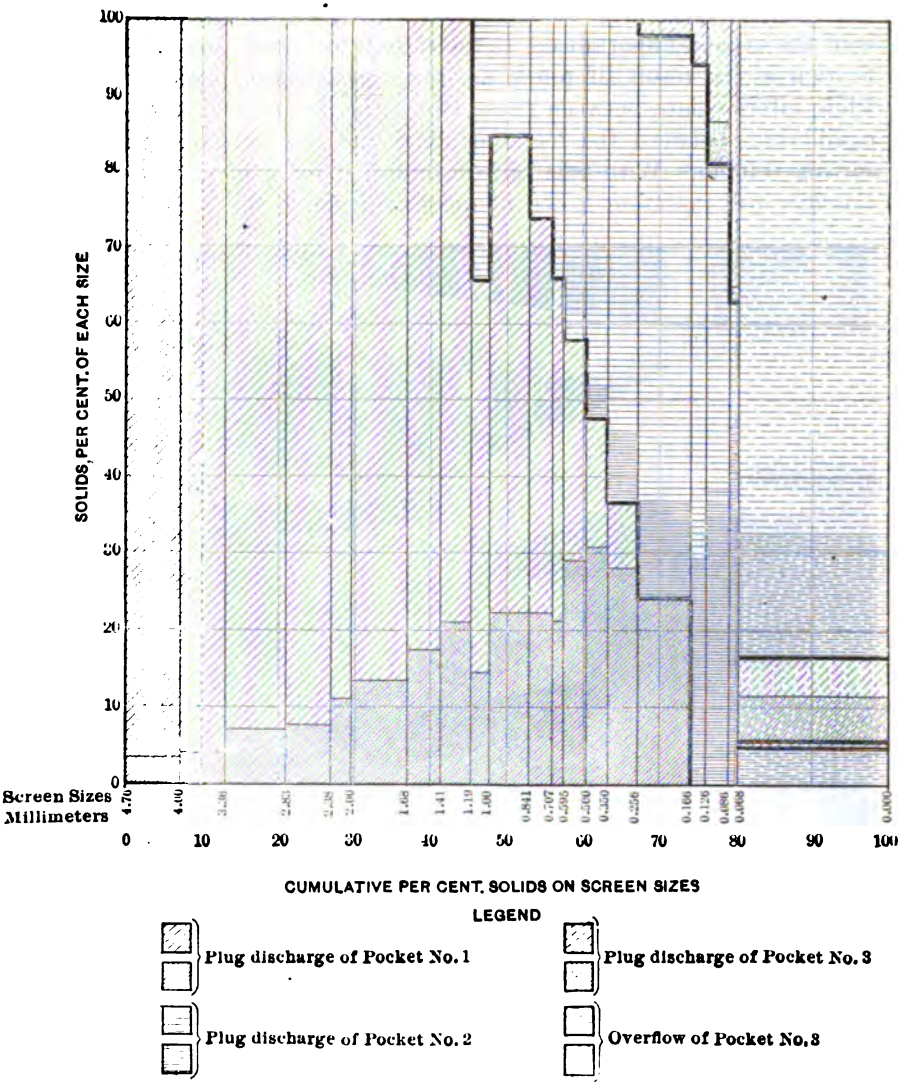
From the theory of hindered settling[1] it is seen that the ratio between

[1] Development of Hindered-settling Apparatus, by R. H. Richards, *Trans.*, xli., 396 (1910).

[2] Application of Hindered-settling to Hydraulic Classifiers, by E. S. Bardwell, this

TABLE I.—*Results of Test on Laboratory Three-pocket Classifier*

| Screen Sizes Millimeters | | Spigot No. 1 | | | | | Spigot No. 2 | | | | | Spigot No. 3 | | | | | Overflow | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Per Cent Solids in Spigot | Per Cent Solids in Total Feed | Free Mineral | | | Per Cent Solids in Spigot | Per Cent Solids in Total Feed | Free Mineral | | | Per Cent Solids in Spigot | Per Cent Solids in Total Feed | Free Mineral | | | Per Cent Solids in Total Feed | Free Mineral | |
| Through | On | | | Per Cent Solids on Screen Sizes | Per Cent Total in Spigot | Per Cent Total in Total Feed | | | Per Cent Solids on Screen Sizes | Per Cent Total in Spigot | Per Cent Total in Total Feed | | | Per Cent Solids on Screen Sizes | Per Cent Total in Spigot | Per Cent Total in Total Feed | | Per Cent Solids in Total Feed | Per Cent Total in Total Feed |
| | 4.00 | 11.7 | 7.2 | 3.7 | 2.3 | 1.3 | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... |
| 4.00 | 3.36 | 10.1 | 6.2 | 4.3 | 2.3 | 1.3 | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... |
| 3.36 | 2.83 | 13.5 | 8.2 | 7.3 | 5.1 | 2.8 | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... |
| 2.83 | 2.38 | 8.6 | 5.3 | 7.6 | 3.4 | 1.9 | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... |
| 2.38 | 2.00 | 4.8 | 2.9 | 11.2 | 2.8 | 1.6 | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... |
| 2.00 | 1.68 | 12.2 | 7.4 | 14.7 | 9.4 | 5.2 | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... |
| 1.68 | 1.41 | 7.4 | 4.5 | 17.7 | 6.8 | 3.8 | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... |
| 1.41 | 1.19 | 6.3 | 3.8 | 21.6 | 7.1 | 3.9 | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... |
| 1.19 | 1.00 | 2.6 | 1.6 | 22.9 | 3.1 | 1.7 | 4.3 | 0.8 | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... |
| 1.00 | 0.841 | 7.4 | 4.5 | 26.5 | 10.2 | 5.6 | 4.3 | 0.8 | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... |
| 0.841 | 0.707 | 3.7 | 2.3 | 30.5 | 6.0 | 3.3 | 4.0 | 0.8 | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... |
| 0.707 | 0.595 | 1.7 | 1.0 | 32.2 | 2.8 | 1.6 | 2.7 | 0.5 | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... |
| 0.595 | 0.500 | 2.8 | 1.7 | 51.0 | 7.4 | 4.0 | 6.4 | 1.2 | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... | .... |
| 0.500 | 50 Mesh | 2.1 | 1.3 | 65.0 | 7.4 | 4.1 | 7.1 | 1.4 | 7.3 | 2.2 | 0.5 | .... | .... | .... | .... | .... | .... | .... | .... |
| 50 Mesh | 60 " | 2.4 | 1.5 | 77.3 | 9.7 | 5.3 | 13.4 | 2.6 | 15.4 | 8.9 | 1.9 | .... | .... | .... | .... | .... | .... | .... | .... |
| 60 " | 90 " | | | | | | 26.4 | 5.1 | 16.3 | 18.5 | 3.9 | 3.1 | 0.1 | .... | .... | .... | .... | .... | .... |
| 90 " | 120 " | | | | | | 9.0 | 1.7 | 30.8 | 11.8 | 2.5 | 2.0 | 0.1 | .... | .... | .... | .... | .... | .... |
| 120 " | 150 " | 2.7 | 1.7 | 100.0 | 14.2 | 7.8 | 13.1 | 2.5 | 47.4 | 26.7 | 5.7 | 18.9 | 0.6 | 3.2 | 1.5 | 0.1 | .... | .... | .... |
| 150 " | 200 " | | | | | | 3.8 | 0.7 | 72.7 | 11.9 | 2.5 | 11.8 | 0.4 | 5.2 | 1.5 | 0.1 | .... | .... | .... |
| 200 " | | | | | | | 5.5 | 1.1 | 84.4 | 20.0 | 4.2 | 64.2 | 2.1 | 39.7 | 97.0 | 3.9 | 16.4 | 25.0 | 19.5 |
| Total | | 100.0 | 61.1 | .... | 100.0 | 55.2 | 100.0 | 19.2 | .... | 100.0 | 21.2 | 100.0 | 3.3 | .. | 100.0 | 4.1 | 16.4 | .. | 19.5 |




The entire chart area represents the total feed to the classifiers.

The area covered by each conventional sign represents the proportion of the original feed distributed to each pocket and to the overflow.

The closely hatched area represents the amount of free sulphide mineral.

FIG. 4.—TEST ON A SERIES OF THREE SINGLE-POCKET CLASSIFIERS (LABORATORY SIZE). CLASSIFICATION CHART BASED ON SCREEN SIZING AND SORTING TESTS.

the cross-sectional areas of the teeter chamber and the sorting column is of the utmost importance. If the ratio is too small, free-settling classification is the result; if too large, the classifier will bank or choke up; while intermediate between these two ratios is one which will establish equilibrium between the teetering mass and the discharge through the sorting column. With this ratio established for a given condition, the classifier will maintain a bed of quicksand above the constriction, with

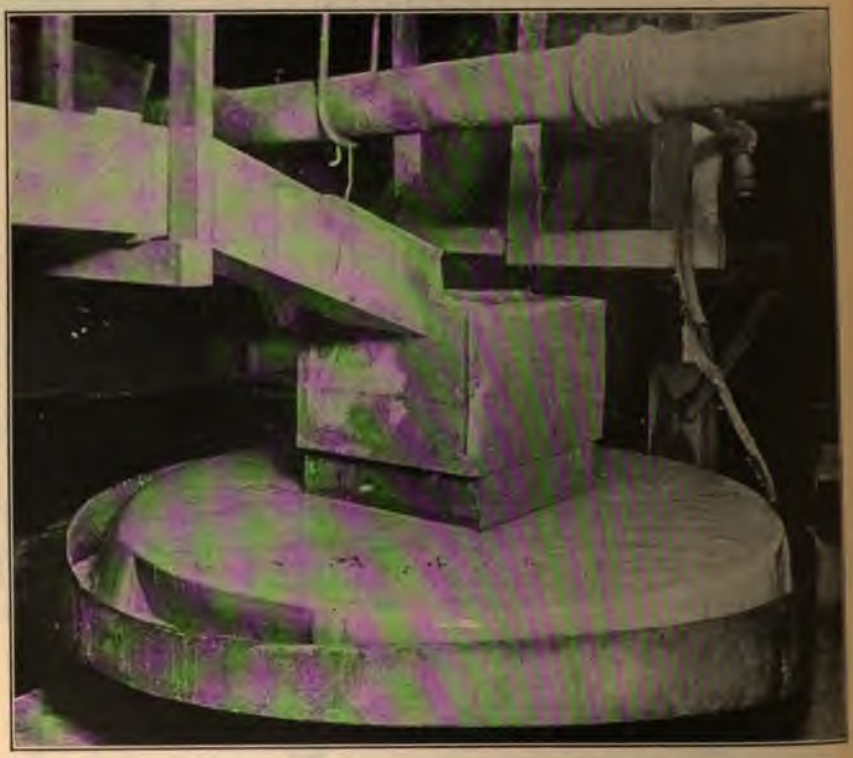


FIG. 6.—DESLIMER IN USE AT B. & M. CONCENTRATOR.

a constant discharge through the spigot. Numerous interesting experiments have been followed out along this line, with the result that the ratios given in the following pages have been found best. The single opening has been discarded for numerous smaller openings, with the idea of equalizing the rising current throughout any given cross-section of the teeter chamber.

The present form of the classifier is installed in six of the eight sections of the Great Falls Concentrator and the remodeled Section No. 1 of the Anaconda Concentrator, of which it may be said to form the nucleus.

## The Classifier in Actual Practice.

The classifiers may be divided into two groups, according to the work performed: namely, desliming classifiers and table-feed classifiers.

### *Desliming Classifiers.*

Fig. 5 gives a cross-sectional view of a standard 7-ft. deslimer, showing the essential point of construction.

Fig. 6 is a photographic view of a deslimer in use at the B. & M. Concentrator, Great Falls. These deslimers are used for desliming pulp from 4 mm. down and have proved very efficient. One particular feature which has been found to hold in practice is that when desliming fine pulp (2.5 mm. or under) no rising hydraulic water is used in the classifier; *i. e.*, the only fresh water used in the deslimer is that used to supply the spigot, which is usually about 50,000 to 90,000 gal. per 24 hr. However, in desliming coarser feed some rising water is generally used. The B. & M. practice is to use a constriction of one 6 in. diameter by 2 in. long pipe, giving a ratio of 3.9. The Anaconda practice is to use a number of smaller pipes with a ratio of 4.6. Which is the better cannot be said, for conditions of practice are considerably different at the two mills. The shortest and best way to show what the classifiers will do in practice is to quote actual figures, obtained under different conditions.

If fed under ideal conditions a 7-ft. deslimer will handle 250,000 gal. of pulp with 200 tons of solids, giving an overflow of not over 2 per cent. of total solids in the overflow on 200 mesh. The spigot will be perfectly clear and very dense.

#### *1. Desliming 4.0 mm. Primary Feed, Overflow from 0.07 to 0.0 mm.*

"Primary" feed contains all the original mine fines and no finishing-roll product. Of necessity it is a very rich feed and contains a large amount of free mineral.

Following are figures representative of the work done at Great Falls on the above class of feed. The sum of the plug and the overflow is taken as the feed, which in all cases checked the actual feed sample.

| | Gallons per 24 hr. | Pounds per 24 hr. | Per Cent. Total Feed | Density, Per Cent. Solids | Assay, Per Cent. Copper |
|---|---|---|---|---|---|
| Spigot | 110,416 | 320,972 | 80.4 | 28.3 | 2.90 |
| Overflow | 205,200 | 78,160 | 19.6 | 4.4 | 2.54 |
| Feed | 134,616 | 399,132 | 100.0 | 28.7 | 2.83 |
| Total, fresh water | 181,000 | ..... | ..... | ..... | ..... |
| Rising water | 83,424 | ..... | ..... | ..... | ..... |

Gallons of water consumed per ton of feed, 905.
Diameter of spigot discharge, 1.25 in. Spigot clear.
Rising velocity through constriction = 226 mm. per second.
Ratio of cross-sectional area of teeter chamber to sorting column = 3.9.
Constriction opening = one pipe 6 in. in diameter, 2 in. long.

*Screen Sizing Figures.*

| Screen Sizes, Millimeters | | Spigot | | Overflow | |
|---|---|---|---|---|---|
| | | Individual Per Cent. Solids | Cumulative Per Cent. Solids | Individual Per Cent. Solids | Cumulative Per Cent. Solids |
| *Thru* | *On* | | | | |
| .... | 4.00 | 1.9 | 1.9 | ..... | ..... |
| 4.00 | 2.38 | 22.0 | 23.9 | ..... | ..... |
| 2.38 | 1.68 | 16.9 | 40.8 | ..... | ..... |
| 1.68 | 1.41 | 6.9 | 47.7 | ..... | ..... |
| 1.41 | 0.841 | 18.9 | 66.6 | ..... | ..... |
| 0.841 | 0.500 | 12.6 | 79.2 | ..... | ..... |
| 0.500 | 0.350 | 4.5 | 83.7 | ..... | ..... |
| 0.350 | 0.166 | 9.7 | 93.4 | ..... | ..... |
| 0.166 | 0.07 | 5.6 | 99.0 | 7.9 | 7.9 |
| 0.07 | ..... | 1.0 | 100.0 | 92.1 | 100.0 |
| | Total | 100.0 | 100.0 | 100.0 | 100.0 |

The overflow is shown as containing 7.9 per cent. of total overflow on 200 mesh, which is no doubt due to excessive use of hydraulic water. Under test condition this same deslimer has shown 2.3 per cent. on 200 mesh, which is normal. However, as these and all figures following, except where otherwise stated, are actual practice figures and not test figures, the writer will explain from his personal observations and experience with the classifier such abnormal figures as might be misleading to the reader unfamiliar with the classifier.

The same deslimer when treating a little coarser feed gave as follows:

| | Gallons per 24 Hours | Pounds per 24 Hours | Per Cent. Total Feed | Density, Per Cent. Solids |
|---|---|---|---|---|
| Spigot | 111,104 | 222,221 | 91.4 | 20.7 |
| Overflow | 196,340 | 21,600 | 8.6 | 1.3 |
| Feed | 205,229 | 243,821 | 100.0 | 15.5 |
| Total, fresh water | 102,215 | ..... | ..... | ..... |
| Rising water | None | ..... | ..... | ..... |

Gallons water consumed per ton of feed, 840.
Diameter of spigot, 1.25 in. Spigot, clear.

The overflow screen sized 6.6 per cent. of total solids on 200 mesh. The water consumption is low, but could have been made considerably lower by running a denser spigot.

*2. Desliming 1.25 mm. Secondary Feed, Overflow from 0.07 to 0.0 mm.*

In this case the "secondary" feed consisted of the undersize of 1.25 by 12 mm. punched-plate trommels screening finishing-roll product. The feed was very dense and afforded excellent feed for a deslimer.

| | Gallons per 24 Hours | Pounds per 24 Hours | Per Cent. Total Feed | Density, Per Cent. Solids |
|---|---|---|---|---|
| Spigot | 52,010 | 272,160 | 96.0 | 44.2 |
| Overflow | 85,790 | 11,230 | 4.0 | 1.6 |
| Feed | 79,800 | 283,390 | 100.0 | 33.1 |
| Total, fresh water | 58,000 | ..... | ..... | ..... |
| Rising water | 16,878 | ..... | ..... | ..... |

Gallons water consumed per ton of feed, 409.
Rising velocity through constriction, 84 mm. per second.
Diameter spigot 0.75 in. Spigot contains trace of slime.

*Screen Sizing Figures.*

| Screen Sizes, Millimeters | Feed | | Spigot | | Overflow | |
|---|---|---|---|---|---|---|
| | Individual Per Cent. Solids | Cumulative Per Cent. Solids | Individual Per Cent. Solids | Cumulative Per Cent. Solids | Individual Per Cent. Solids | Cumulative Per Cent. Solids |
| On 1.41 | 8.4 | 8.4 | 8.4 | 8.4 | ..... | ..... |
| On 0.841 | 38.1 | 46.5 | 41.1 | 49.5 | ..... | ..... |
| On 0.500 | 29.0 | 75.5 | 29.1 | 78.6 | ..... | ..... |
| On 0.350 | 5.4 | 80.9 | 6.1 | 84.7 | ..... | ..... |
| On 0.166 | 9.5 | 90.4 | 9.0 | 93.7 | ..... | ..... |
| On 0.07 | 4.5 | 94.9 | 4.3 | 98.0 | 0.2 | 0.2 |
| Through 0.07 | 5.1 | 100.0 | 2.0 | 100.0 | 99.8 | 100.0 |
| Total | 100.0 | 100.0 | 100.0 | 100.0 | 100.0 | 100.0 |

The above results were obtained from one of the secondary deslimers in use at Anaconda running under normal conditions. The constriction opening in this case was 11 1.5-in. nipples 3 in. long giving a ratio between the area of the teeter chamber and the sorting column of 4.6.

*3. Desliming Huntington Mill Discharge, Overflow from 0.07 to 0.0 mm.*

The feed was the discharge of one or more 5-ft. Huntington mills clothed with 1 by 12 mm. punched-plate screens. The practice was to deslime the above feed and treat the spigot on finishing Wilfleys. The writer has observed tables fed in this manner handle close to 50 tons per 24 hr.,

and make 80 per cent. of the feed into 0.45 per cent. Cu. tailings. One of the reasons for this splendid work is the distinct valley produced on the table as the hindered-settling effect of the classifier, resulting in the clean separation of free mineral and gangue grains. Oftentimes this valley was from 10 to 12 in. wide and extended the entire length of the diagonal division line between the mineral and the gangue. Very little middlings was made on these tables. I shall give two sets of figures under this type, showing a medium volume of feed and an excessive volume of feed.

| | Gallons per 24 Hr. | Pounds per 24 Hr. | Per Cent. Total Feed | Density Per Cent. Solids | Assay Per Cent. Cumulative |
|---|---|---|---|---|---|
| Spigot | 118,300 | 307,108 | 75.0 | 25.8 | 1.01 |
| Overflow | 539,000 | 102,200 | 25.0 | 2.3 | 1.37 |
| Feed | 536,100 | 409,308 | 100.0 | 8.6 | 1.10 |
| Total fresh water | 121,200 | ..... | ..... | ..... | ..... |
| Rising water | 15,184 | ..... | ..... | ..... | ..... |

Gallons water consumed per ton of feed, 590.
Diameter of spigot, 1⅛ in. Spigot, clear.
Constriction opening, one 6-in. pipe 2 in. long.
Constriction ratio, 3.9.
Rising velocity through constriction = 64 mm. per second.

*Screen Sizing Figures.*

| Screen Sizes Millimeters | Feed | | Spigot | | Overflow | |
|---|---|---|---|---|---|---|
| | Individual Per Cent. Solids | Cumulative Per Cent. Solids | Individual Per Cent. Solids | Cumulative Per Cent. Solids | Individual Per Cent. Solids | Cumulative Per Cent. Solids |
| On 1.41 | 3.4 | 3.4 | 5.4 | 5.4 | ..... | ..... |
| On 1.00 | 9.0 | 12.4 | 11.7 | 17.1 | ..... | ..... |
| On 0.841 | 13.5 | 25.9 | 16.0 | 33.1 | ..... | ..... |
| On 0.707 | 10.9 | 36.8 | 13.9 | 47.0 | ..... | ..... |
| On 0.500 | 11.7 | 48.5 | 13.2 | 60.2 | ..... | ..... |
| On 0.350 | 9.9 | 58.4 | 14.7 | 74.9 | ..... | ..... |
| On 0.166 | 12.4 | 70.8 | 15.5 | 90.4 | ..... | ..... |
| On 0.07 | 8.7 | 79.5 | 8.6 | 99.0 | 9.4 | 9.4 |
| Through 0.07 | 20.5 | 100.0 | 1.0 | 100.0 | 90.6 | 100.0 |
| Total..... | 100.0 | 100.0 | 100.0 | 100.0 | 100.0 | 100.0 |

The excessive fine material in the feed, 20.5 per cent. through 200 mesh, is due to the use of roll-product slime as sluicing water in the mill aprons.

Following are figures showing the work of the same deslimer under a much smaller volume of feed pulp:

| | Gallons per 24 Hr | Pounds per 24 Hr. | Per Cent. Total Feed | Density Per Cent. Solids |
|---|---|---|---|---|
| Spigot | 106,596 | 177,120 | 80.1 | 15.0 |
| Overflow | 186,120 | 44,150 | 19.9 | 2.7 |
| Feed | 182,716 | 221,270 | 100.0 | 11.6 |
| Total fresh water | 110,000 | ..... | ..... | ..... |
| Rising water | 10,489 | ..... | ..... | ..... |

Gallons fresh water consumed per ton of feed, 1,000.
Same size spigot and constriction as above.
Rising velocity through constriction = 41 mm. per second.

*Screen Sizing Figures.*

| Screen Sizes Millimeters | Feed | | Spigot | | Overflow | |
|---|---|---|---|---|---|---|
| | Individual Per Cent. Solids | Cumulative Per Cent. Solids | Individual Per Cent. Solids | Cumulative Per Cent. Solids | Individual Per Cent. Solids | Cumulative Per Cent. Solids |
| On 1.41 | 5.9 | 5.9 | 5.1 | 5.1 | ..... | ..... |
| On 1.00 | 12.1 | 18.0 | 12.0 | 17.1 | ..... | ..... |
| On 0.841 | 15.9 | 33.9 | 18.5 | 35.6 | ..... | ..... |
| On 0.707 | 9.8 | 43.7 | 13.0 | 48.6 | ..... | ..... |
| On 0.500 | 9.7 | 53.4 | 13.5 | 62.1 | ..... | ..... |
| On 0.350 | 6.2 | 59.6 | 7.9 | 70.0 | ..... | ..... |
| On 0.166 | 10.9 | 70.5 | 14.7 | 84.7 | ..... | ..... |
| On 0.07 | 6.7 | 77.2 | 10.9 | 95.6 | 1.2 | 1.2 |
| Through 0.07 | 22.8 | 100.0 | 4.4 | 100.0 | 98.8 | 100.0 |
| Total..... | 100.0 | 100.0 | 100.0 | 100.0 | 100.0 | 100.0 |

In desliming classifiers the volume of the feed pulp is the most important factor. Experiments have shown that with a deslimer with an 84-in. diameter top the pulp volume should never exceed 250,000 gal. to make an overflow 100 per cent. through 200 mesh. The reason for this is apparent.

Effective cross-section near overflow level = 5,290 sq. in.
Overflow = feed volume plus rising water.
Rising water = none.
Overflow = 250,000 gal. per 24 hr.

$$\frac{250{,}000 \times 231 \times 25.4}{86{,}400 \times 5{,}290} = 3.2 \text{ mm. per second rising current near overflow level.}$$

Richards[1] gives the free-settling velocity of quartz whose average diameter is 0.0747 mm. as 3.57 mm. per second. The feed must be constant, however, if we expect to use no rising water in the classifier.

FIG. 8.—VIEW OF TABLE-FEED CLASSIFIER.

From the foregoing, the following conditions are most favorable for desliming 2.5 mm. pulp:

1. Maximum feed volume, 250,000 gal.
2. Maximum feed density, 17 per cent. solids.
3. Maximum tonnage, 250; average tonnage, 200.
4. Rising velocity at overflow level, 3.2 mm.
5. Rising velocity through constriction varies with the size of material, from close to zero to 40 mm.

For material coarser than 2.5 mm. the density of pulp through the sorting column (constriction) must be lower, to prevent the possibility of choke-ups. As a consequence a rising current must be maintained in

[1] *Ore Dressing*, vol. iii, p. 1422 (1909).

the sorting column or slime will enter the spigot. The finer pulp (2.5 mm. or below) seems to act as a filter for the slime feed and oftentimes results have been reported showing the extraction of deslimed water from the

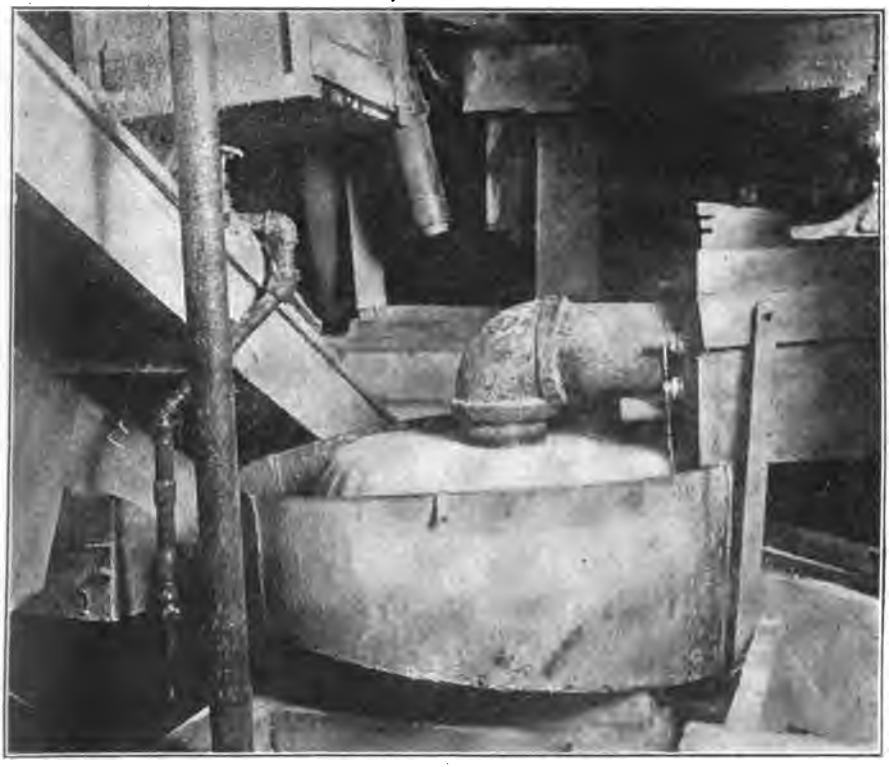


FIG. 9.—OVERFLOW OF TABLE-FEED CLASSIFIER.

feed and the addition of the same to the spigot. This can only be accounted for by the filtering effect of the teeter chamber. With feed above 2.5 mm. this phenomenon has seldom been observed.

## TABLE-FEED CLASSIFIERS.

The second division of these classifiers are called table-feed classifiers because their function is to overflow a product which may be best concentrated on some type of shaking table. The chief use to which this classifier is put is to overflow a 0.75 mm. table feed, either primary, or secondary, or mixed.

Fig. 7 is a cross-section of one of the table-feed classifiers, of the type installed at the Washoe Concentrator. Unfortunately, no figures could be obtained showing the work of these classifiers in practice. At the

B. & M. Concentrator an inner cone has been added to the original design and has found universal use in this type of classifier. The development of the feed cone was begun in May, 1912, the first classifier with an inner

FIG. 10.—CLASSIFIER WITH INNER FEED CONE.

cone being installed, I believe, on May 9, 1912. Although we have no figures showing the classification of the Anaconda classifiers at the Washoe Mill, Fig. 8 gives a very clear conception of the actual work done by them. Dr. Richards states in his paper, The Development of Hindered-Settling Apparatus, that the advantage of hindered-settled feed on a shaking table is the production of a valley between the mineral and the sand, thereby aiding the separation of the different products.

FIG. 11.—SPIGOT OF TABLE-FEED CLASSIFIER.

In Fig. 8 we see this valley as obtained in actual practice. The photograph was taken after shutting off the feed and dressing water simultaneously and then shutting down the table approximately 10 sec. after the feed and water were cut off. The table was handling from 15 to 18 tons of feed per 24 hr. The tailings at the deepest point were 0.75 in. deep, the concentrates 0.25 in. deep, and the middlings about $\frac{1}{16}$ to $\frac{1}{8}$ in. deep. The tailings will probably assay 0.30 per cent. of copper. The feed to the table ranged from 0.75 to 0.07 mm. and was overflowed from the classifier shown in Fig. 8. The apparent broken spot on the Wilfley is due to reflections.

*4. Classifying 2.5 mm. Deslimed Secondary Feed, Overflow from 1.00 to 0.07 mm.*

The feed was deslimed rolled product through 2.5-mm. round-hole trommel. The overflow was heavy and classification not very close. The classifier was one of the type shown in Fig. 9, no inner cone being used.

| | Gallons per 24 Hr. | Pounds per 24 Hr. | Per Cent. Total Feed | Density Per Cent. Solids |
|---|---|---|---|---|
| Spigot | 129,254 | 203,472 | 37.7 | 16.7 |
| Overflow | 408,205 | 335,750 | 62.3 | 9.2 |
| Feed | 223,434 | 539,222 | 100.0 | 24.2 |
| Total fresh water | 314,025 | ..... | ..... | ..... |
| Rising water | 192,906 | ..... | ..... | ..... |

Gallons water consumed per ton of feed, 1,165.

*Screen Sizing Figures.*

| Screen Size Millimeters | Spigot | | Overflow | |
|---|---|---|---|---|
| | Individual Per Cent. Solids | Cumulative Per Cent. Solids | Individual Per Cent. Solids | Cumulative Per Cent. Solids |
| On 2.38 | 2.6 | 2.6 | ..... | ..... |
| On 1.41 | 34.3 | 36.9 | 3.4 | 3.4 |
| On 1.00 | 27.7 | 64.6 | 8.4 | 11.8 |
| On 0.841 | 21.2 | 85.8 | 17.4 | 29.2 |
| On 0.500 | 12.0 | 97.8 | 33.5 | 62.7 |
| On 0.350 | 2.2 | 100.0 | 10.0 | 72.7 |
| On 0.166 | ..... | ..... | 18.9 | 91.6 |
| On 0.07 | ..... | ..... | 6.1 | 97.7 |
| Through 0.07 | ..... | ..... | 2.3 | 100.0 |
| Total | 100.0 | 100.0 | 100.0 | 100.0 |

Fig. 9 shows the overflow of the above classifier.

### *5. Classifying 2.5 mm. Feed, Overflow 0.75 mm.*

The classifier was equipped with an inner feed cone (Fig. 10). The feed consisted of 2.5-mm. round-hole trommel undersize, deslimed.

| | Gallons per 24 Hr. | Pounds per 24 Hr. | Per Cent. Total Feed | Density Per Cent. Solids | Assay Per Cent. Cumulative |
|---|---|---|---|---|---|
| Spigot | 116,000 | 314,064 | 62.8 | 21.5 | 2.56 |
| Overflow | 338,408 | 186,278 | 37.2 | 5.9 | 2.88 |
| Feed | 175,300 | 500,342 | 100.0 | 21.8 | 2.69 |
| Total fresh water | 279,108 | ..... | ..... | ..... | ..... |
| Rising water | 175,671 | ..... | ..... | ..... | ..... |

Gallons water consumed per ton of feed, 1,100.
Diameter of spigot, 1.25 in.
Constriction opening, 12 2-in. pipes 2 in. long, or 40.0 sq. in.
Ratio of area of teeter chamber to sorting column, 2.81.
Rising velocity through constriction = 316 mm. per second.
Rising velocity at bottom of annular space = 144 mm. per second.
Rising velocity at top of annular space = 122 mm. per second.

*Screen Sizing Figures.*

| Screen Sizes, Millimeters | Feed | | Spigot | | Overflow | |
|---|---|---|---|---|---|---|
| | Individual Per Cent. Solids | Cumulative Per Cent. Solids | Individual Per Cent. Solids | Cumulative Per Cent. Solids | Individual Per Cent. Solids | Cumulative Per Cent. Solids |
| On 2.00 | 7.4 | 7.4 | 12.3 | 12.3 | ..... | ..... |
| On 1.68 | 12.0 | 19.4 | 18.0 | 30.3 | ..... | ..... |
| On 1.41 | 11.1 | 30.5 | 13.8 | 44.1 | ..... | ..... |
| On 1.00 | 13.4 | 43.9 | 21.0 | 65.1 | ..... | ..... |
| On 0.841 | 11.7 | 55.6 | 14.8 | 79.9 | 6.7 | 6.7 |
| On 0.707 | 7.1 | 62.7 | 9.2 | 89.1 | 6.1 | 12.8 |
| On 0.500 | 7.6 | 70.3 | 6.1 | 95.2 | 11.0 | 23.8 |
| On 0.350 | 7.5 | 77.8 | 2.9 | 98.1 | 16.0 | 39.8 |
| On 0.166 | 10.5 | 89.3 | 1.9 | 100.0 | 27.8 | 67.6 |
| On 0.07 | 8.9 | 98.2 | ..... | ..... | 26.1 | 93.7 |
| Through 0.07 | 2.8 | 100.0 | ..... | ..... | 6.3 | 100.0 |
| Total | 100.0 | 100.0 | 100.0 | 100.0 | 100.0 | 100.0 |

The figures show this classification to be excellent, although the classifier was designed to receive a feed of 250,000 gal. per 24 hr., whereas it is receiving only 175,000 gal. In the spigot fully 80 per cent. of the fine material, through 0.707 mm., is free mineral grains, which we would expect in any spigot, even the perfectly classified.

A second set of figures shows the work performed on a feed containing more fines.

| | Gallons per 24 Hr. | Pounds per 24 Hr. | Per Cent. Total Feed | Density. Per Cent. Solids | Assay. Per Cent. Cumulative |
|---|---|---|---|---|---|
| Spigot | 115,300 | 283,392 | 47.8 | 19.3 | 3.15 |
| Overflow | 356,174 | 310,348 | 52.2 | 8.9 | 4.58 |
| Feed | 158,300 | 593,740 | 100.0 | 25.7 | 3.89 |
| Total fresh water | 313,174 | ..... | ..... | ..... | ..... |
| Rising water | 209,210 | ..... | ..... | ..... | ..... |

Gallons water per ton of feed, 1,050.
Spigot, 1.25 in. in diameter.
Constriction area, 40 sq. in.
Constriction ratio, 2.81.
Rising velocity through constriction = 373 mm. per second.
Rising velocity at bottom of annular space = 152 mm. per second.
Rising velocity at top of annular space = 129 mm. per second.

*Screen Sizing Figures.*

| Screen Sizes, Millimeters | Feed | | Spigot | | Overflow | |
|---|---|---|---|---|---|---|
| | Individual Per Cent. Solids | Cumulative Per Cent. Solids | Individual Per Cent. Solids | Cumulative Per Cent. Solids | Individual Per Cent. Solids | Cumulative Per Cent. Solids |
| On 2.00 | 7.2 | 7.2 | 13.9 | 13.9 | ..... | ..... |
| On 1.68 | 11.8 | 19.0 | 23.6 | 37.5 | ..... | ..... |
| On 1.41 | 7.2 | 26.2 | 16.5 | 54.0 | ..... | ..... |
| On 1.00 | 12.9 | 39.1 | 20.3 | 74.3 | ..... | ..... |
| On 0.841 | 9.6 | 48.7 | 13.6 | 87.9 | 7.6 | 7.6 |
| On 0.707 | 8.8 | 57.5 | 6.9 | 94.8 | 5.6 | 13.2 |
| On 0.500 | 8.3 | 65.8 | 3.8 | 98.6 | 8.7 | 21.9 |
| On 0.350 | 10.6 | 76.4 | 1.4 | 100.0 | 16.6 | 38.5 |
| On 0.166 | 12.0 | 88.4 | ..... | ..... | 29.9 | 68.4 |
| On 0.07 | 9.0 | 97.4 | ..... | ..... | 26.7 | 95.1 |
| Through 0.07 | 2.6 | 100.0 | ..... | ..... | 4.9 | 100.0 |
| Total..... | 100.0 | 100.0 | 100.0 | 100.0 | 100.0 | 100.0 |

The work done by this classifier, as shown by the two sets of figures, may be said to be the best obtained with any type of the Anaconda classifier. The separation at 0.75 mm. is remarkable and shows the possibilities of the machine. This particular unit is used at the B. & M. Concentrator to furnish Hancock jig feed and finishing-table feed. Figs. 10 and 11 show the overflow and spigot of this installation as in use to-day. Note the appearance of the overflow in the photograph. It is very even except for the agitation produced by the inner cone throwing backwards into the overflow, material which is carried up the outside of the cone.

### *6. Classifying 4.0 mm. Original Feed, Overflow 0.75 mm.*

The feed was deslimed original material, being the undersize of 5.0 mm. round-hole trommels. Classifier equipped with feed cone.

| | Gallons per 24 Hr. | Pounds per 24 Hr. | Per Cent. Total Feed | Density, Per Cent. Solids | Assay, Per Cent. Cumulative |
|---|---|---|---|---|---|
| Spigot | 97,292 | 420,076 | 65.0 | 27.9 | 3.24 |
| Overflow | 253,806 | 224,994 | 35.0 | 9.1 | 3.42 |
| Feed | 207,708 | 645,070 | 100.0 | 22.9 | 3.30 |
| Gallons fresh water | 143,390 | ..... | ..... | ..... | ..... |
| Rising water | 62,898 | ..... | ..... | ..... | ..... |

Gallons water per ton of feed, 445.
Spigot diameter, 1.25 in.
Constriction area, 40 sq. in.
Constriction ratio, 2.81.
Rising velocity through constriction = 136 mm. per second.
Rising velocity at bottom annular space = 109 mm. per second.
Rising velocity at top annular space = 92 mm. per second.

*Screen Sizing Figures.*

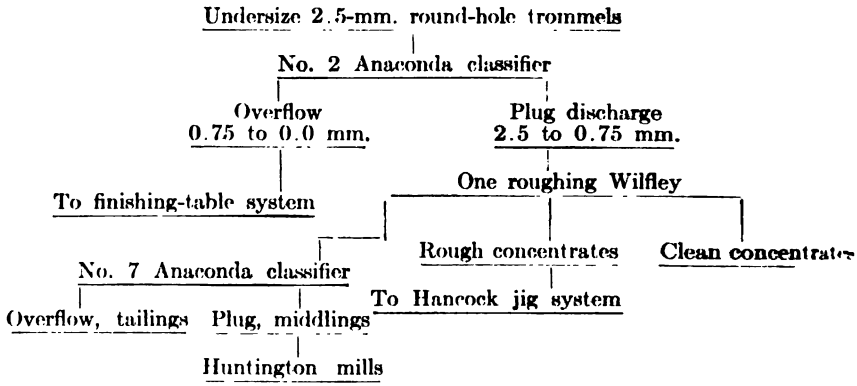
| Screen Sizes, Millimeters | Feed | | Spigot | | Overflow | |
|---|---|---|---|---|---|---|
| | Individual Per Cent. Solids | Cumulative Per Cent. Solids | Individual Per Cent. Solids | Cumulative Per Cent. Solids | Individual Per Cent. Solids | Cumulative Per Cent. Solids |
| On 4.00 | 1.9 | 1.9 | 2.7 | 2.7 | ..... | ..... |
| On 2.38 | 22.0 | 23.9 | 28.0 | 30.7 | ..... | ..... |
| On 1.41 | 23.8 | 47.7 | 30.2 | 60.9 | ..... | ..... |
| On 0.841 | 18.9 | 66.6 | 22.8 | 83.7 | 7.3 | 7.3 |
| On 0.500 | 12.6 | 79.2 | 7.8 | 91.5 | 16.4 | 23.7 |
| On 0.350 | 4.5 | 83.7 | 4.2 | 95.7 | 8.9 | 32.6 |
| On 0.166 | 9.7 | 93.4 | 3.7 | 99.4 | 33.0 | 65.6 |
| On 0.07 | 5.6 | 99.0 | 0.6 | 100.0 | 25.8 | 91.4 |
| Through 0.07 | 1.0 | 100.0 | ..... | ..... | 8.6 | 100.0 |
| Total..... | 100.0 | 100.0 | 100.0 | 100.0 | 100.0 | 100.0 |

### *7. Classifying Middlings-Tailings Roughing-Table Product, Overflow Coarse Tailings.*

At the present time there are no classifiers in the B. & M. Concentrator overflowing tailings. This is due chiefly to the lack of head room, as the mill is old and not designed along modern lines.

The B. & M. system of concentration as installed at the Washoe mill includes four of these classifiers in a 1,500-ton unit, although two of

them only are used most of the time. Fig. 12 is a view of one of the tailing-classifiers in use at the Washoe mill. The classifier in this case is discharged through the lower spigot direct to Huntington mills. The feed is the middlings-tailings product from the roughing tables and the overflow averages 0.45 per cent. of copper. Two spigots are provided for the classifier, the original idea being to use one in case the other choked. However, it has been found unnecessary to provide the second spigot, the later types being cast with but one (see Fig. 7). The following data are not taken from regular practice but are taken from a several days' test run and suffice to show what the classifier will do under this class of feed. The so-called middlings-tailings consist of true middlings grains and coarse tailings grains with no free mineral. The test was carried out according to the following flow sheet:




The numbers applied to the classifiers are arbitrary and are used merely for convenience. Fig. 15 shows the cross-section of the No. 2 classifier. Fig. 13 shows the No. 7 classifier, and Fig. 14 shows the distribution of products on the roughing Wilfley.

| Product | Gallons per 24 Hr. | Pounds per 24 Hr. | Per Cent. Total Feed | Density, Per Cent. Solids | Assay, Per Cent. Cu | Per Ct. Total Copper in Feed |
|---|---|---|---|---|---|---|
| Feed | ..... | 132,546 | 100.0 | ..... | 1.39 | 100.0 |
| Clean concentrates | 1,048 | 16,330 | 12.3 | 83.3 | 5.5 | 48.5 |
| Rough concentrates | 21,600 | 41,696 | 31.5 | 16.7 | 1.24 | 28.0 |
| Plug middlings | ..... | 43,594 | 32.9 | ..... | 0.69 | 16.3 |
| Overflow tailings | ..... | 30,926 | 23.3 | ..... | 0.43 | 7.2 |
| Total | ..... | 132,546 | 100.0 | ..... | 1.39 | 100.0 |

The No. 7 classifier used 127,000 gal. of fresh water, or 350 gal. per ton of feed. As seen from the sketch, Fig. 13, the classifier was a very small

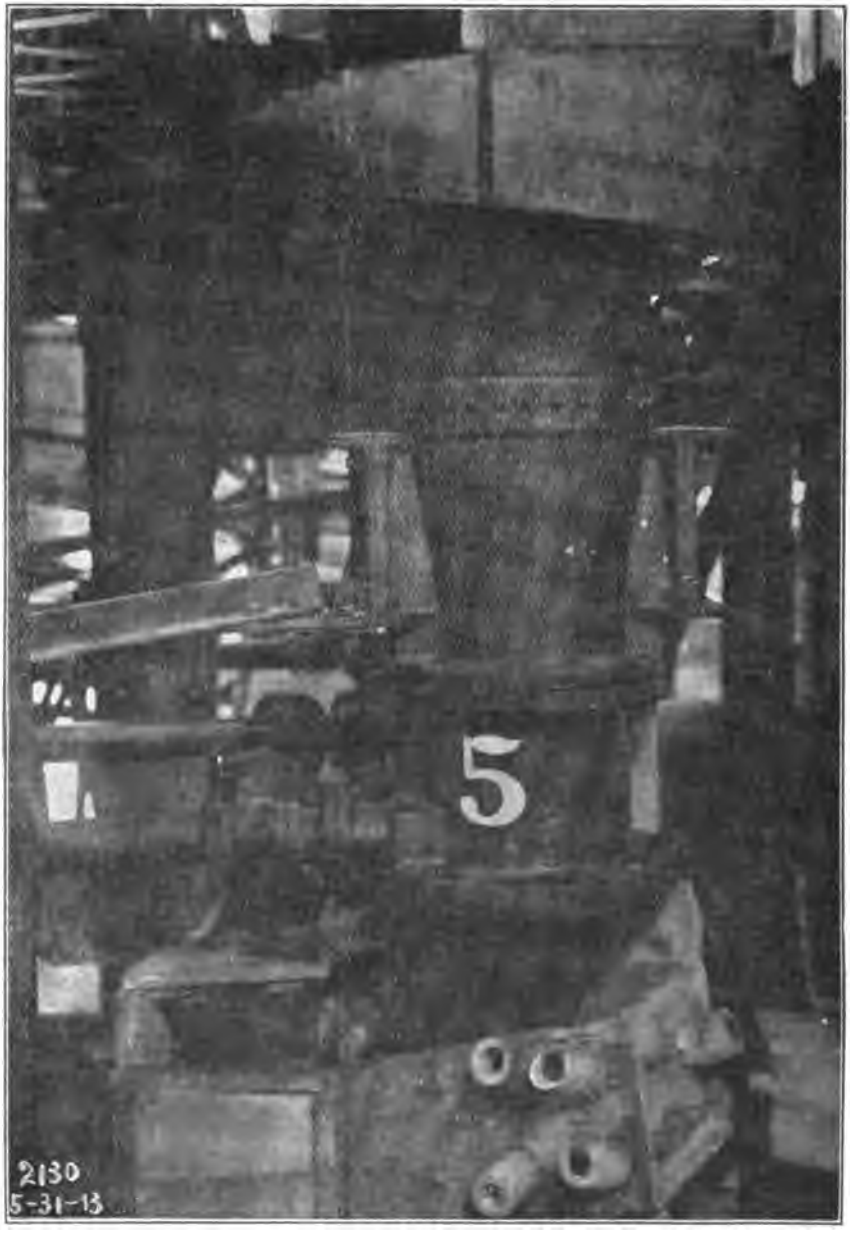



FIG. 12.—TAILINGS CLASSIFIER AT THE WASHOE MILL.

one, handling only 36 tons of feed. Still, there is every possibility of building a larger machine with just as great an efficiency. The most essential feature of this system is that the roughing Wilfley must produce a middlings-tailings product absolutely free of fine mineral. The writer has seen roughing Wilfleys at the B. & M. Concentrator doing very good work on 130 tons of 2.5 to 7.5 mm. feed.

*Screen Sizing Figures.*

| Screen Size Millimeters | Plug Middlings | | Overflow Tailings | |
|---|---|---|---|---|
| | Individual Per Cent. Total Solids | Cumulative Per Cent. Total Solids | Individual Per Cent. Total Solids | Cumulative Per Cent. Total Solids |
| On 1.41 | 54.1 | 54.1 | 30.8 | 30.8 |
| On 0.841 | 43.2 | 97.3 | 62.3 | 93.1 |
| Through 0.841 | 2.7 | 100.0 | 6.9 | 100.0 |
| Total | 100.0 | 100.0 | 100.0 | 100.0 |

*Roughing Table Products—Clean and Rough Concentrates.*

| Screen Size Millimeters | Clean Concentrates | | Rough Concentrates | |
|---|---|---|---|---|
| | Individual Per Cent. Total Solids | Cumulative Per Cent. Total Solids | Individual Per Cent. Total Solids | Cumulative Per Cent. Total Solids |
| On 1.41 | 2.7 | 2.7 | 3.9 | 3.9 |
| On 0.841 | 4.3 | 7.0 | 30.1 | 34.0 |
| On 0.500 | 13.4 | 20.4 | 57.8 | 91.8 |
| On 0.350 | 15.2 | 35.6 | 5.3 | 97.1 |
| On 0.166 | 51.9 | 87.5 | 2.9 | 100.0 |
| On 0.07 | 11.5 | 99.0 | ..... | ..... |
| Through 0.07 | 1.0 | 100.0 | ..... | ..... |
| Total | 100.0 | 100.0 | 100.0 | 100.0 |

The screen sizing figures are self explanatory. The screen sizing of the rough concentrates shows the segregation of "through 0.841 on 0.500 mm." material, amounting to 57.8 per cent. of the total rough concentrates, or 18.2 per cent. of the total feed to the table. It may be well to state here that the No. 2 classifier preceding the roughing table, while doing good work, was one of the earliest fitted with the feed cone. One of the present-day cone classifiers would eliminate a great percentage of this fine ma-

terial. Some interesting tests have been made on this No. 2 classifier and the results follow. This particular No. 2 was the pioneer of the present inner feed cone classifier as used at Great Falls.

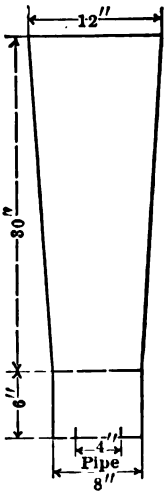



FIG. 13.—SECTION OF NO. 2 CLASSIFIER.

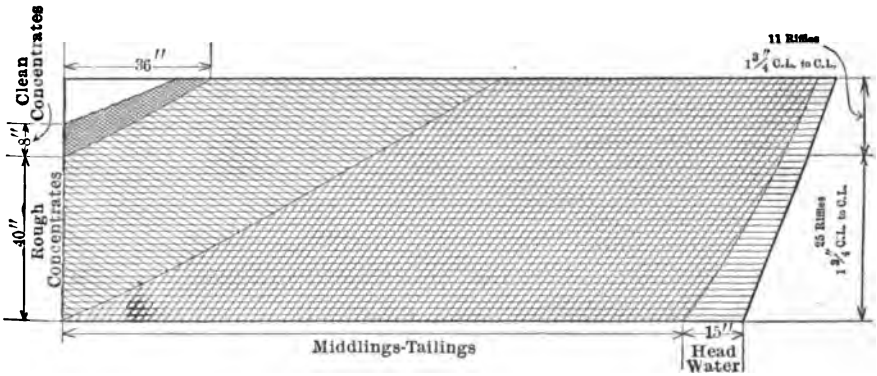



FIG. 14.—DISTRIBUTION OF PRODUCTS ON WILFLEY TABLE.

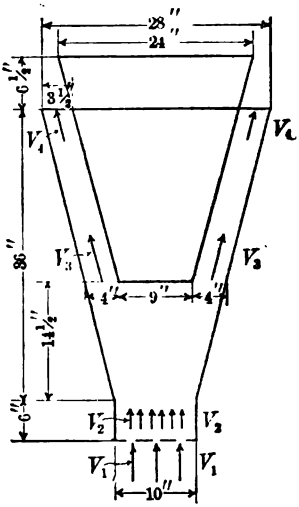
For the method of designing the Anaconda classifier reference is made to a paper before this meeting by E. S. Bardwell, The Application of Hindered Settling to Hydraulic Classifiers.[1]

*8. Classifying 2.5 mm. Roll Product (Not Deslimed) in No. 2 Classifier.*

The following test figures are actual practice figures. The classifier was not altered or adjusted in preparation for the test. Samples were

[1] This *Bulletin*.

taken on two days practically at 1-hr. intervals and each sample was screen sized and sorted into free mineral grains, true middlings, and tailings grains. The classifier was fed direct from the trommel undersize, taking all of the feed at this point of a 500-ton section. The primary object of the test was to show the work of the classifier under varying




| | |
|---|---|
| Area top of annular space | = 269.4 sq. in. |
| Area of bottom annular space | = 163.4 sq. in. |
| Area of teeter chamber | = 78.8 sq. in. |
| Area of construction opening | = 33.46 sq. in. |

FIG. 15.—SECTION OF NO. 7 CLASSIFIER.

feed. The feed varied during this run from 90 to 250 tons per 24 hr., although the pulp volume was fairly constant. The tabulations show remarkably good classification under this changing feed.

The classifier was provided with a 1.25-in. spigot and used an average of 124,000 gal. of total fresh water per 24 hr., this being determined as an average of many samples.

The following tabulations and classification charts tell the story at the time of each different sample more clearly than words.

The symbols used in the tabulation are as follows:

$V_1$ = Rising velocity through constriction in millimeters per second.
$V_2$ = Rising velocity in teeter chamber in millimeters per second.
$V_3$ = Rising velocity at bottom of annular ring in millimeters per second.
$V_4$ = Rising velocity near top of annular ring in millimeters per second.

A = Per cent. of variation of feed pulp, gallons per 24 hr. above and below average.
B = Per cent. of variation of feed pulp solids, pounds per 24 hr. above and below average.
C = Average diameter in millimeters of quartz in plug discharge.
D = Average diameter in millimeters of mineral in plug discharge.
E = Ratio of C and D.
F = Tons of plug discharge per square inch of constriction opening per 24 hr.
G = Tons of plug discharge per square inch of cross-section of teeter chamber per 24 hr.

*Details of No. 2 Classifier Tests.*

| Test Number | Time Taken | $V_1$ [a] | $V_2$ [b] | $V_3$ | $V_4$ | A | B | C | D | E | F | G |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | 5-14-12 | | | | | | | | | | | |
| 1 | 9:00 | 87 | 37 | 110 | 67 | +1.5 | +21.2 | 1.42 | 0.58 | 2.62 | 4.78 | 2.03 |
| 2 | 10:30 | 76 | 32 | 93 | 56 | —17.0 | —31.3 | 1.30 | 0.58 | 2.24 | 2.95 | 1.26 |
| 3 | 11:40 | 74 | 31 | 107 | 65 | —1.1 | —10.2 | 1.40 | 0.70 | 2.00 | 3.67 | 1.56 |
| 4 | 12:40 | 100 | 43 | 112 | 68 | +1.5 | +29.6 | 1.28 | 0.54 | 2.37 | 4.75 | 2.02 |
| 5 | 1:40 | 94 | 40 | 130 | 79 | +18.9 | +13.6 | 1.47 | 0.77 | 1.91 | 4.26 | 1.81 |
| 6 | 2:40 | 80 | 34 | 116 | 70 | +3.7 | —53.9 | 1.51 | 0.91 | 1.66 | 2.06 | 0.88 |
| 7 | 3:40 | 89 | 38 | 108 | 66 | —0.7 | +14.0 | 1.40 | 0.80 | 1.75 | 4.63 | 1.97 |
| | 5-15-12 | | | | | | | | | | | |
| 8 | 11:20 | 100 | 43 | 107 | 65 | —4.6 | +2.8 | 1.54 | 0.65 | 2.36 | 4.33 | 1.84 |
| 9 | 1:15 | 105 | 45 | 115 | 70 | +2.6 | +17.7 | 1.29 | 0.72 | 1.79 | 4.40 | 1.87 |
| 10 | 2:15 | 101 | 43 | 111 | 67 | —2.5 | —2.5 | 1.34 | 0.82 | 1.63 | 3.75 | 1.60 |
| 11 | 3:15 | 102 | 43 | 114 | 69 | —2.5 | —1.1 | 1.28 | 0.77 | 1.66 | 3.79 | 1.61 |
| Average... | ..... | 92 | 39 | 111 | 67 | 0.0 | 0.0 | 1.39 | 0.71 | 1.95 | 3.94 | 1.68 |

[a] Volume of solids in plug discharge allowed for.

[b] Assuming the volume of solids in the teeter chamber to be equivalent to the volume passing through the constriction for the same length of column. Actually $V_2$ probably is nearly equal to $V_1$, due to the fact that the density of the pulp in the teeter chamber is greater than the density of the pulp passing through the constriction opening.

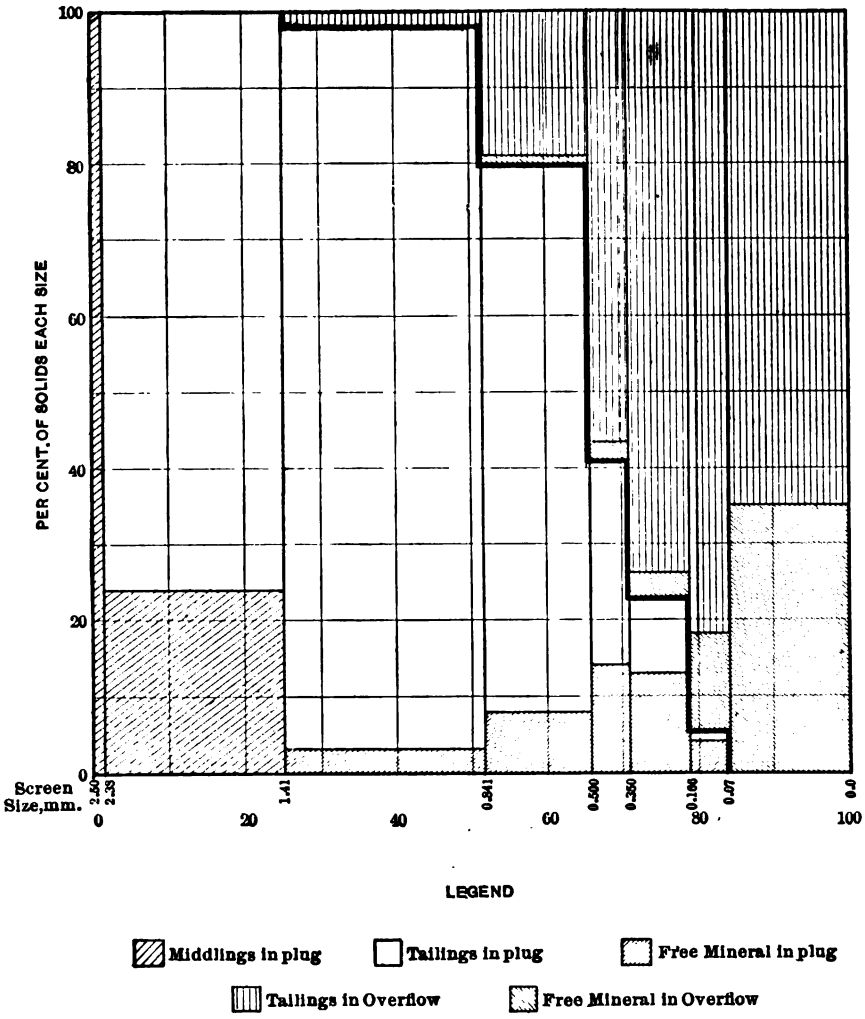
## *No. 2 Classifier*

| Screen Size Millimeters | Spigot: Per Cent. Total Solids in Spigot | Spigot: Per Cent. Total Solids in Feed | Spigot: Free Mineral: Per Cent. Total Solids on Screen Size | Spigot: Free Mineral: Per Cent. Total Free Mineral: Spigot | Spigot: Free Mineral: Per Cent. Total Free Mineral: Feed | Spigot: Middlings: Per Cent. Total Solids on Screen Size | Spigot: Middlings: Per Cent. Total Solids in Spigot | Spigot: Middlings: Per Cent. Total Solids in Feed | Spigot: Tailings: Per Cent. Total Solids on Screen Size | Spigot: Tailings: Per Cent. Total Solids in Spigot | Spigot: Tailings: Per Cent. Total Solids in Feed | Overflow: Per Cent. Total Solids in Overflow | Overflow: Per Cent. Total Solids in Feed | Overflow: Free Mineral: Per Cent. Total Solids on Screen Size | Overflow: Free Mineral: Per Cent. Total Mineral in Overflow | Overflow: Free Mineral: Per Cent. Total Mineral in Feed | Overflow: Tailings: Per Cent. Total Solids on Screen Size | Overflow: Tailings: Per Cent. Total Solids in Overflow | Overflow: Tailings: Per Cent. Total Solids in Feed |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Test No. 4** | | | | | | | | | | | | | | | | | | | |
| 2.38 | 3.6 | 2.2 | ..... | ..... | ..... | 100.0 | 3.6 | 2.2 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 1.41 | 32.2 | 20.0 | 6.5 | 19.5 | 8.0 | 13.2 | 4.3 | 2.7 | 80.3 | 25.7 | 16.0 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 0.841 | 39.4 | 24.6 | 6.6 | 24.2 | 10.0 | ..... | ..... | ..... | 93.4 | 36.6 | 22.8 | 0.8 | 0.3 | ..... | ..... | ..... | 100.0 | 0.8 | 0.3 |
| 0.500 | 17.6 | 11.0 | 13.3 | 21.8 | 9.1 | ..... | ..... | ..... | 86.7 | 15.2 | 9.5 | 6.9 | 2.6 | 5.9 | 1.6 | 1.0 | 94.1 | 6.5 | 2.4 |
| 0.350 | 2.4 | 1.5 | 31.6 | 6.9 | 2.9 | ..... | ..... | ..... | 68.4 | 1.6 | 1.0 | 6.5 | 2.5 | 7.8 | 2.0 | 1.2 | 92.2 | 6.0 | 2.3 |
| 0.166 | 3.9 | 2.4 | 44.5 | 16.1 | 6.7 | ..... | ..... | ..... | 55.5 | 2.0 | 1.3 | 18.5 | 7.0 | 8.8 | 6.5 | 3.8 | 91.2 | 16.8 | 6.4 |
| 0.07 | 0.6 | 0.4 | 50.0 | 2.9 | 1.2 | ..... | ..... | ..... | 50.0 | 0.3 | 0.2 | 16.4 | 6.2 | 17.9 | 11.8 | 6.9 | 82.1 | 13.5 | 5.1 |
| Th. 0.07 | 0.3 | 0.2 | 100.0 | 8.6 | 3.6 | ..... | ..... | ..... | ..... | ..... | ..... | 50.9 | 19.1 | 38.0 | 78.1 | 45.6 | 62.0 | 31.6 | 11.9 |
| Total | 100.0 | 62.3 | 10.7 | 100.0 | 41.5 | 7.9 | 7.9 | 4.9 | 81.4 | 81.4 | 50.8 | 100.0 | 37.7 | 24.8 | 100.0 | 58.5 | 75.2 | 75.2 | 28.4 |
| **Test No. 5** | | | | | | | | | | | | | | | | | | | |
| 2.38 | 4.1 | 2.6 | ..... | ..... | ..... | 100.0 | 4.1 | 2.6 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 1.41 | 43.3 | 27.6 | 2.5 | 11.6 | 4.5 | 97.5 | 42.2 | 26.9 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 0.841 | 34.8 | 22.2 | 6.6 | 24.3 | 9.3 | ..... | ..... | ..... | 93.4 | 32.5 | 20.7 | 3.0 | 1.1 | 9.1 | 1.0 | 0.6 | 90.9 | 2.9 | 1.0 |
| 0.500 | 12.6 | 8.1 | 20.3 | 27.2 | 10.4 | ..... | ..... | ..... | 79.7 | 10.0 | 6.4 | 10.9 | 3.9 | 7.6 | 3.1 | 1.9 | 92.4 | 10.0 | 3.6 |
| 0.350 | 2.3 | 1.5 | 57.7 | 14.3 | 5.5 | ..... | ..... | ..... | 42.3 | 1.0 | 0.6 | 7.1 | 2.6 | 7.8 | 2.1 | 1.3 | 92.2 | 6.5 | 2.4 |
| 0.166 | 2.2 | 1.4 | 76.5 | 18.2 | 7.0 | ..... | ..... | ..... | 23.5 | 0.5 | 0.3 | 20.1 | 7.3 | 13.2 | 10.0 | 6.2 | 86.8 | 17.4 | 6.3 |
| 0.07 | 0.7 | 0.4 | 57.1 | 4.4 | 1.7 | ..... | ..... | ..... | 42.9 | 0.3 | 0.2 | 17.2 | 6.2 | 23.9 | 15.5 | 9.5 | 76.1 | 13.1 | 4.8 |
| Th. 0.07 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | 41.7 | 15.1 | 43.5 | 68.3 | 42.1 | 56.5 | 23.5 | 8.5 |
| Total | 100.0 | 63.8 | 9.4 | 100.0 | 38.4 | 46.3 | 46.3 | 29.5 | 44.3 | 44.3 | 28.2 | 100.0 | 36.2 | 26.6 | 100.0 | 61.6 | 73.4 | 73.4 | 26.6 |
| **Test No. 6** | | | | | | | | | | | | | | | | | | | |
| 2.38 | 2.6 | 2.0 | ..... | ..... | ..... | 100.0 | 2.6 | 2.0 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 1.41 | 51.5 | 39.2 | 2.9 | 23.5 | 9.4 | 97.1 | 50.0 | 38.0 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 0.841 | 28.8 | 22.0 | 4.5 | 20.4 | 8.2 | 4.5 | 1.3 | 0.9 | 90.8 | 26.2 | 10.9 | 0.8 | 0.2 | ..... | ..... | ..... | 100.0 | 0.8 | 0.2 |
| 0.500 | 11.2 | 8.5 | 11.7 | 20.4 | 8.2 | ..... | ..... | ..... | 88.4 | 9.0 | 7.6 | 5.7 | 1.4 | 3.7 | 0.7 | 0.4 | 96.3 | 5.4 | 1.3 |
| 0.350 | 2.2 | 1.7 | 35.3 | 12.2 | 4.9 | ..... | ..... | ..... | 64.8 | 1.4 | 1.1 | 5.7 | 1.4 | 7.4 | 1.4 | 0.8 | 92.6 | 5.2 | 1.2 |
| 0.166 | 2.8 | 2.1 | 40.5 | 17.3 | 6.9 | ..... | ..... | ..... | 59.5 | 1.6 | 1.2 | 22.8 | 5.4 | 17.4 | 12.8 | 7.7 | 82.6 | 18.9 | 4.5 |
| 0.07 | 0.9 | 0.7 | 42.9 | 6.2 | 2.4 | ..... | ..... | ..... | 57.1 | 0.6 | 0.4 | 15.7 | 3.7 | 30.6 | 15.6 | 9.4 | 69.4 | 10.9 | 2.6 |
| Th. 0.07 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | 49.3 | 11.7 | 43.2 | 69.5 | 41.7 | 56.8 | 28.1 | 8.6 |
| Total | 100.0 | 76.2 | 6.4 | 100.0 | 40.0 | 53.0 | 53.0 | 40.0 | 39.7 | 39.7 | 30.2 | 100.0 | 23.8 | 30.7 | 100.0 | 60.0 | 69.3 | 69.3 | 16.4 |

| Screen Size Millimeters | Per Cent. Total Solids in Spigot | Per Cent. Total Solids in Feed | Spigot: Free Mineral: Per Cent. Total Solids on Screen Size | Spigot: Free Mineral: Per Cent. Total Free Mineral: Spigot | Spigot: Free Mineral: Per Cent. Total Free Mineral: Feed | Spigot: Middlings: Per Cent. Total Solids on Screen Size | Spigot: Middlings: Per Cent. Total Solids in Spigot | Spigot: Middlings: Per Cent. Total Solids in Feed | Spigot: Tailings: Per Cent. Total Solids on Screen Size | Spigot: Tailings: Per Cent. Total Solids in Spigot | Spigot: Tailings: Per Cent. Total Solids in Feed | Overflow: Per Cent. Total Solids in Overflow | Overflow: Per Cent. Total Solids in Feed | Overflow: Free Mineral: Per Cent. Total Solids on Screen Size | Overflow: Free Mineral: Per Cent. Total Mineral in Overflow | Overflow: Free Mineral: Per Cent. Total Mineral in Feed | Overflow: Tailings: Per Cent. Total Solids on Screen Size | Overflow: Tailings: Per Cent. Total Solids in Overflow | Overflow: Tailings: Per Cent. Total Solids in Feed |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Test No. 7** | | | | | | | | | | | | | | | | | | | |
| 2.38 | 4.4 | 3.0 | ..... | ..... | ..... | 100.0 | 4.4 | 3.0 | ..... | ..... | ..... | ..... | | | | | | | |
| 1.41 | 36.3 | 25.1 | 4.2 | 14.7 | 7.4 | 50.8 | 18.5 | 12.8 | 45.0 | 16.3 | 11.3 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 0.841 | 36.2 | 25.0 | 6.8 | 23.8 | 11.9 | ..... | ..... | ..... | 93.2 | 33.6 | 23.4 | 0.8 | 0.2 | ..... | ..... | ..... | 100.0 | 0.8 | 0.2 |
| 0.500 | 16.5 | 11.4 | 13.6 | 21.6 | 10.8 | ..... | ..... | ..... | 86.4 | 14.2 | 9.9 | 3.8 | 1.2 | 7.2 | 1.1 | 0.6 | 92.8 | 3.5 | 1.1 |
| 0.350 | 3.0 | 2.1 | 45.0 | 13.1 | 6.5 | ..... | ..... | ..... | 55.0 | 1.7 | 1.1 | 2.8 | 0.8 | 9.5 | 1.1 | 0.6 | 90.5 | 2.5 | 0.8 |
| 0.166 | 2.8 | 2.0 | 70.2 | 18.8 | 9.3 | ..... | ..... | ..... | 29.8 | 0.9 | 0.5 | 25.5 | 7.9 | 8.8 | 9.7 | 4.8 | 91.2 | 23.3 | 7.2 |
| 0.07 | 0.6 | 0.4 | 100.0 | 8.0 | 4.0 | ..... | ..... | ..... | ..... | ..... | ..... | 15.5 | 4.9 | 17.3 | 11.3 | 5.7 | 82.7 | 12.8 | 3.9 |
| Th. 0.07 | 0.2 | 0.1 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | 51.6 | 15.9 | 35.0 | 76.8 | 38.4 | 65.0 | 33.5 | 10.3 |
| Total | 100.0 | 69.1 | 10.4 | 100.0 | 49.9 | 22.9 | 22.9 | 15.8 | 66.7 | 66.7 | 46.2 | 100.0 | 30.9 | 23.6 | 100.0 | 50.1 | 76.4 | 76.4 | 23.5 |
| **Test No. 8** | | | | | | | | | | | | | | | | | | | |
| 2.38 | 12.8 | 9.2 | ..... | ..... | ..... | 100.0 | 12.8 | 9.2 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 1.41 | 37.8 | 27.1 | 1.7 | 7.6 | 3.3 | 98.3 | 37.0 | 26.5 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 0.841 | 29.8 | 21.3 | 5.3 | 18.9 | 8.2 | ..... | ..... | ..... | 94.7 | 28.3 | 20.2 | 0.7 | 0.2 | ..... | ..... | ..... | 100.0 | 0.7 | 0.2 |
| 0.500 | 13.1 | 9.4 | 15.9 | 24.9 | 10.8 | ..... | ..... | ..... | 84.1 | 11.0 | 7.9 | 6.0 | 1.7 | 7.9 | 1.7 | 1.0 | 92.1 | 5.6 | 1.6 |
| 0.350 | 2.8 | 2.0 | 44.4 | 15.2 | 6.5 | ..... | ..... | ..... | 55.6 | 1.6 | 1.2 | 6.9 | 1.9 | 7.0 | 1.7 | 1.0 | 93.0 | 6.4 | 1.8 |
| 0.166 | 2.8 | 2.0 | 72.7 | 24.2 | 10.4 | ..... | ..... | ..... | 27.3 | 0.8 | 0.5 | 21.6 | 6.1 | 14.0 | 10.9 | 6.2 | 86.0 | 18.5 | 5.3 |
| 0.07 | 0.9 | 0.6 | 80.0 | 9.2 | 3.9 | ..... | ..... | ..... | 20.0 | 0.2 | 0.2 | 17.5 | 5.0 | 24.6 | 15.5 | 8.8 | 75.4 | 13.2 | 3.7 |
| Th. 0.07 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | 47.3 | 13.5 | 41.0 | 70.2 | 39.9 | 59.0 | 27.8 | 7.9 |
| Total | 100.0 | 71.6 | 8.3 | 100.0 | 43.1 | 49.8 | 49.8 | 35.7 | 41.9 | 41.9 | 30.0 | 100.0 | 28.4 | 27.8 | 100.0 | 56.9 | 72.2 | 72.2 | 20.5 |
| **Test No. 9** | | | | | | | | | | | | | | | | | | | |
| 2.38 | 0.7 | 0.4 | ..... | ..... | ..... | 100.0 | 0.7 | 0.4 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 1.41 | 33.6 | 21.4 | 2.7 | 11.1 | 4.3 | 97.3 | 32.7 | 20.8 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 0.841 | 39.4 | 25.1 | 3.6 | 17.5 | 6.9 | ..... | ..... | ..... | 96.4 | 38.0 | 24.2 | 1.2 | 0.4 | 10.0 | 0.5 | 0.3 | 90.0 | 1.0 | 0.4 |
| 0.500 | 18.7 | 11.9 | 12.0 | 27.8 | 11.0 | ..... | ..... | ..... | 88.0 | 16.4 | 10.5 | 8.2 | 3.0 | 6.8 | 2.6 | 1.6 | 93.2 | 7.7 | 2.8 |
| 0.350 | 3.2 | 2.0 | 34.6 | 13.4 | 5.3 | ..... | ..... | ..... | 65.4 | 2.1 | 1.3 | 8.2 | 3.0 | 6.8 | 2.6 | 1.6 | 93.2 | 7.7 | 2.8 |
| 0.166 | 3.6 | 2.3 | 58.9 | 26.2 | 10.3 | ..... | ..... | ..... | 41.1 | 1.5 | 0.9 | 23.1 | 8.4 | 11.7 | 12.4 | 7.5 | 88.3 | 20.4 | 7.4 |
| 0.07 | 0.8 | 0.5 | 41.7 | 4.0 | 1.6 | ..... | ..... | ..... | 58.3 | 0.5 | 0.3 | 15.6 | 5.7 | 20.1 | 14.5 | 8.8 | 79.9 | 12.5 | 4.6 |
| Th. 0.07 | | | | | | | | | | | | 43.7 | 15.9 | 45.0 | 67.4 | 40.8 | 55.0 | 29.0 | 10.5 |

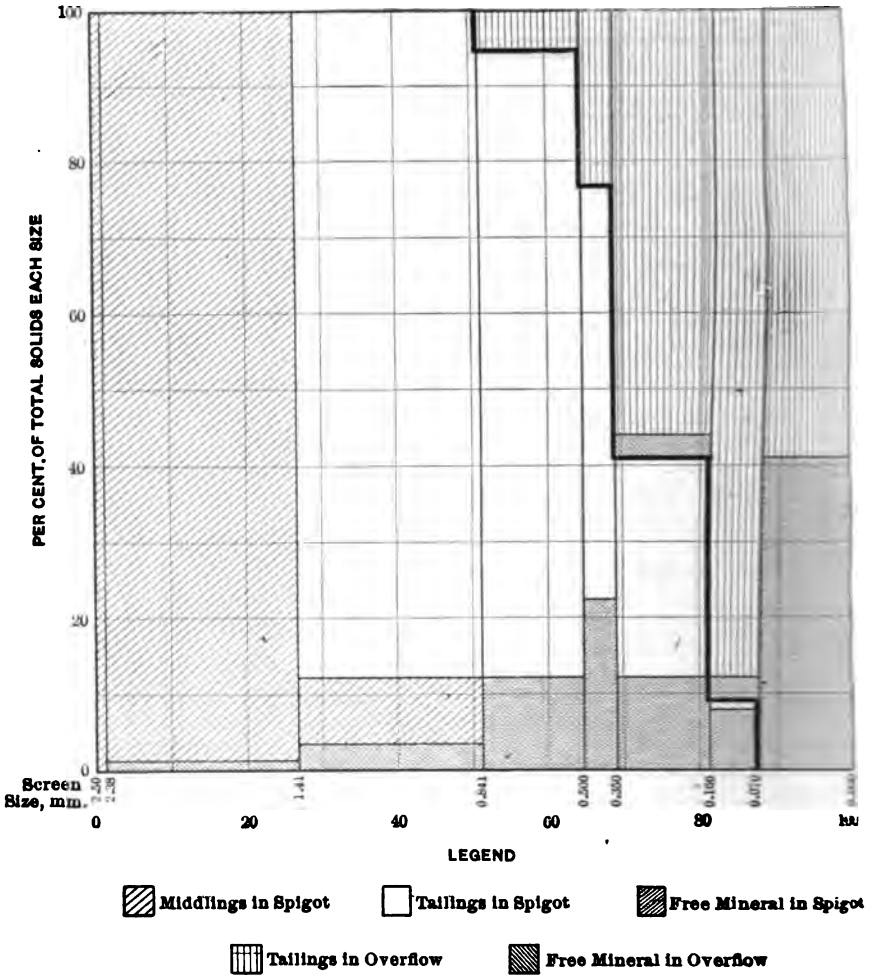
## *No. 2 Classifier*

| Screen Size, Millimeters | Per Cent. Total Solids in Spigot | Per Cent. Total Solids in Feed | Spigot: Free Mineral: Per Cent. Total Solids on Screen Size | Spigot: Free Mineral: Per Cent. Total Free Mineral: Spigot | Spigot: Free Mineral: Per Cent. Total Free Mineral: Feed | Spigot: Middlings: Per Cent. Total Solids on Screen Size | Spigot: Middlings: Per Cent. Total Solids in Spigot | Spigot: Middlings: Per Cent. Total Solids in Feed | Spigot: Tailings: Per Cent. Total Solids on Screen Size | Spigot: Tailings: Per Cent. Total Solids in Spigot | Spigot: Tailings: Per Cent. Total Solids in Feed | Overflow: Per Cent. Total Solids in Overflow | Overflow: Per Cent. Total Solids in Feed | Overflow: Free Mineral: Per Cent. Total Solids on Screen Size | Overflow: Free Mineral: Per Cent. Total Mineral in Overflow | Overflow: Free Mineral: Per Cent. Total Mineral in Feed | Overflow: Tailings: Per Cent. Total Solids on Screen Size | Overflow: Tailings: Per Cent. Total Solids in Overflow | Overflow: Tailings: Per Cent. Total Solids in Feed |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Test No. 10** | | | | | | | | | | | | | | | | | | | |
| 2.380 | 0.9 | 0.6 | ..... | ..... | ..... | 100.0 | 0.9 | 0.6 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 1.41 | 37.1 | 24.3 | 3.0 | 13.6 | 5.9 | 97.0 | 36.0 | 23.6 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 0.841 | 38.5 | 25.3 | 5.4 | 25.2 | 10.9 | ..... | ..... | ..... | 94.6 | 36.4 | 23.9 | 1.2 | 0.4 | 5.7 | 0.4 | 0.2 | 93.3 | 1.2 | 0.4 |
| 0.500 | 17.1 | 11.2 | 12.4 | 25.6 | 11.1 | ..... | ..... | ..... | 87.6 | 15.0 | 9.9 | 8.2 | 2.8 | 7.1 | 2.8 | 1.6 | 92.9 | 7.6 | 2.6 |
| 0.350 | 2.6 | 1.7 | 35.0 | 11.0 | 4.7 | ..... | ..... | ..... | 65.0 | 1.7 | 1.1 | 7.7 | 2.6 | 6.5 | 2.4 | 1.4 | 93.5 | 7.2 | 2.5 |
| 0.166 | 3.0 | 2.0 | 58.8 | 20.9 | 9.1 | ..... | ..... | ..... | 41.2 | 1.2 | 0.8 | 19.8 | 6.8 | 11.6 | 11.2 | 6.3 | 88.4 | 17.5 | 6.0 |
| 0.07 | 0.8 | 0.5 | 36.8 | 3.7 | 1.6 | ..... | ..... | ..... | 63.2 | 0.5 | 0.3 | 16.0 | 5.5 | 9.3 | 7.2 | 4.1 | 90.7 | 14.5 | 5.0 |
| Th. 0.07 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | 47.1 | 16.3 | 34.0 | 76.0 | 43.1 | 66.0 | 31.4 | 10.8 |
| Total | 100.0 | 65.6 | 8.3 | 109.0 | 43.3 | 36.9 | 36.9 | 24.2 | 54.8 | 54.8 | 35.0 | 100.0 | 34.4 | 20.6 | 100.0 | 56.7 | 79.4 | 79.4 | 27.3 |
| **Test No. 11** | | | | | | | | | | | | | | | | | | | |
| 2.380 | 1.3 | 0.8 | ..... | ..... | ..... | 100.0 | 1.3 | 0.8 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 1.410 | 38.0 | 24.8 | 3.5 | 12.8 | 5.6 | 96.5 | 36.5 | 24.0 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... |
| 0.841 | 37.7 | 24.6 | 5.5 | 19.8 | 8.6 | ..... | ..... | ..... | 94.5 | 35.6 | 23.2 | 1.2 | 0.4 | 18.2 | 0.9 | 0.5 | 81.8 | 1.0 | 0.4 |
| 0.500 | 16.4 | 10.7 | 17.7 | 27.9 | 12.2 | ..... | ..... | ..... | 82.3 | 13.6 | 8.8 | 8.0 | 2.8 | 4.3 | 1.4 | 0.8 | 95.7 | 7.7 | 2.7 |
| 0.350 | 3.0 | 2.0 | 50.0 | 14.5 | 6.4 | ..... | ..... | ..... | 50.0 | 1.6 | 1.0 | 8.4 | 2.9 | 15.1 | 5.0 | 2.8 | 84.9 | 7.0 | 2.5 |
| 0.166 | 2.9 | 1.9 | 76.5 | 20.9 | 9.2 | ..... | ..... | ..... | 24.5 | 0.7 | 0.4 | 23.3 | 8.1 | 25.0 | 23.1 | 13.0 | 75.0 | 17.5 | 6.0 |
| 0.070 | 0.7 | 0.5 | 63.6 | 4.1 | 1.7 | ..... | ..... | ..... | 36.4 | 0.3 | 0.3 | 15.3 | 5.3 | 7.5 | 4.5 | 2.6 | 92.5 | 14.2 | 4.0 |
| Th. 0.070 | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | ..... | 43.8 | 15.2 | 37.5 | 65.1 | 36.6 | 62.5 | 27.3 | 9.5 |
| Total | 100.0 | 65.3 | 10.4 | 100.0 | 43.7 | 37.8 | 37.8 | 24.8 | 51.8 | 51.8 | 33.7 | 100.0 | 34.7 | 25.3 | 100 0 | 56.3 | 74.7 | 74.7 | 26.0 |



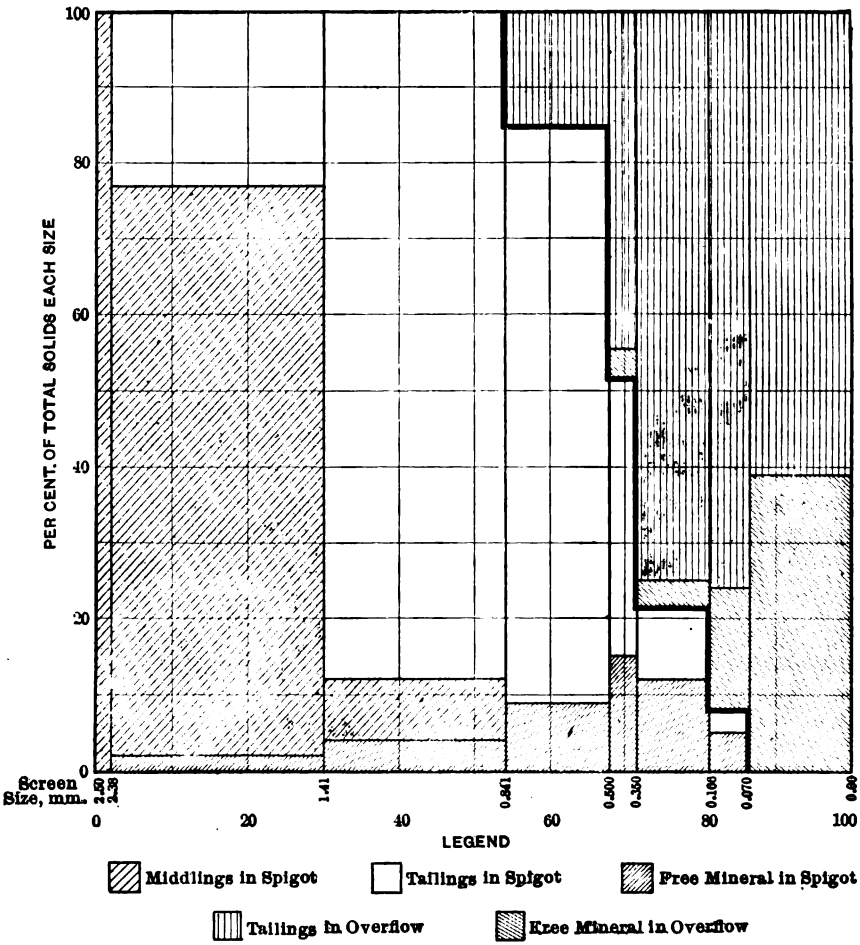

The entire area represents feed to classifier. Free mineral consists of 15 per cent. insoluble concentrates. Heavy black line separates plug and overflow. Through 0.07 free mineral consists of 60 per cent. insoluble concentrates.

CLASSIFICATION CHART NO. 2 CLASSIFIER. TEST NO. 1.



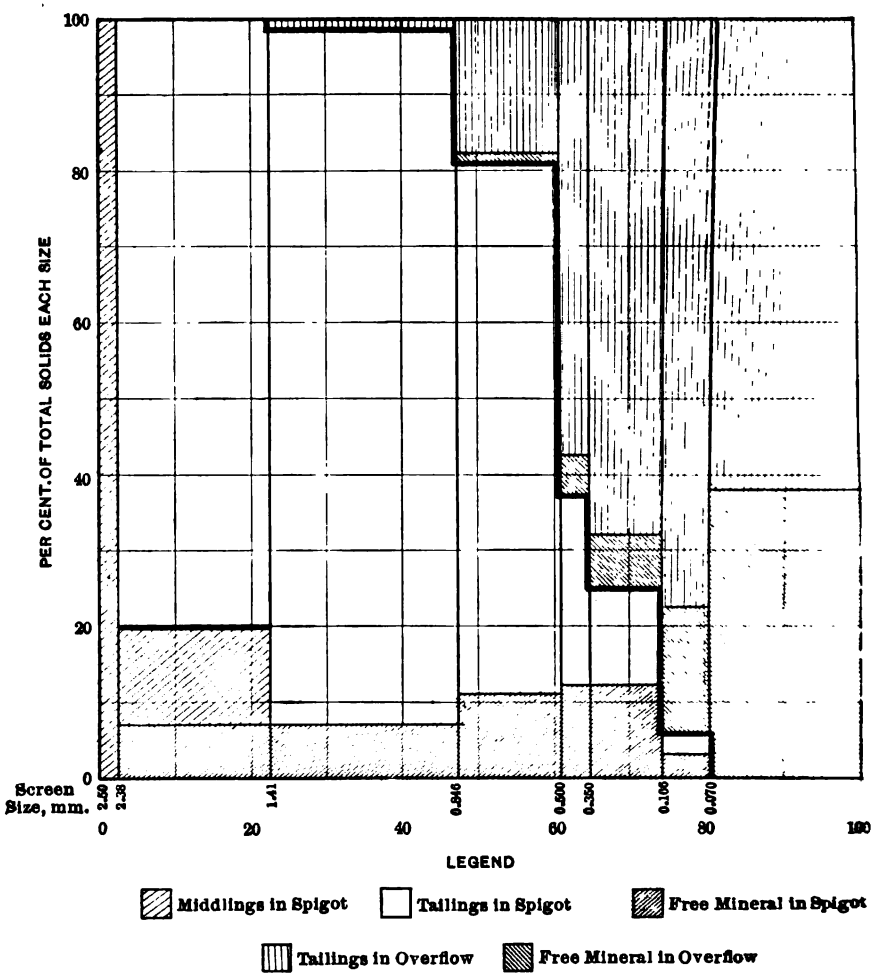

The entire area represents the total feed to classifier. Middlings on 2.38 all true middlings. Middlings on 1.41 consist of free mineral and tailings. All free mineral on 0.07 mm. consists of 15 per cent. insoluble concentrates. All free mineral through 0.07 mm. consists of 60 per cent. insoluble.

CLASSIFICATION CHART NO. 2 CLASSIFIER. TEST NO. 2.



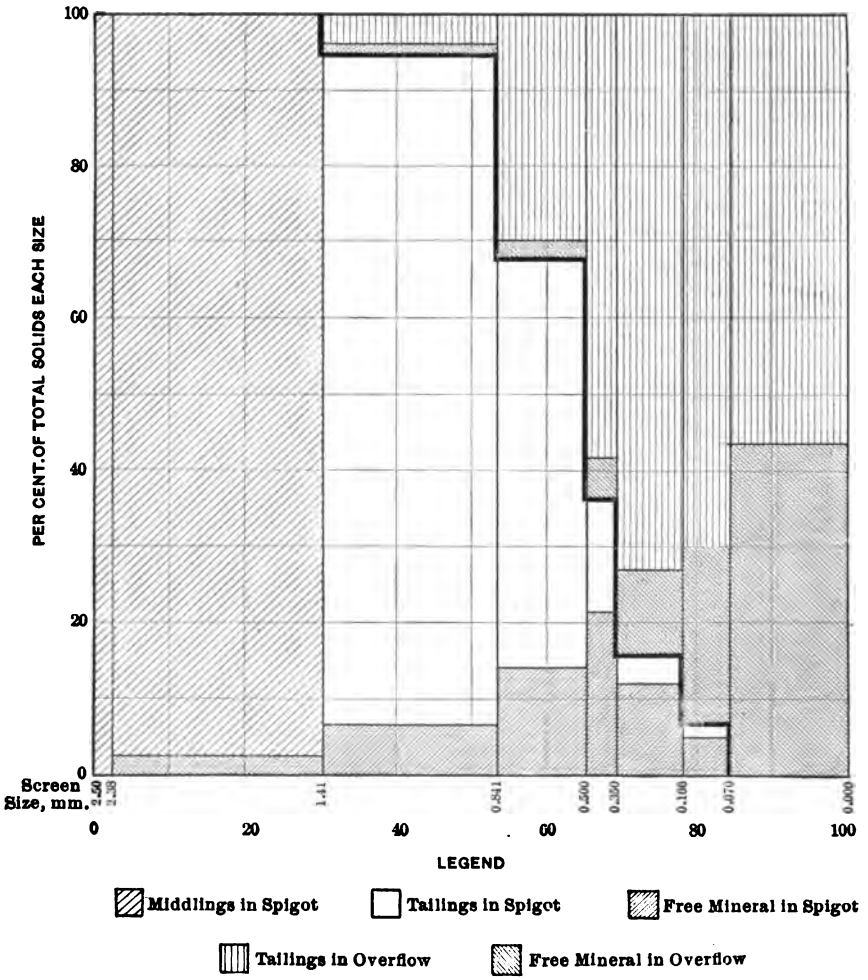

The entire area represents the total feed to classifier. Middlings on 2.38 mm. all true middlings. Middlings on 1.41 and 0.841 contained considerable free mineral. Free mineral = 15 per cent. insoluble for all sizes on 0.07 mm. Free mineral through 0.07 mm. contains 60 per cent. insoluble.

CLASSIFICATION CHART NO. 2 CLASSIFIER. TEST NO. 3.



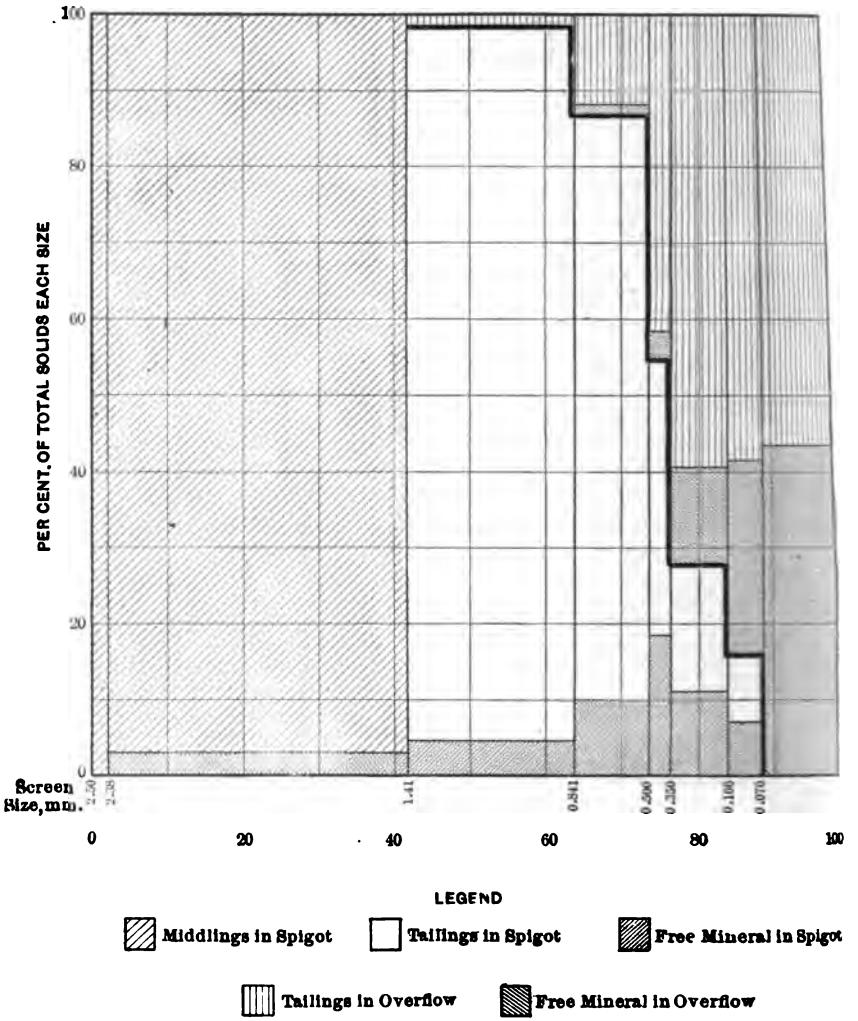

The entire area represents the total feed to classifier. All middlings true middlings Free mineral in material through 0.07 calculated to 60 per cent. insoluble. All other free mineral calculated to 15 per cent. insoluble.

CLASSIFICATION CHART NO. 2 CLASSIFIER. TEST NO. 4.



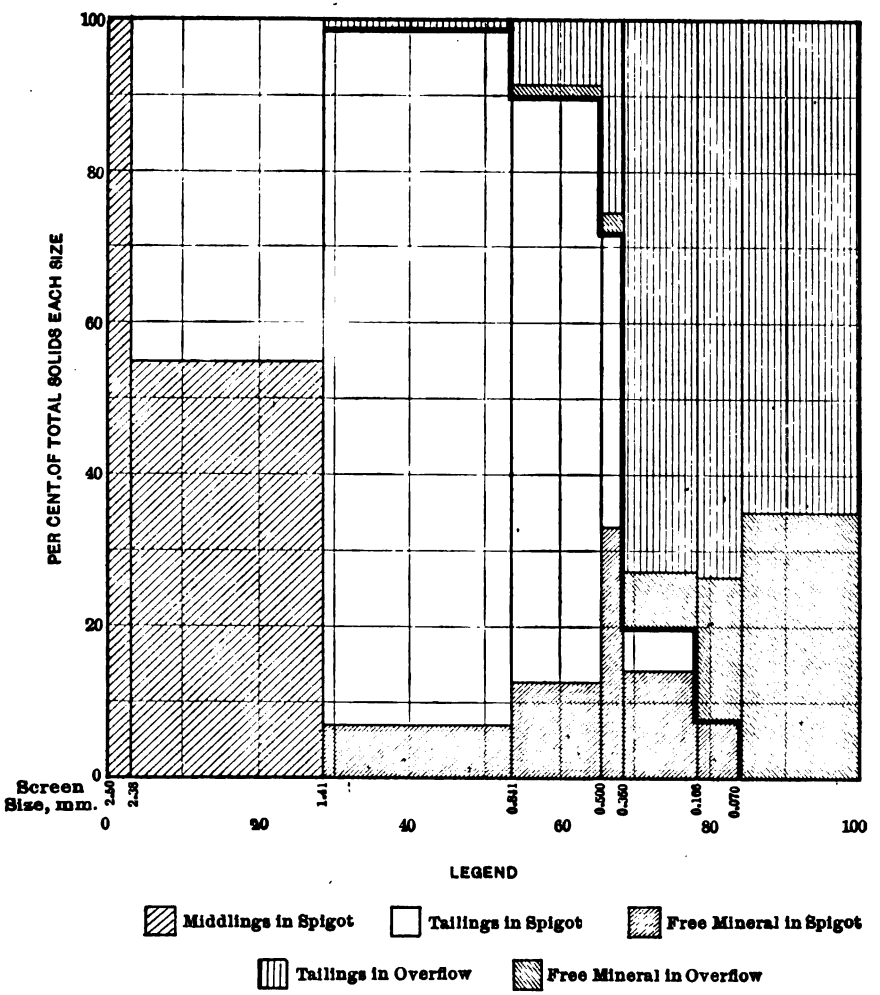

The entire area represents the total feed to classifier. Middlings contain some tailings material. Free mineral in material through 0.07 calculated to 60 per cent. insoluble. All other free mineral calculated to 15 per cent. insoluble.

CLASSIFICATION CHART NO. 2 CLASSIFIER. TEST NO. 5.



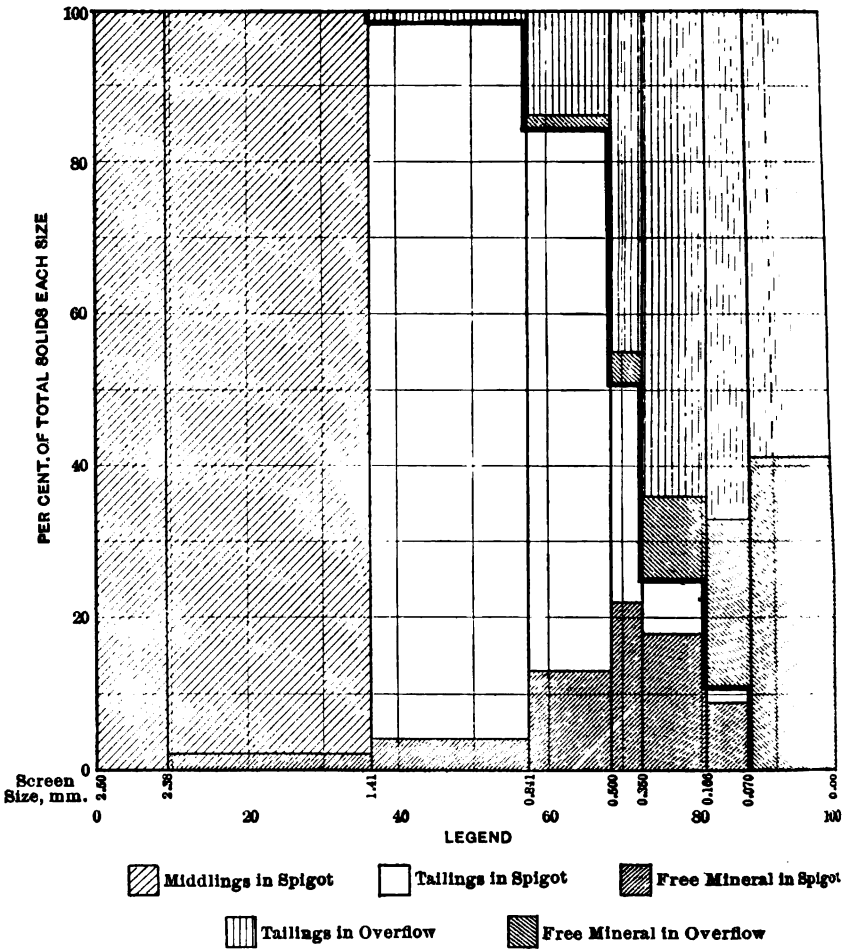

The entire area represents the total feed to classifier. Middlings considerable tailings. Free mineral in material through 0.07 calculated to 60 per cent. insoluble. All other free mineral calculated to 15 per cent. insoluble.

CLASSIFICATION CHART NO. 2 CLASSIFIER. TEST NO. 6.



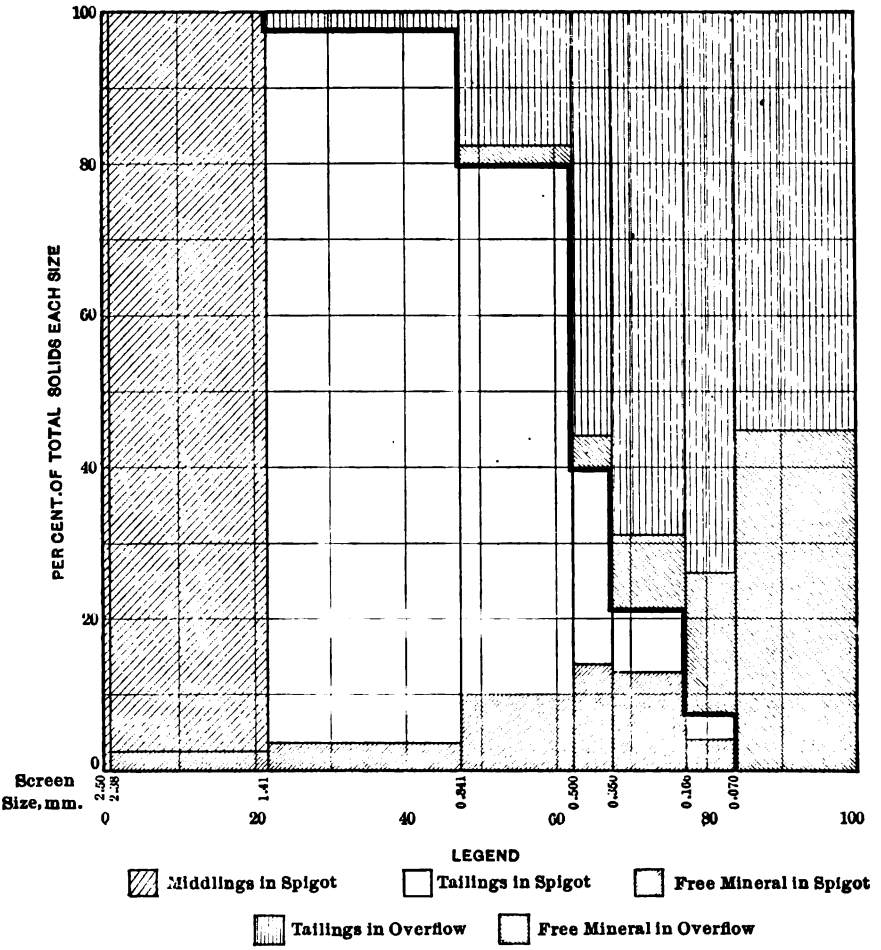

The entire area represents the total feed to classifier. All middlings true middlings. Free mineral in material through 0.07 calculated to 60 per cent. imsoluble. All other free mineral calculated to 15 per cent. insoluble.

CLASSIFICATION CHART NO. 2 CLASSIFIER. TEST NO. 7.



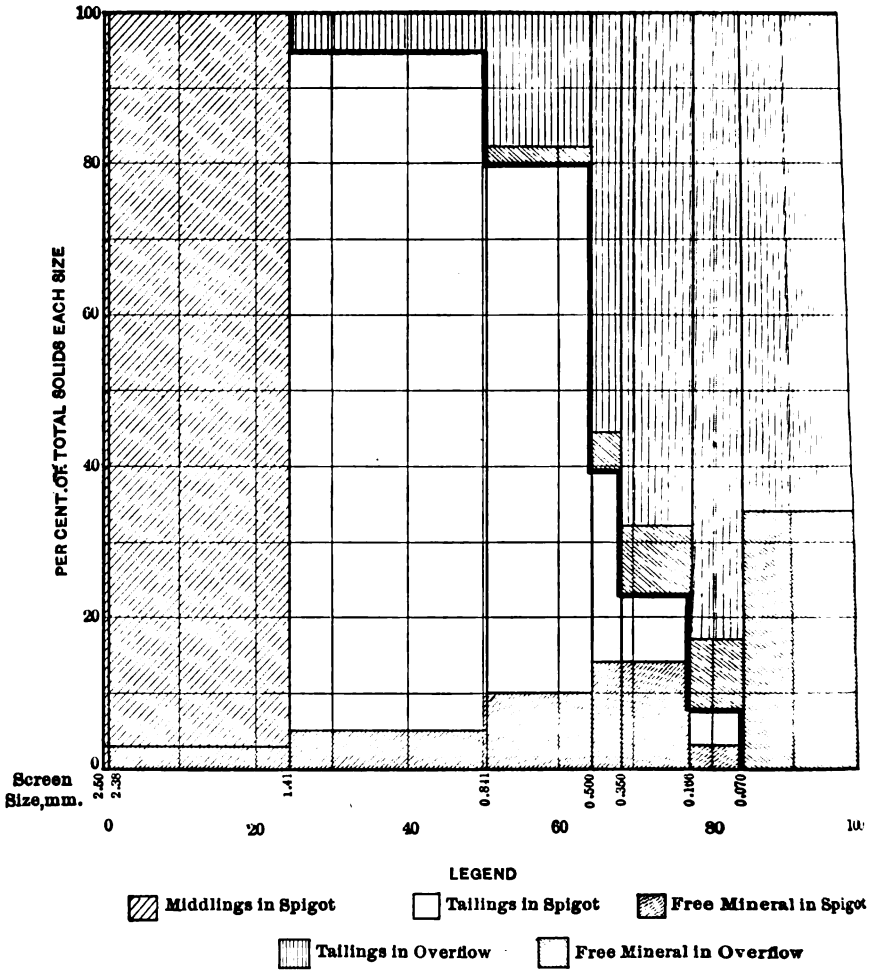

The entire area represents the total feed to classifier. Middlings contained some tailings. Free mineral in material through 0.07 calculated to 60 per cent. insoluble. All other free mineral calculated to 15 per cent. insoluble.

CLASSIFICATION CHART NO. 2 CLASSIFIER. TEST NO. 8.



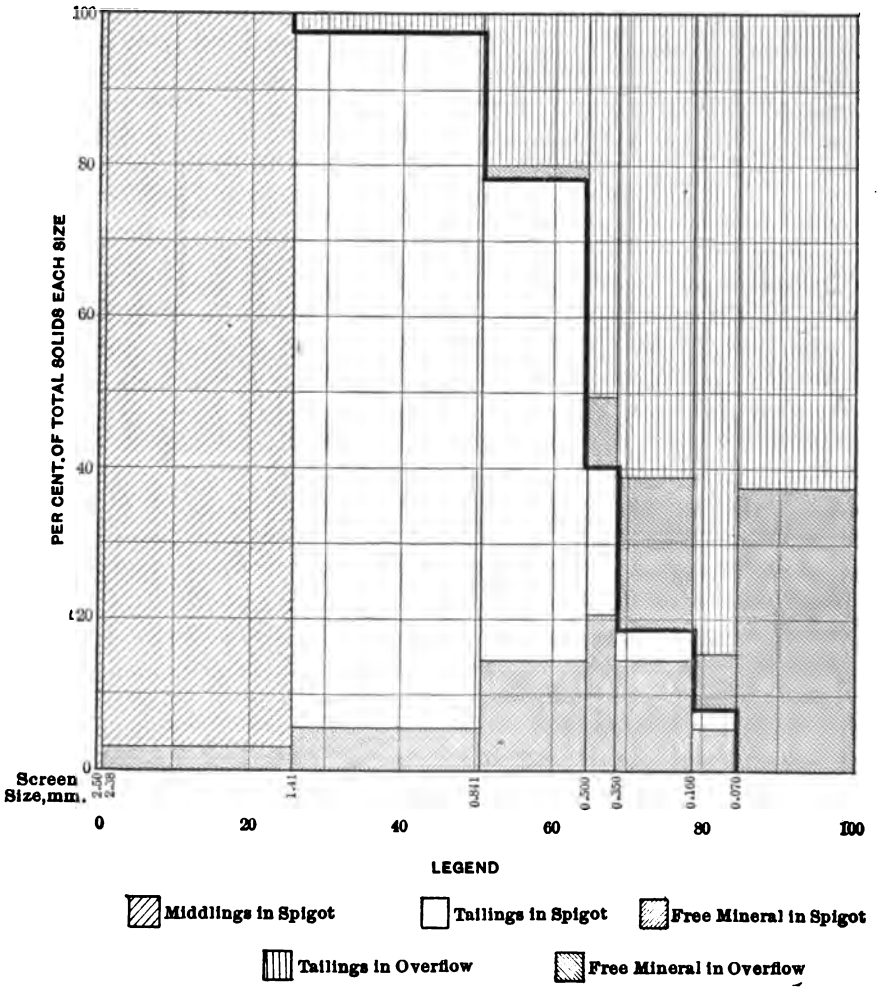

The entire area represents the total feed to classifier. All middlings true middlings. Free mineral in material through 0.07 calculated to 60 per cent. insoluble. All other free mineral calculated to 15 per cent. insoluble.

CLASSIFICATION CHART No. 2 CLASSIFIER. TEST No. 9.




The entire area represents the total feed to classifier. All middlings true middlings. Free mineral in material through 0.07 calculated to 60 per cent. insoluble. All other free mineral calculated to 15 per cent. insoluble.

CLASSIFICATION CHART NO. 2 CLASSIFIER. TEST NO. 10.




The entire area represents the total feed to classifier. Middlings all true middlings. Free mineral in material through 0.07 calculated to 60 per cent. insoluble. All other free mineral calculated to 15 per cent insoluble.

CLASSIFICATION CHART NO. 2 CLASSIFIER. TEST NO. 11.

Particular attention is called to the consistency of classification, and especially the low water consumption.

Richards's velocity[1] for full teeter of 1.37 mm. grains is 53 mm. per second. That obtained in the classifier was 39, but this calculation was made assuming the same density in the teeter chamber as in the sorting column. As a matter of fact, had this density been determined, the rising velocity in the teeter chamber would have checked Richards's velocity. The low consumption of water is due to a great extent to the inner cone.

### *9. The Inner Cone.*

The inner cone, in the writer's opinion, is a great improvement over the type of classifier without the cone. The chief function of the cone is to produce a constant rising current under all conditions of operation. It also acts as a feed box and reduces liability of choke-ups. By utilizing every drop of feed water for the production of the rising current, the amount of rising fresh water is not much greater than that required to produce the quicksand or full teeter condition in the teeter chamber. This fact not only reduces considerably the total fresh water consumed, but produces a much better average classification than a classifier without the cone. In a classifier with no inner cone, if the solids in the feed pulp decrease the teeter chamber may disappear entirely and we have simply a free-settling classifier with considerable fine material being discharged through the spigot. But if in an inner cone classifier the density decreases, *i. e.*, the volume remains constant but the solids decrease, we still have the same rising current, which has a tendency to lift out all of the fine material and a good part of the coarser feed. The inner cone has given best results when so designed that the velocity at the top of the annular space is the same as the free-settling velocity of the maximum quartz grain we wish to overflow and the velocity at the bottom of the annular space is approximately 25 per cent. greater. Some particles are drawn up into the space which eventually settle down and are discharged through the spigot.

As seen in Fig. 10, the overflow is constantly under a head of from 5 to 6 in. in the inner cone. Any tendency to bank, results in the increase of this head and a cleaning up of the banking material. However, banking or choking is rare with the inner cone type.

The advantages of the inner cone may be summed up as follows:

1. A uniform close classification is obtained.
2. The water consumption is decreased.
3. The liability of choke-ups is decreased.

---

[1] *Ore Dressing*, vol. iii, p. 1462 (1909).

4. Boiling in the overflow is prevented, thereby preventing coarse material being thrown over.

The chief disadvantage is the requirement of 8 in. of extra head room.

In any classifier or classifier system, the most important demands to be satisfied are: (1), flexibility of each unit and of the classifier system as a whole; (2), maximum tonnage; and (3), minimum water consumption. Let us see how the Anaconda classifier answers these fundamental requirements.

(1). The Anaconda classifier system is remarkably adapted to meet any given set of mill conditions. The classifier units can be operated under the direct system or under the indirect system. In the direct system the slime is the product of the last classification. In the indirect system the slime is the first product removed. The direct system is best adapted to conditions where mill head room is exceedingly valuable. The total pulp, usually a screen undersize, is fed directly into a table-feed classifier which overflows all material from a given size, usually 0.75 mm., down to 0.0 mm.

The overflow is easily laundered or pumped to some point where it can be further classified into slime and one or more classified table feeds. The valuable point here is that the second and third classifiers in this series can be placed immediately preceding the particular concentrating machine best adapted for the treatment of each product, allowing the practice of running very dense spigots as feed to the concentrators, which are usually some type of shaking table. Reference to Fig. 8 will show the manner of separation of such a feed into concentrates, middlings, and tailings by a Wilfley table. The concentrates are very clean, usually containing from 9 to 15 per cent. of gangue; the middlings are very small in tonnage, while the tailings are very dense and assay usually from 0.30 to 0.40 per cent. of Cu.

The indirect system first removes the slime (98 per cent. through 200 mesh), which can be segregated by itself in a plant equipped for slime treatment. The density of the slime produced in the indirect system is from 160 to 200 per cent. as dense as in the direct system. This is due to the fact that no hydraulic rising water is required for the separation of the slime alone. The spigot of the deslimer is fed directly into a table feed classifier, which affords to the classifier an absolutely constant volume of feed pulp. This fact is the most important feature of the indirect system. A variable volume of feed pulp to a deslimer is immaterial provided the deslimer is designed to handle the maximum volume. A variable feed volume to a table-feed classifier is serious in that the rising velocity varies proportionately with the feed volume.

As to the flexibility of each unit, the tabulation following Fig. 10 shows

the results of a variable tonnage. This particular unit, which is one of the earliest inner cone types, received as its feed the total undersize of two 2.5 mm. round-hole trommels. This feed was far from constant, varying from 90 to 255 tons per 24 hr., and yet the classifier maintained practically the same degree of classification under all conditions. A variation of 40 per cent. of its rated tonnage, either way, is not serious in an Anaconda classifier. The hydraulic water is the only adjustment necessary, and this is only a slight one.

The effect on the degree of classification due to variations of the percentage of fine material (mine fines) in the feed is minimized in the Anaconda classifier. In any mill which is not equipped with a central crushing plant the variations in the percentage of mine fines in the original ore and even the percentages of fines due to crushing are considerable. To perform consistent work a classifier must be able to absorb these fluctuations in the quality of its feed. On pp. 2168 and 2169 are shown two complete sets of figures illustrating to a certain extent how well this function is performed by the classifier.

A comparison of these two sets of figures shows the same quality of work on the rich fine feed as upon the coarser and more impoverished feed. Both sets of figures were obtained under actual operating conditions.

(2). The maximum tonnage of the classifier is limited only by practice. In Montana it is the custom to design each unit to treat from 150 to 350 tons per 24 hr., depending upon conditions of practice. Table-feed classifiers seldom treat less than 250 tons, while deslimers working on very fine feed, 2.0 mm. or under, will average from 180 to 250 tons. Deslimers treating material whose limiting size is larger than 2.0 mm. have a capacity of from 250 to 400 tons per 24 hr. The large tonnage handled by a single unit is one reason for its great flexibility.

(3). The consumption of water by these classifiers is remarkably low. In all desliming Anaconda classifiers practically no rising hydraulic water is used, leaving only that which is added in the spigot to be charged against the classifier. However, the addition of water in the spigot reduces the consumption of water in the subsequent classifier. The deslimed overflow of the table-feed classifier is dewatered, and the water thus removed is used as hydraulic water in other classifier units of the system or as dressing water for concentrating machines. Thus the total water consumption for a system of classifiers is less than is shown by the sum of the individual units. The average water consumption in practice is as follows:

Desliming classifiers:—600 gal. per ton of feed.

Table-feed classifiers:—450 to 900 gal. per ton of feed.

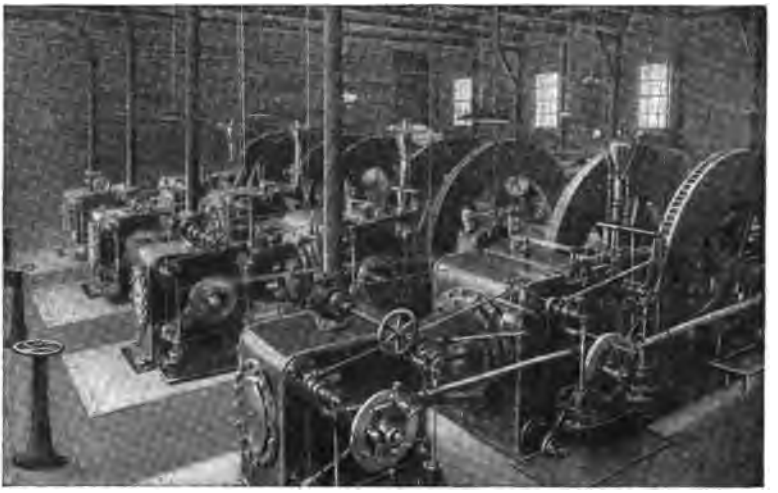
The relatively large consumption of water by the deslimer is due to

operating with a spigot less dense than is used in table-feed classifiers.

Fulfilling as it does the requirements of flexibility, efficiency of classification, high tonnage, and minimum water consumption, together with simplicity of construction, the Anaconda classifier is a great stride in the progress on modern ore concentration.

The writer wishes to thank A. E. Wiggin for his assistance in obtaining data from the Washoe Concentrator, and A. E. Wheeler and C. W. Goodale for reading and criticizing the paper.

# Sullivan "WC" Air Compressors



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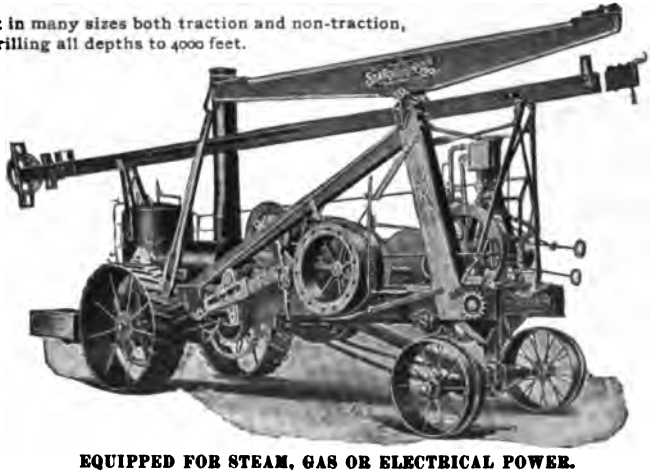

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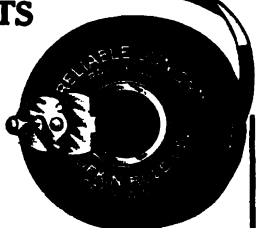


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**PROPOSALS FOR MEMBERSHIP.** A blank proposal for membership is included in each Year Book and Bulletin. In the event of not being able to secure such a one, applications for membership can be made out on any blank piece of paper, provided they include the class to which the candidate is proposed, whether member, associate or junior member; the signature of three members or associates (junior members must also have the endorsement of two of their instructors); a brief history of the candidate, including date and place of birth, his education and technical record, and finally a signed statement by the applicant that he desires to become a member.

AMERICAN INSTITUTE OF MINING ENGINEERS
29 West 39th Street, New York, N. Y.

# PROPOSAL FOR MEMBERSHIP

*Mr.* _____
(Name in Full)

*Occupation* _____

*Address* _____

*is hereby proposed by the undersigned, as a* _____

_____

*of the American Institute of Mining Engineers.*

_____
_____
_____
} Signatures of three Members or Associates.

*Place of birth* _____ *Year of birth* _____

*Education, general and technical, when, where and how acquired, with degrees, if any.*

| *Dates* | |
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*Record of experience. Briefly, the past and present employment, with names of employers, companies and associates. (Proper names, names of companies, etc., should be written without abbreviations.)*

| *Dates* | |
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*Present position*_____

_____

_____ *Signature*

*Dated*_____ *191*

## EXTRACTS FROM THE CONSTITUTION.

### ARTICLE II.—MEMBERS.

SEC. 1. The membership of the Institute shall comprise four classes, namely: 1. Members; 2. Honorary Members; 3. Associates; 4. Junior Members. * * *

SEC. 2. The following classes of persons shall be eligible for membership in the Institute, namely: Members, all professional mining engineers, geologists, metallurgists, or chemists, and all persons actively engaged in mining and metallurgical engineering, geology, or chemistry; as Associates, persons desirous of being connected with the Institute who in the opinion of the Board of Directors are suitable.

As Junior Members, all students in good standing in engineering schools who have not taken their degrees and who are nominated by at least two of their instructors. * * *

Every candidate for election as a Member, Associate, or Junior Member must be proposed for election by at least three Members or Associates, must be approved by the Committee on Membership, as prescribed in the By-Laws, and must be elected by the Board of Directors.

# HOW G-E MOTORS STOOD RECENT FLOODS

The following letter is one of many received from custom using G-E motors. These letters are uniformly enthusia and reveal the reason why G-E motors are selected withstanding moisture.

The great ability to withstand heat and moisture was of the reasons why G-E motors were selected to ope the Panama Canal gates, pumps, etc.

GRAY COAL COMPANY

Miners and Shippers of

ILLINOIS FIRST VEIN COAL

Danville, Illinois

GENERAL ELECTRIC CO., Chicago, Ill. April 11, 19

Gentlemen :—

In reply to your letter regarding our motors which have been under water at var times, will say that on March 18th, 1912 there occurred an ice jam at the Illinois Tra System bridge, causing the water to back up in the Vermilion river and overflow the ba in our operations.

At this time we had one of your 35-horse motors attached to a centrifugal pump which became completely submerged, remaining so for 92 days before we were able to get it out of the mud which had been washed in.

After getting this motor out we covered it with a waterproof to keep it from the weather and it remained on the bank through the summer months. When in the early fall, we were in need of another motor we thought we would test this motor and to our surprise it was right and has run daily until this flood of the past week when it became submerged again on March 24th and remained under water until April 7th, when it was tested and found all right.

Last January our 24" tile became clogged from the heavy rains which submerged another 35-horse motor and this motor was under water some 10 days.

This motor being in a pump house we thought it best to build a fire to dry it out when we did for a number of days when connections were made and found this motor all right also.

We have in our operation two 35-horse and two 25-horse motors of your make and all these motors have been submerged in water excepting one of the 25-horsepower, yet it seemed to be in good condition when the electrician made his tests and are now doing duty same as before.

We certainly feel proud of your motors, as you possibly can guess, for they have saved us a number of dollars from the fact that the insulation was able to stand such a test as the elements have given it in our operations.

Yours very truly,

GRAY COAL CO., John L. Crist, Pres

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